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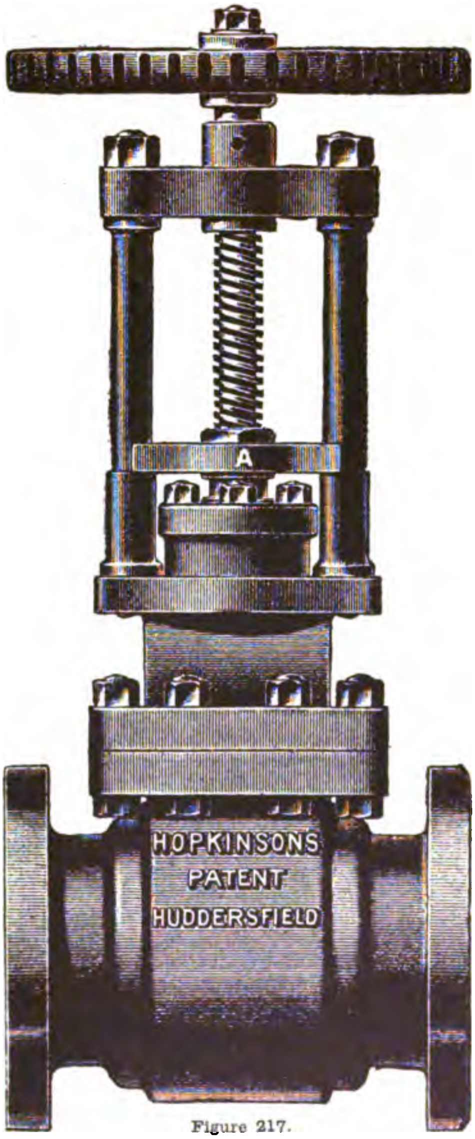

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PREFACE TO FIRST EDITION.

Even a slight acquaintance with the duties and responsibilities of a Colliery Manager will lead to the conclusion that he had need be almost omniscient within his own province. Besides his responsibility for satisfactory results in the opening-out and working of a colliery, under the ever-varying conditions of coal-mining enterprise, there rests upon him a heavy legal as well as moral responsibility which no true man would wish to shirk, and in the discharge of which he has to prepare for that which happens more often, perhaps, in his career than in that of most professional men—viz., *the unexpected.* It becomes him, therefore, to fit himself beforehand in every possible way for the discharge of his onerous duties. In so doing he will have to acquire the rudiments of Geology, Chemistry, and Electrical Engineering; a good deal more than the rudiments of Mechanical Engineering, Surveying, and Plan-making; and to make himself master of the mysteries comprised in the comprehensive terms Practical Mining and Ventilation. Further, he must be thoroughly versed in the obligations imposed upon him and his subordinates by the Acts of Parliament bearing on the subject of Coal Mining, and by the Special Rules in force in any given district.

The Author is well aware, after twenty years' experience, that the best and indeed the only satisfactory preparation for the efficient discharge of the duties of the Colliery Manager is that which is to be gained in the laborious school of experience; but he knows also that the wise use of a carefully prepared and comprehensive HANDBOOK—such as he ventures to believe the volume now in the reader's hand will be found to be—would have been to him an incalculable boon in the earlier years of his course, and hardly less so subsequently as a book of reference and of practical guidance. No doubt excellent handbooks already exist, dealing with various branches of the subject; but the Author's aim has been in the following pages to prepare a work more complete in itself than any of its predecessors, and one comprising the most recent information as to the ever-progressing science and art of Coal Mining.

In the use of the work by students, it is believed that the questions and answers given on the various subjects will be found specially helpful.

The Author gratefully acknowledges the assistance which has been given him in preparing this work for the press by his friends Mr. J. T. Robson, H. M.'s Inspector of Mines for South Wales, and the Rev. J. E. Flower, M.A., of Wandsworth; and also the courtesy shown by the Council of the Mining Institute of Scotland in granting him permission to quote the result of their Commission of Mining Engineers, as given in their *Transactions* (vol. iii., pp. 51 to 124). Besides much valuable information which has been obtained from the *Transactions* of the North of England and South Wales Institutes of Mining Engineers, he has also received considerable assistance from the columns of the *Practical Engineer* and the *Colliery Guardian*, and from other sources, reference to which will be found in various pages of the work.

The greatest care has been taken to ensure accuracy in every respect, but if any mistake has escaped notice the Author will be thankful to have it pointed out to him.

The preparation of this HANDBOOK—which for several years has occupied such time as the writer has been able to give to it—has been to him a most congenial task; and he ventures, in sending it forth, to express the hope that the work may be found as useful to the student, and to those engaged in the management of collieries, as the preparation of it has been pleasant and profitable to himself.

PONTYPRIDD,
January, 1891.

PREFACE TO FOURTH EDITION.

A fresh edition having been called for, the Author has seized the opportunity carefully to revise the whole work in order to bring it in every respect up to date.

It will be found that an important addition, involving about eighty pages, with thirty new illustrations, has now been made to Chapter XII., owing to the special attention of the Author having been recently directed to the subject of Mine Ventilation and the use of Regulators in airways.

This further enlargement of the volume, without any addition to the original price, will, it is believed, greatly enhance the value of the work as a whole, both to the practical Colliery Manager and to the student.

In spite of the utmost care in preparation, it can hardly be expected that a work involving so many details, and sharing the Author's attention with other engrossing duties, should be wholly free from errors. He will be grateful, therefore, to be informed of any mistakes which may have escaped notice, and he will also gladly receive and consider any suggestions with which he may be favoured with a view to still further increasing the usefulness of the work. He may be allowed to repeat here the thanks already expressed to those correspondents by whom such kindly services have been already rendered.

CHEPSTOW,
May, 1898.

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THE

COLLIERY MANAGER'S HAND-BOOK.

CHAPTER I.

GEOLOGY.

Stratified and Unstratified Rocks—Dip—Rise—Outcrop—Strike—Diversified arrangement of Rocks—Faults—Classification of the Rocks into Systems—Igneous Rocks—Description of the different systems of Stratified Rocks.

THE Science of Geology in its widest sense comprises all that is known concerning the constitution and history of our globe. It has been the life-study of many eminent men, and as one of the youngest of the sciences has made amazing progress; but our knowledge, gained by direct observation, is still limited to within a mile or two of the surface, and there is consequently room for much speculation as to the condition of the interior of the earth.

An examination of the rocks soon led to their being divided into *stratified* and *unstratified*. The former have the appearance of having been deposited in layers one above another, by aqueous action; whilst *unstratified* rocks occur as amorphous masses and seem to have been more or less completely fused. Granite is a specimen of the latter group of rocks. It is now generally held by geologists, physicists, and astronomers, that the earth must, at one time, have been entirely in a state of fusion, and that the crust in cooling became a hard unstratified mass. In process of time the waters began to destroy this crust, and to deposit layer after layer of the derived material. This process of disintegration by mechanical and chemical agencies, and deposition of the resulting detritus, with intermittent eruption of heated matter from the earth's interior, has gone on through countless ages until the earth's crust has obtained its present condition. The sedimentary rocks were not always deposited on horizontal surfaces, and the result of deposition on an inclined surface would be to form a bed at nearly the same angle as the surface on which it was deposited.

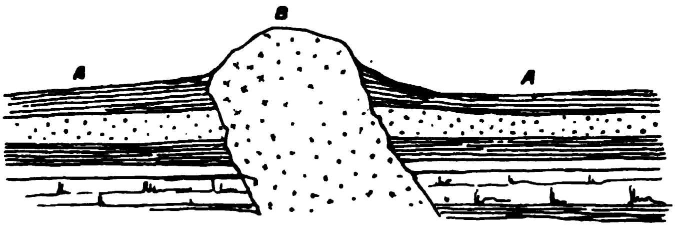


Fig. 1.—STRATIFIED AND UNSTRATIFIED ROCKS.
A A Stratified, and B Unstratified.

As beds were deposited over extensive areas there may have been many variations of surface causing the beds to be inclined at different angles and in different directions, but in most cases where the strata are inclined, they were originally deposited in a position more or less horizontal, and have been subsequently disturbed by earth-movements. Inclined strata are shown at B in Fig. 2.

In almost the earliest of the aqueous rocks signs of vegetable and animal life are found in the shape of *fossils*. As the organisms died, the waters covered them, depositing again other layers of sand, clay, &c. In course of time the waters receded, or, more probably, the land was elevated, and fresh forms of life appeared, to be themselves buried in their turn. The fossils represent either the creatures which inhabited the waters in which sedimentation was proceeding, or those which lived on the neighbouring land, and had their remains carried down and entombed in the sediment.

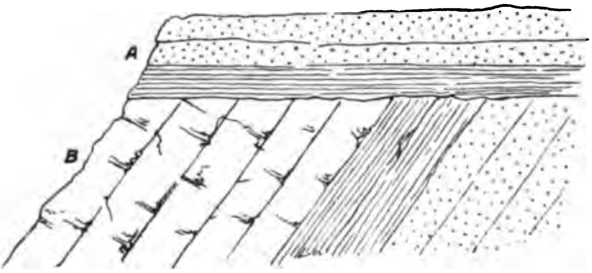


Fig. 2.—Horizontal and Inclined Strata, Showing Unconformity.

The great fact to be remembered in regard to these stratified rocks is, that they have been deposited in regular order, and that each has its characteristic fossils. Not that every formation of rock was deposited all over the world, but the order of their occurrence, if there, remains generally the same. Sometimes, indeed, a formation is altogether wanting, or the order of their position reversed, but these are exceptions and can usually be accounted for by denudation or by volcanic agency, or by other earth-movements.

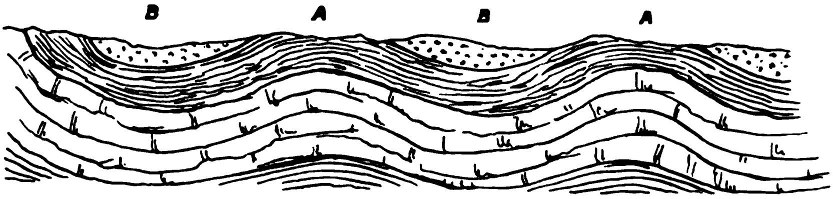


Fig. 3.—Curved Strata.
A, Anticlinal. B, Synclinal.

The angle at which a stratum inclines to the horizon is called its *dip* when viewed in the direction of the fall, and the *rise* when viewed in the contrary direction. When an inclined stratum comes to the surface its edge is called the *outcrop*, and the line of outcrop along a horizontal surface is termed its *strike*.

The dip is always at right angles to the strike, so that if the dip be given, the strike is also known; but if the strike only be given, the dip cannot be known from it, because the dip may incline to either side of the strike. When strata dip in opposite directions from a ridge or axis of elevation they form an *anticline* or *saddleback* as at A, Fig. 3, and when they dip towards a common line of depression as at B, they are said to be *synclinal*, and the depression so formed is spoken of as a *trough* or *basin*. These terms apply only to the folds of the strata and not to the features of the surface of the ground. When strata are bent and twisted they are termed *contorted*. When strata lie upon each other in parallel order, they are said to be *conformable;* but when one set reclines upon another at a different angle they are termed *unconformable*. Thus, in Fig. 2, the strata A are conformable to each other, and so are the strata B, but there is a strong *unconformity*, or discordance of stratification, between A and B. The strata are said to be *monoclinal* when, though inclined, they all slope the same way; and *periclinal* when dipping in every direction from a common centre. When strata terminate abruptly in a bold bluff edge, they are said to form an *escarpment*.

The dislocations, fractures, and fissures produced in the rocky crust by subterranean movements are known by such terms as *faults*, *slips*, *hitches*, *heaves*, *leaps*, *throws*, *troubles*, *&c.*

Faults may be so thin as to be easily mistaken for the ordinary jointing of the rocks they traverse. More often, however, there is a considerable space between their "walls" or "cheeks." This space is sometimes filled up with *débris* from the adjoining rocks, or with mineral matter deposited from solutions circulating within them, such as iron pyrites, oxide of iron, and other metallic substances, together with quartz, clay, &c. When a fissure is filled with matter injected from the earth's interior, such as basalt, it forms a *dyke;* when filled with mineral matter, like quartz, it constitutes a *vein*, and if a metallic mineral, or *ore*, be present, the vein is often termed a *lode*.

Fissures due to these disturbances may contain carbonaceous matters where they traverse the coal measures, which may at times be accompanied by metallic minerals. There is no essential difference between faults found in the coal-measures and the mineral veins ("fissure veins") of a metalliferous district. For instance, the great Minera lead lode is, throughout a part of its course, the chief fault of the Denbighshire coal-field.

The displacement in faults is not necessarily caused at one time; there is frequently clear evidence of repeated movement. They may occur in groups of two or three, with parallel or nearly parallel bearings, sometimes all dipping in the same direction, but usually two sets dip in opposite directions. There is often connected with these groups another series at right angles to them of a somewhat later origin.

The IGNEOUS ROCKS are now usually divided into a *plutonic* series, including such rocks as granite, which seem to have solidified at a great depth under enormous pressure; and a *volcanic* series, including rocks like basalt, which have consolidated at or near the surface.

The study of the igneous rocks is attended with much difficulty. Their mineral composition is usually complex, and often two rocks having the same composition present a totally different appearance, owing to the fact that they were cooled under different circumstances. They have no order of superposition, but have been erupted from below, and are found traversing the stratified rocks in veins or dykes, sills, or intrusive sheets, and irregular bosses. Moreover they contain no organic remains by which their relative ages can be satisfactorily established. The igneous rocks contain many valuable minerals, building stone, &c.

The METAMORPHIC ROCKS form an obscure group, often regarded as the lowest of the stratified rocks. Natural agencies like subterranean heat, water, and pressure, have so profoundly affected certain rocks that their original condition cannot be determined: hence the origin of such rocks as gneiss and mica-schist is still open to much discussion. Some metamorphic rocks are undoubtedly altered sediments and others altered igneous rocks. Serpentine, for instance, is believed to result from the chemical alteration of certain igneous rocks; whilst clay slate, regarded by some as a metamorphic rock, evidently results from the mechanical alteration of sedimentary matter of an argillaceous character. The Metamorphic rocks are very productive in a mercantile point of view, yielding marble, slate, serpentine, and quartz; whilst metallic veins containing copper, lead and tin frequently traverse these rocks. Gold and silver and many precious stones are among the valuables they contain. Plumbago or graphite, one of the three forms of carbon, is also a product of the Metamorphic System.

Rocks which contain no fossils are sometimes termed *azoic* or *hypozoic;* whilst those which yield traces of only the earliest forms, or what are supposed to be such, have been designated *eozoic.*

The Stratified Rocks have been arranged in three large groups, named according to their chronological succession *Primary*, *Secondary*, and *Tertiary*. But these names are now often replaced by the respective terms *Palæozoic*, *Mesozoic*, and *Cainozoic* (or better, *Cænozoic*), these terms having regard to the nature of the organic remains in the rocks, and indicating respectively the "ancient," "middle," and "recent" types of fossils. The following table of strata shows the sequence of the great groups, the oldest being placed at the bottom of the scheme:—

Quaternary	Recent. Pleistocene.
Tertiary or Cainozoic	Pliocene. Miocene. Oligocene. Eocene.
Secondary or Mesozoic	Cretaceous. Oolitic. Liassic. Triassic.
Primary or Palæozoic	Permian. Carboniferous. Devonian. Silurian. Ordovician. Cambrian.
Archæan	Pre-Cambrian.

Each of these groups will now be briefly described, commencing with the oldest and proceeding thence in regular ascending order.

The Archæan System includes an enormous thickness of rocks, principally gneiss and crystalline schists, which are generally regarded as metamorphic rocks. They are largely developed in the Scotch Highlands, in the outer Hebrides, and in the North of Ireland. It has been shown, in recent years, that the archæan rocks of the North-west Highlands have been subjected to stupendous earth-movements, whereby huge masses of rock have been violently disturbed and moved for miles along thrust-planes. The archæan rocks of Canada, studied by the late Sir W. Logan, consist of a lower group, about 30,000 feet thick, and an upper group which in places may be 20,000 feet in thickness: the former has been called the *Laurentian* and the latter the *Huronian* series, both names having geographical significance. Certain bands of crystalline limestone in the Laurentian series have yielded a curious structure, regarded by Sir W. Dawson as a gigantic reef-forming foraminifer and termed *Eozoon*—the oldest known form of life. Its organic nature is, however, extremely doubtful. It is notable that the Canadian Laurentians contain much graphite, or plumbago, and apatite, or phosphate of lime.

In certain parts of Pembrokeshire, Carnarvonshire, Anglesey, Shropshire and elsewhere are some very ancient rocks, consisting of altered stratified deposits and contemporaneous igneous rocks, which are generally termed *Pre-cambrian*. They have been extensively studied at St. David's by Dr. Henry Hicks. The Torridon sandstone of Scotland is also now regarded as Pre-cambrian.

THE CAMBRIAN SYSTEM, named by Sedgwick, is largely developed in North Wales, where it comprises (reckoning from below upwards) the *Harlech* and *Llanberis beds*, forming a great thickness of slates, grits, and sandstones; the *Menevian* beds, named by Hicks and Salter from the classical name of St. David's; the *Lingula flags*, so called from a little fossil formerly known as *Lingula;* and the *Tremadoc slates*, named by Sedgwick from a typical locality in Carnarvonshire. Cambrian fossils, though not numerous, are of profound interest, inasmuch as they represent the earliest known types of life. The most interesting are the Crustaceans, called *trilobites*, and a recent classification of the Cambrians, now commonly adopted, is based on the occurrence of characteristic trilobites. The Lower Cambrians form the *Olenellus* zone; the middle Cambrians the *Paradoxides* zone; and the Upper Cambrians the *Olenus* zone. Most of the best slates occur in the Cambrian system; those of Llanberis and Penrhyn being worked in the Llanberis series. The gold of North Wales is found in quartz veins, associated with the igneous rocks of the Cambrian system.

THE ORDOVICIAN SYSTEM, named by Lapworth after the old tribe of the Ordovices in North Wales, comprises the strata formerly termed Cambro-Silurian, and equivalent to the "Lower Silurian" of the Geological Survey. The Ordovicians include the *Arenig series* of slates and volcanic rocks, developed in the Arenigs, the Arans and Cader Idris; the *Llandeilo group* of shales and sandstones, so named from the locality in Carmarthenshire; and the *Bala* or *Caradoc series*, which includes the famous Bala limestone, rich in marine fossils. The old igneous rocks of Snowdon are referable to the Bala period; whilst those of the Lake District of Cumberland belong to the Llandeilo, and probably in part also to the Bala, age. The characteristic fossils in many of the Ordovician shales are the little *graptolites*, allied apparently to the modern sertularians. Phosphatic nodules occur in the Bala beds of North Wales.

THE SILURIAN SYSTEM, named by Murchison from the ancient British Silures, is typically developed in parts of South Wales, and in Monmouthshire, Gloucestershire, Herefordshire and Shropshire. It comprises the *Llandovery series*, known also as the *May Hill group*, both being geographical names; the *Wenlock series*, with the well-known limestone, at one time largely quarried at Dudley; and the *Ludlow* group, consisting of an upper and a lower member, separated by the Aymestry limestone. Both limestones yield splendid fossils, especially corals, crinoids, trilobites, brachiopods, and cephalopods. The veins of silver-lead ore, formerly of great value in Cardiganshire, occur in the Lower Llandovery rocks.

THE DEVONIAN SYSTEM received its name from Sedgwick and Murchison in 1836 in consequence of its development in Devonshire. It consists of a lower group of conglomerates, shales and sandstones; a middle group of limestones; and an upper group, principally of sandstone. The limestone, known as the Great Devon Limestone, is rich in corals, and is quarried as a marble as well as a building-stone. The Devonian slates of West Devon and Cornwall are disturbed by huge bosses of granite, forming a series of heights stretching from Dartmoor to the Land's End. The slates known locally as *killas* are, for the most part, imperfectly cleaved, but good roofing slates are worked at Delabole and Tintagel in North Cornwall. Granite is extensively quarried. Near the junction of the killas with the granite rich lodes of tin- and copper-ores occur; and it is notable that while these veins usually course in a nearly E. and W. direction, the lead-lodes run almost N. and S.

THE OLD RED SANDSTONE is probably the equivalent of the Devonian rocks, but appears to have been deposited in lakes and not in the sea. In Scotland a series of these ancient lakes has been mapped out by Sir A. Geikie. A large area in Herefordshire, Monmouthshire, and part of S. Wales, is occupied by Old Red rocks, consisting of marls, sandstones, pebble-beds, and impure nodular limestones termed "cornstone." The Caithness and Arbroath paving-flags belong to this system. The Old Red Sandstone yields the remains of curious ganoid and other fishes, and large extinct crustaceans. In the South of Ireland fresh-water lacustrine beds occur of the age of the Upper Old Red. The upper and the lower groups in Scotland are divided by a strong break. It is notable that the upper old red passes conformably into the overlying carboniferous rocks, and is sometimes regarded, locally, as representing the shore-deposits of the early carboniferous sea.

No geological era has bequeathed to us a more valuable deposit than the CARBONIFEROUS PERIOD, which succeeds the Old Red Sandstone. Interstratified with the rocks we find those valuable seams of coal which are the greatest wealth of a country.

The system is typically separable into four well-marked groups—viz., the Lower Limestone Shales or Carboniferous Slates, the Mountain Limestone, the Millstone Grit, and the Coal Measures.

The Carboniferous strata throughout are composed of frequent alternations of sandstones, shales, limestones, coals and ironstones.

The *Lower Limestone Shales* form a series of beds between the Old Red Sandstone and the Carboniferous Limestone. The *culm measures* of Devonshire may belong partly to this series. In Northumberland the Lower Carboniferous strata have been called the *Tuedian* beds. The Lower Carboniferous beds are very scantily developed in some districts and in others attain a great thickness, so that it is difficult to state their average thickness, although frequently given at 1,000 feet, the Mountain Limestone at from 500 to 3,000, the Millstone Grit at 600, whilst the true Coal Measures vary from 3,000 to 12,000 feet thick. The *Calciferous Sandstone series*, of Scotch geologists, includes the lower part of the Mountain Limestone and the Lower Limestone shales.

The *Carboniferous* or *Mountain Limestone* is, perhaps, the most distinct and unmistakable group in the whole crust of the earth. Its limestones and fossils are so marked and peculiar as to form a guiding post to the miner and geologist. By far the greater part of the workable coal seams of Scotland is included in the Mountain Limestone group. The Mountain Limestone is absent from both the Shrewsbury and South Staffordshire coal-fields, the true coal measures there resting on the Cambrian and Silurian rocks; whilst in Worcestershire, the Forest of Wyre Coal Measures rest on a bed of Old Red Sandstone. In Shropshire the Silurian rocks form the general foundation to the carboniferous formations.

Between the Mountain Limestone and the Millstone Grit in Yorkshire and the North of England is a thick series of black shales with sandstones, limestones, and even coal seams, called by the late Prof. Phillips, the *Yoredale rocks*. They take their name from Yoredale, or Wensleydale, in Yorkshire; and are otherwise known as the *Upper Limestone Shales*. The *Bernician* beds of Northumberland were named by Prof. Lebour, and include representatives of the Yoredale rocks as well as of the Carboniferous Limestone.

The *Millstone Grit* is composed of a series of hard and coarse sandstones and shales, usually of a grey, white, or yellow colour, but occasionally red. It is sometimes absent and rarely attains a greater thickness than 1,000 feet, although it is said to reach, in some places, a maximum of 5,000 feet. Occasionally it contains thin seams of coal, but is known to miners by the name of the Farewell Rock, suggesting that, on a sinking reaching this rock, farewell has been said to the coal seams of the true Coal Measures.

In Derbyshire the Mountain Limestone consists of an enormous mass of calcareous rocks almost destitute of sedimentary matter and entirely so of coal. Further north in Lancashire and Yorkshire workable coal seams are found at a stage earlier than the true Coal Measures—in the Millstone Grit. Still further north, in Northumberland, several beds of coal are found near the base of the Mountain Limestone formation. The coals of the Mountain Limestone of Scotland occupy a position similar to that of those in Northumberland, but in Scotland the sedimentary strata are more largely developed.

The true *Coal Measures* furnish us with those valuable beds of coal which contribute so much to our country's prosperity and power. The series sometimes occurs immediately above the Mountain Limestone, the Millstone Grit being then wanting. One of the most notable features in its composition is the frequent recurrence of seams of coal, or of bituminous shale, all speaking of an enormous profusion of vegetable growth. The organic remains of the Coal Measures are peculiarly well defined. The fishes are chiefly of large size, and in certain fields there are evidences of terrestrial life in the skeletons of amphibians, fragments of land shells, and remains of insects. The great feature of the period, however, is the abundant flora, which comprises forms which are now only distantly represented in tropical swamps and jungles, and point possibly to a tropical condition of climate. The coal-measure flora consists chiefly of lycopods, or club-mosses, some of gigantic size; horse-tails, or *equisetaceæ;* ferns and conifers. In some coals, and associated rocks, the resinoid spores of lycopods are well preserved.

It must be borne in mind that the coal is composed of carbon, hydrogen, and oxygen, elements which enter into the composition of vegetable organisms, so that the coal seams are not the result of direct mechanical deposition of mineral matter, like the associated sandstones and shales. The most reasonable theory as to the origin of coal seems to be that it is the remains of vegetable matter which became decomposed and mineralized on the spot where it grew, and where it is now found. This vegetation might have grown on the borders of great lakes, estuaries, and vast lagoons. In process of time these layers of vegetation, might, by subsidence, be carried down beneath the sea-level, and the water consequently flowed over them, depositing layers of sand, silt and mud, which we now find alternating with our coal seams as beds of sandstone and shale. The downward movement must have been irregular or intermittent, and in the long pauses the sediment would fill up the lagoons, and fresh jungles would spring up, which, when the downward movement set in afresh, would be in turn entombed beneath the silt and mud of the sea waters. Each coal seam thus represents the vegetation of an old land surface, and the alternations of sandstone, shale, fireclay, &c., represent the different sediments which were brought together by the combined action of the sea and estuarine waters during the slow and irregular subsidence of their beds. It will be noticed that nearly all coal-fields are basin-shaped. The synclinal form is, however, usually due to earth-movements subsequent to the formation of the coal seams. Each coal seam has characteristics peculiarly its own, often prevailing over very wide areas. Besides the distinctive features in the formation of coal seams, the coal obtained from one differs from that yielded by another, and frequently one coal seam gives different qualities of coal. The differences arise chiefly from the difference of chemical composition, as the hydrogen, oxygen and carbon are present in the coal in varying states of combination.

The theory that coal has been formed in its present position, so strongly advocated by Logan, after his observations on the underclays with their stigmarian rootlets, has of late been warmly opposed by certain French geologists, who feel justified in reverting to the old view that most coal consists of altered vegetable matter drifted into estuaries and lagoons, and therefore removed from its original site of growth.

Different samples of coal give out amounts of heat during combustion which are not always anticipated from their chemical composition, or expected from analysis. For this reason the different coals used in boiler tests should be experimented on, to ascertain the actual heat of combustion, as well as that estimated from analysis, otherwise the results of the tests made may be quite misleading.

Commercially the Carboniferous is the most important and valuable system to man. It yields building stone of the best quality, flagstones for paving, grits for grindstones, limestones for many purposes, marbles, fire-clay, ironstone, ochre, alum, copperas, and coal of various qualities. The Mountain Limestone yields ores of lead, zinc, antimony, and sometimes silver and gold.

Succeeding the Coal Measures we have the **PERMIAN SYSTEM**, so named from its development in the ancient kingdom of Perm in Russia. The system consists of red sandstones and marls, conglomerates and breccias, with limestones which are usually magnesian, or dolomitic. In some places they contain 44 per cent. of carbonate of magnesia associated with the carbonate of lime. The Permian fauna represents that of the late palæozoic era. In Germany the ***Kupferschiefer***, represented in this country by the marl-slate of Durham, has been long mined as an ore of copper.

The dolomite or magnesian limestone of the North of England is used as a source of magnesian salts, and also as a building-stone. The Permians vary in thickness, but may be as much as 3,500 feet in this country.

The **TRIAS** or **NEW RED SANDSTONE**, the latter name being given because reddish hues prevail throughout its sandstones and shales in the British Isles, consists of two members—a lower group of sandstones and conglomerates, called the ***Bunter***, in allusion to the variegated tints of the rocks; and an upper group, consisting chiefly of red marls and sandstones, and known as the ***Keuper***. Where typically developed, as in Germany, these two members are separated by the ***Muschelkalk***, or shell-limestone; and the term Trias has reference to this threefold character. The limestone is wanting in England, and as the best fossils are to be found in this rock we have not the local facilities for an extensive knowledge of the flora and fauna of the period. The plants of the Triassic strata have a strong resemblance to the flora of the Lias and Oolite above, consisting of ferns, cycads, and conifers. The industrial products yielded by the system are sandstones of varying quality, gypsum, and rock-salt. All the salt of Cheshire, whether mined as rock-salt or pumped as brine, is obtained from the Keuper or New Red Marl: the Lower Keuper sandstones occasionally contain copper-ores and other metallic minerals, as at Alderley Edge, in Cheshire. The Trias is at least 4,000 feet thick in England.

The LIAS, which follows the Trias, or is separated from it by a thin series of strata known as the ***Rhœtic*** or ***Penarth beds***, is composed of dark argillaceous limestones, and bluish clays, forming the ***Lower Lias***, separated by the marlstone, or ***Middle Lias***, from the group of shales and limestones which constitute the ***Upper Lias***. The fauna of the period is diversified and interesting. Upwards of 120 species of ammonites have been discovered, and belemnites are also very numerous. The most interesting feature in the life of the age is the appearance of marine reptiles of extraordinary size and structure, such as the Ichthyosaurus and Plesiosaurus.

The well-known ironstone of Cleveland, in Yorkshire, which lies in thick beds, is in the Middle Lias. Ironstones in other parts have also been found in the Lias. The argillaceous limestones are largely quarried for mortar and hydraulic cement. Jet is obtained from the Upper Lias of Yorkshire, and alum-shales were formerly worked to a large extent near Whitby.

The Lias is about 1,200 feet thick.

THE OOLITES.—Above the Lias in the South and West of England are the Oolites. These consist of alternations of oolitic limestones, calcareous grits, shelly conglomerates, yellowish sands, and thick bedded, bluish grey clays more or less calcareous. The Oolites and the Lias are now frequently united as one large group, known as the *Jurassic system.*

The flora of the oolitic era was extensive, as may be gathered from the fact that at Brora, in Sutherlandshire, there is the thickest stratum of coal found in any British Secondary rock. It has been worked intermittently for a long period, the seam being 3½ feet thick. In Yorkshire there is a group of coal-bearing rocks of Lower Oolitic age, consisting of sandstones, shales, and ironstones, with seams of coal, some of which may reach a thickness of 18 inches. These "Moorland coals" rest on underclays, and are associated with many vegetable fossils, especially the remains of cycads, ferns, and horse-tails. The Oolites yield limestones like those of Bath, Portland, Ketton, and Ancaster, excellent for building purposes. They also yield Fuller's earth. Lignite or wood-coal is likewise found, but is of no economic value. The Oolites are about 1,800 to 2,000 feet thick.

THE CRETACEOUS SYSTEM.—The lowest member of the Cretaceous group is the Wealden; this deposit is of fresh-water origin, the bones of terrestrial animals and the relics of land plants, testifying that the Wealden owes its existence to some great river which brought down mud, sand, &c., from the continent it drained, depositing the detritus in a delta. The Cretaceous beds above the Wealden are of marine origin. They include the Lower Greensand, Gault clay, and Upper Greensand, followed by the Chalk, which is the most conspicuous member of the system. Some geologists unite the Lower Greensand with the Wealden beds, under the name of the *Neocomian* group. The Chalk is for the most part a white earthy limestone, upwards of 1,000 feet thick, with bands of flints in the upper portion. The organic remains are sponges, corals, star fishes, molluscs, crustacea, fishes and reptiles. Fossil plants are comparatively rare. Industrially, the chief products of the system in Great Britain are chalk and flint. Several of the workable coal seams of Vancouver Island, and British North America, belong to the Cretaceous epoch. The Cretaceous system is over 4,000 feet thick.

The TERTIARY SYSTEM embraces a large series of clays, sands, pebble-beds and other strata, overlying the Cretaceous rocks, and well developed in the two areas known as the London and Hampshire basins, the latter extending into the Isle of Wight. The Tertiaries have been arranged in a number of groups, of which the principal are termed in ascending order, *Eocene*, *Oligocene*, *Miocene*, and *Pliocene*—these names having reference to the proportion of recent and extinct mollusca in their fossil fauna.

At the base of the Eocenes are the strata known as "The Lower London Tertiaries," comprising the Thanet sands, the Woolwich and Reading series, and the Blackheath or Oldhaven beds. Then follows the London Clay—a stiff clay of bluish or brown colour, reaching a maximum thickness of about 500 feet, and containing zones of septaria, or nodules of argillaceous limestone. Iron pyrites and selenite are also common in this clay. The Bagshot beds, above the London clay, include the famous potters' clays and pipe-clays of Poole, in Dorsetshire, and Bovey Tracey in Devon. At the latter locality are beds of lignite, known as "Bovey Coal," associated with numerous plant remains, but of little or no industrial value. The Bovey Coal was formerly regarded as miocene, but most geologists now hold that although miocene strata are very largely developed in parts of the Continent, they are not represented in Britain.

During the early Tertiary period volcanic action was energetic in the British area, as attested by the thick sheets of basaltic rocks, or old lava-flows, in the N.E. of Ireland and the West of Scotland. The celebrated columnar basalt of the Giant's Causeway belongs to this period.

The Oligocene strata are represented in this country by the fluvio-marine series of the Isle of Wight. In the absence of Miocenes, we then pass to the Pliocene beds, which are fairly developed in East Anglia, where they consist chiefly of the shelly sands known as " Crags." The total thickness of the British Tertiaries may reach about 2,000 feet.

Most geologists regard the strata above the Tertiaries as forming a distinct group known as the POST-TERTIARY OR QUATERNARY SYSTEM. This group includes the Pleistocene deposits, which consist chiefly of clays, sands, and gravels, in the formation of which ice is believed to have played a more or less important part, and which are consequently referred to the Glacial epoch, or Great Ice Age. This period of the earth's history was marked by the prevalence of arctic conditions over a large part of the northern hemisphere, but it is not always easy to determine whether the " drift " has been accumulated by the action of land-ice or of floating ice. Ice-borne boulders and erratics are common in many parts of Britain, and the great Northern drift comes down as far south as Finchley, on the north of London.

A remarkable instance of rock believed to have been deposited in glacial times, is that at the Stevens Mine, in Mount M'Clellan, California, its altitude being 2,500 feet. At a depth of from 60 to 200 feet, the vein, consisting of silica, calcite and ore, and also the surrounding wall rocks, are a solid frozen mass. Many other mines in the same vicinity have similar belts of frozen ground, and as they are at considerable depths from the surface, where there are no openings for air currents, the theory is that the frozen mine is due to imbedded icebergs of the glacial period. There has been no diminution in the frost in descending, and the frozen material is so hard as to render the miner's pick and drill of no use in excavating it.

All the accumulations formed since the Glacial period are known as *Recent* or *Modern* deposits. At its close the present distribution of sea and land seems to have been established, and the earth to have then had its present flora and fauna, with the exception of some local removals of certain plants and animals and the general extinction of a few species. Theoretically the accumulations of the present era are not only of high interest in themselves, but of prime importance in furnishing a key to the complicated phenomena of former epochs. Practically they present many important features to the farmer, engineer, and navigator, and furnish us industrially with such products as brick clay, sand, marl, and peat, and in other countries pumice, sulphur, brown coal, and amber.

CHAPTER II.

SEARCH FOR COAL.

In Untried Districts by the Application of Geological Knowledge—Search in both an Unknown and Proved Coal-field by Boring—The Operation of Hand-boring by the Use of Rigid Rods—Boring with the Diamond Drill—Particulars of Deep Bore-holes—Application of the Result obtained by a Series of Bores—Search for Coal in South Staffordshire—Search for Coal in Kent.

THE search for coal in an altogether unknown district is putting the geologist's knowledge to a practical test. He carefully examines the district, making use of roadsides, quarries, cliffs, protruding rocks, the beds of streams, railway cuttings (if any), and ploughed fields. He may by these means be able to ascertain whether the rocks are carboniferous, and if so he must pursue his investigations further. An actual outcrop of a coal seam may be discovered, or the dark appearance of the soil over a freshly ploughed field may indicate the proximity of one, in which case he would remove the soil at places most likely to prove the existence or otherwise of coal, and having found it, he would probably drive a heading into it to prove its dip for some little distance, the direction and amount of which would be noted. But should he have proved the rocks to be unquestionably carboniferous (and the fossils should be an infallible guide to him) his failure by a personal examination to find coal will by no means be conclusive, and he will have to resort to boring. Sometimes, too, the coal measures are covered by the secondary or tertiary rocks, beneath a comparatively thin covering of which the coal measures may be found. Even in a district where coal has been proved and worked, it is often necessary to bore, as before deciding on the site for a shaft it is of importance to know the direction and amount of dip, the nature of the rocks to be sunk through, and the quality and number of the seams to be won. The outcrop of a seam, it must be remembered, affords but little indication of the quality of the coal, as it is much deteriorated from being exposed to atmospheric influences. A personal survey having met with encouragement, boring should be resorted to so that the depth and thickness of the seam or seams, and their inclination, may be ascertained to guide the sinking of shafts to win the coal.

The operation of boring may be classified under two distinct heads. The first proceeds by means of a suspended tool, to which is given a percussive motion by either hand or steam-power, and also a rotary motion either by the apparatus itself or by the attendant, the latter motion taking place between each fall of the tool. The second consists in giving motion to a rotating tool which is hollow in the form of a tube, as in a drill used by a Diamond Boring Co.

The ordinary process of hand-boring consists first in erecting the *headgear*, Fig. 4, over the bore-hole for the purpose of supporting the tackle by which the bore-rods are drawn out of or lowered into the hole, when it is necessary to clean out the hole or change the cutting chisel. The headgear may be made of three good, sound Norway spars 8 or 10 inches in diameter, 40 feet high, and formed as a triangle; or four legs may be used with cross pieces; a pair of blocks are hung from the top, or they may be further multiplied according to the depth of the hole. A rope is passed through the blocks, one end of which is attached to a winch and the other hangs loose over the end of the bore-hole. The windlass should be 12 inches in diameter and be fitted with a pawl and brake, so as to regulate the speed with which the rods are lowered. The following are the tools used as described by Greenwell in his *Mine Engineering*:—

The **Ordinary Bore-Rods**, Fig. 5, 1 inch square, made of best wrought iron 6 or 9 feet long with a male screw at one end and a female screw at the other.

Lengtheners are from 4 to 30 inches long, and of the same section as ordinary rods.

Chisels, see Fig. 6, are 18 inches long and from 2 to 2⅓ inches in breadth on the face, which is of the best steel; they weigh about 4½ lbs.

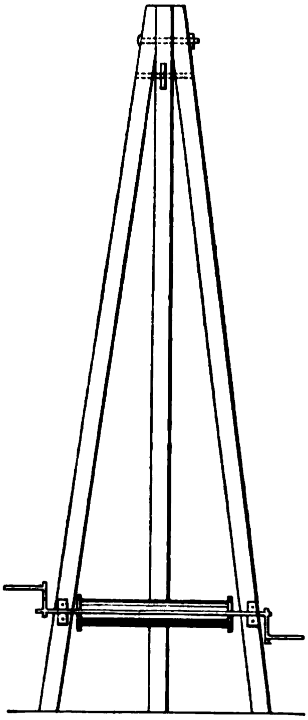


Fig. 4.—Head Gear and Windlass for Boring.

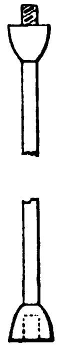


Fig. 5.—Ordinary Bore-Rod.

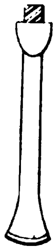


Fig. 6.—Chisel.

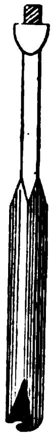


Fig. 7.—Wimble.

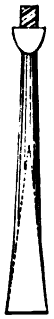


Fig. 8.—Bêche.

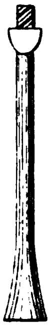


Fig. 9.—Rounder.

The **Wimble**, Fig. 7, is 3 feet long, the lower 2 feet being cylindrical, the bottom being partly covered for the purpose of retaining the core in the instrument. It is made of wrought iron of a size suitable "to follow" the chisel.

There is an opening a little up one side of it to admit of the loosened material entering it. The wimble is also used for boring through clay.

The **Sludger** is also 3 feet long and somewhat resembles the wimble; the lower cylindrical part has no opening in it, but the bottom has a clack for the purpose of retaining borings of a soft nature.

The **Bêche**, Fig. 8, is used for the purpose of extracting broken pieces of rod in case of fracture. It is about 25 in. long and hollow for about 16 in. up. The diameter of the cavity at the bottom is 1¾ in., tapering up to ⅝ in.

In case of the rods breaking, the part above the fracture is withdrawn, the bêche being screwed on to take the place of the broken rod. It is then lowered sharply on to the broken rods in the hole, the force causing them to enter the tapering cavity of the bêche, which grips them whilst they are drawn up.

Rounders, Fig. 9, have the appearance of the bêche, but are solid and well steeled at the bottom. They are used for breaking off any irregularity in the holes.

The **Bracehead**, Fig. 10, is a piece of oak or ash 3 feet long and 3 inches in diameter in the centre tapering slightly towards either end, and this is passed through an eye formed in a piece of iron screwed on to the top rod. A man stands at each side to lift the rods and turn them partly round in the hole. For greater depths than 10 fathoms a double bracehead is used. It consists of two similar pieces of wood passed through two eyes in the iron at right angles to each other. Four men stand one at each arm of the double bracehead, to lift and turn the rods, and thus a depth of 20 fathoms is attained.

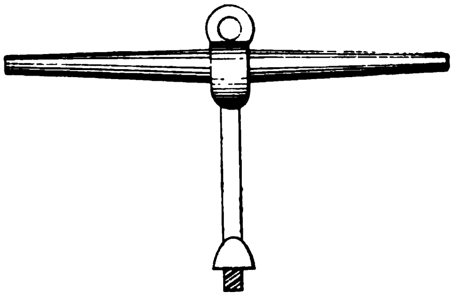


Fig. 10.—Bracehead.

A **Brake**, Fig. 11, is used below a depth of 20 fathoms. This is simply a lever of Memel fir 10 or 12 feet long, the fulcrum being 18 inches or 2 feet from one end. At its extremity is placed an iron crook, a rope being attached to it and to the bracehead. Two or more men at the other extremity press down the brake and so raise the rods, another man then turns the bracehead partly round, and the men at the brake let it go suddenly, causing the chisel to cut into the ground. When it is desired to take the rods out, the bracehead is unscrewed and a *runner*, Fig. 12 (A), is attached to the rope which fits on to the *topit*, Fig. 12 (B), which is then screwed on to take the place of the bracehead. The windlass is used for drawing up the rods as far as the shear-legs will allow, when a *key*, Fig. 12 (C), is passed under the bottom joint above the bore-hole, and this key holds them in position whilst another key is used for unscrewing them.

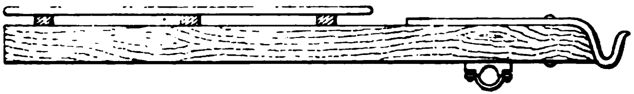
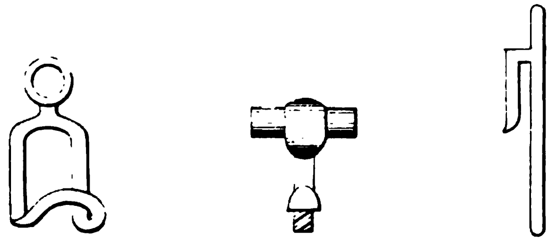


Fig. 11.—Brake or Lever for Boring.


(A) Runner. (B) Topit. (C) Key.
Fig. 12.—Boring Tools.

Another topit is screwed on the remaining portion of the rods, and by means of the runner another portion lifted by the windlass. The process is continued till all the rods are drawn out.

Fig. 13 shows a group of improved boring tools, as manufactured by Messrs. Thornewill & Warham, of Burton-on-Trent, and supplied by them singly or in sets.

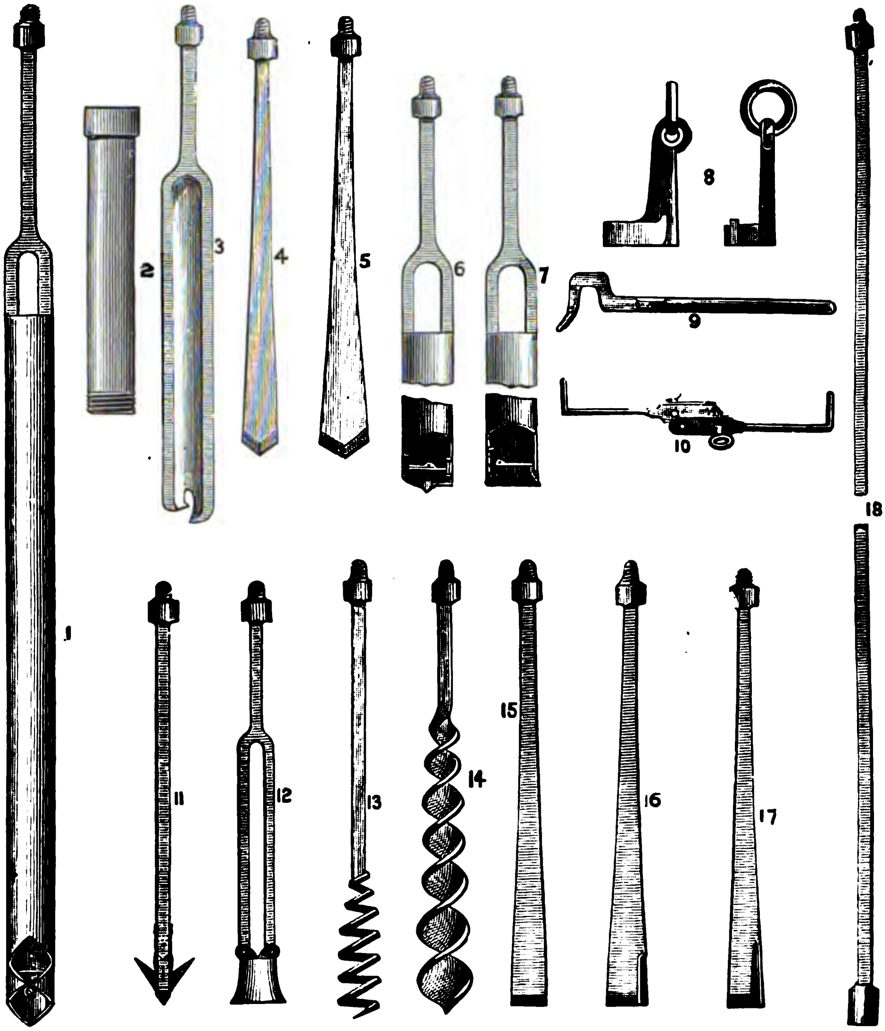


Fig. 13.—Messrs. Thornewill and Warham's Improved Boring Tools.

List of Boring Tools in Fig. 13.

1. Shoe-nose shell with valve for bringing up loose stuff.
2. Wrought-iron screwed well bore pipes.
3. Auger for clay and stiff soil.
and 5. V-nose chisels for hard ground.
6. Shell-auger with valve for loose and wet soil.
7. Bell-shell with valve for loose gravel.
8. Lifting dog for raising rods.
9. Pair of rod-wrenches for screwing and unscrewing rods.
10. Levers for turning rods.
11. Spring dart for drawing pipes in bore-holes.
12. Bell-box for bringing up broken bits.
13. Spiral worm for extracting broken rods.
14. Worm auger for loosening stuff in bore-holes.
15. Square-nose chisel.
16. S-nose chisel for hard strata.
17. T-nose chisel for hard strata.
18. Rods with screw joints in 5 and 10-foot lengths.

1. Shoe-nose shell with valve for bringing up loose stuff. After the cutting tool has been worked for some time the loosened material interferes with its further action, and so it is withdrawn from the bore-hole, and changed for the shoe-nose shell which is then lowered into the hole. Its pointed end is designed to assist its descent. When lowered as far as possible, it is turned round, the rotary motion causing it to descend further as the eroded material passes upward through the valve which afterwards keeps it in place during the ascent.

2. Wrought-iron screwed well bore-pipes. These are used where from the nature of the ground the bore-hole would otherwise partially or wholly close, and are generally made in 10-foot lengths. The first pipe is well steeled and sharpened and very carefully driven or pressed down until its head is nearly level with the ground. By means of pipe-clamps another length is then screwed to the first, the whole being afterwards driven down another length. More pipes are added until the bottom of the hole is reached. Much care is needed to keep the pipes perpendicular. Where a hole is lined with pipes the diameter has of course to be reduced on continuing the boring below them, and if a number of such sets have to be inserted allowance must be made beforehand for the necessary successive reductions of the bore-hole. If this is ignored, it may become necessary on reaching some soft running bed, to enlarge the hole from the surface to admit of the lining. If the boring is to be carried to a great depth, it is almost certain that lining will be requisite.

3. Auger for clay and stiff soil. This tool is used in the same way as an ordinary auger. The clay passes into the interior of the tool and is there withdrawn as a core without the aid of a valve.

4 and 5. V-nose chisels for hard ground. These are shaped so that in coming into contact with the material to be cut the best effect is obtained; they are more suitable for hard rock, and better stand the frequent sharpening required, than the square-nose chisel 15 which, however, is used in all moderately firm ground. In some very hard strata the S-nose chisel 16 and the T-nose 17 are found to cut better.

6. Shell-auger with valve for loose and wet soil. This is used where no cutting tool is required and where 1 and 3 are also unsuitable. After being lowered in the bore-hole till it touches the loosened material the tool is turned round in the same way as 1 and 3. It is made longer than 3, being the same length as 1 with a valve which retains the material in the act of withdrawing it. If the valve fits tightly it effectually prevents the return of the pasty, or liquid material which the tool is specially intended for, unless it becomes clogged by grit so as to prevent the proper closing of the valve. As the valve wears some leakage may be expected, it therefore requires attention, as indeed do all valves, springs, and working parts during the continuance of the boring.

7. The Bell-shell with valve for loose gravel is used in the same way as 1, 3 and 6, the tool being square on the bottom and shaped so as to pass through the gravel, which is afterwards retained in the interior by means of the valve until reaching the surface when it is taken out for examination.

8. Lifting dog for raising rods. The rope is secured to the ring of a dog; the lower portion of the dog is then passed under but clear of the shoulder formed at the head of the top rod 18. The portion of the dog which receives the rod is made of a size which allows of the bore-rod passing into it, yet, being less than the shoulder of the rod it comes into contact with the projection on being raised sufficiently. By means of the windlass and rope the rods are raised as far as the head gear will allow. Another lifting dog is then placed on the ground under the nearest rod-joint, the whole length of rods being lowered a few inches till the shoulder at the joint enters the lifting dog, which then takes the weight of the rods, being itself supported by the ground around the bore-hole, while the pair of rod-wrenches 9 are used to unscrew the rods. The unscrewed length is placed in a suitable position for easy re-attachment when required, after which the rope is lowered by the windlass, disconnected from the lifting dog last used, and

attached to that sustaining the rods. By means of the windlass these are raised as far as the headgear will allow, and another portion of rods unscrewed as before.

10. Levers for turning the rods. One or both of the screws shown on the drawing may be withdrawn and the two portions of the tool placed against two sides of the rod 18 under the shoulder formed at the head. The screws are replaced and tightened after which the levers may be used for lifting the rods in the bore-hole and for turning them; or they may be used for turning only.

11. Spring-dart for drawing pipes in bore-holes. Sometimes after completing a bore-hole it is abandoned; as there is no longer any object in keeping it open, any lining pipes used during construction are, if possible, recovered. The spring dart is attached at the end of an ordinary rod and lowered within the tubes. The projections at the bottom are pressed out by springs which yield as the dart is lowered and spring out beyond the sides directly the end of the tube is reached. When the rods are raised the projecting parts catch the pipes, and if the weight be not too great, the whole may be lifted at once and portions be unscrewed as they come to the surface. But if the weight is too great for the spring-dart, other means have to be devised for the recovery of the pipes, such as cutting them with a special instrument and lifting them in sections, or unscrewing them length by length by a conical plug which also grips them for lifting; but it is extremely difficult to recover lining tubes from a deep bore-hole.

12. Bell-box for bringing up broken bits. This is tapered at the bottom and has a screw thread cut inside. If a section of rods is allowed to run back into the bore-hole, or if they should break from any cause while boring is going on, the bell-box is lowered at the end of ordinary rods and turned round on reaching the top of the rods in the bore-hole. If successful in gripping them they are withdrawn, if unsuccessful the spiral-worm for extracting broken rods 13 is tried. It is placed at the end of the ordinary rods and as it is turned round the spring at the lower extremity encloses the upper portion of broken rods, the weight of which tends to elongate the spring and make its hold the tighter. 12 and 13 are also sometimes applied to remove a stone which may get dislodged and form an obstruction to the descent of the rods in the bore-hole.

14. Worm-auger for loosening stuff in bore-holes. If the eroded material at the bottom of the bore-hole becomes consolidated through the shoe-nose shell pressing upon it when removing material from above, it may be necessary to loosen this before it can be removed and 14 is designed to effect this object. On being turned it penetrates any material so pressed together, and leaves it loosened when the tool is withdrawn to give place to 1.

18. Rods with screw joints in 5 and 10-foot lengths. These are the same as the ordinary bore-rods previously described: the shoulders formed at the extremities are for strengthening the rods and giving a means of sustaining and turning them.

Borings are usually contracted for, the master borer finding all labour and materials. A contractor's scale of charges is usually 3*s.* 6*d.* a yard for the first 10 yards; 7*s.* a yard for the second 10 yards; 10*s.* 6*d.* for the third 10 yards, and so on in arithmetical progression, increasing by 3*s.* 6*d.* at each 10 yards. To find the cost then of a boring executed in the ordinary way we have:—

$$C = D\left\{\frac{3s.\ 6d. + \left(\frac{D}{10}\times 3s.\ 6d.\right)}{2}\right\}$$

where C = the cost and D the depth in yards. Thus a boring 100 yards would cost

$$100\left\{\frac{3s.\ 6d. + \left(\frac{100}{10}\times 3s.\ 6d.\right)}{2}\right\} = £96\ 5s.\ 0d.$$

but these prices would not apply to basalt, whin, or other excessively hard rock.

The depth of a bore-hole is measured by the number of rods, and the kinds of strata judged by the borings brought up by the tools used. Changes in the stratification are noted by the charge-man, who is guided by the sound and sensation through the hands when the rods fall, and he marks each change on the rods.

As the depth of a hole bored in this way increases the rate of progress decreases, and when a great depth has been reached a considerable amount of time is occupied by the necessary changes in the tools, and very little in actual boring. In cutting hard rocks far beneath the surface this system is extremely slow and expensive. A steam engine may be used, so as to give more power in lifting the weight of rods, which becomes excessive as the bore-hole reaches a great depth, but even then the process does not admit of rapid boring. The vertical engine usually gives motion by means of belting to a shaft carrying a cam roller, and this lifts a wooden beam or lever near that end from which the boring rods are suspended. At every two or three revolutions of the small engine the cam roller makes one, thus raising the beam and allowing it to fall. The other end of the beam rests in a trestle, which acts as the fulcrum, and allows of change by sliding the trestle nearer the rod-end of the lever. Between it and the cam roller is placed a strong, upright column to receive the lever on its descent, and this is protected on the top by a cushion of india-rubber to break the shock.

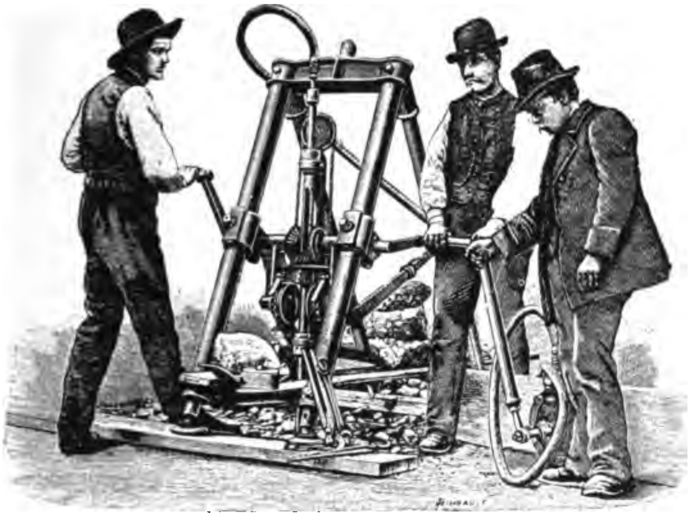


Fig. 14.—SCHRAM'S DIAMOND PROSPECTING DRILL.

The ordinary method of hand boring is slow and costly, especially in hard rocks, and there is a great disadvantage in the form of drill which cuts the rock into dust or slime in the bore-hole, so that when samples are brought to the surface for examination it is difficult, if not impossible, for even experienced borers to classify the strata. The solid cores of the rock obtained by the Diamond drill afford a much more satisfactory means of determination, as the samples withdrawn may be judged by their appearance, their markings, fossils, thickness of bed, and chemical tests of composition.

A hand-power prospecting diamond drill is arranged in the manner shown in Fig. 14, which is suitable for boring holes at any angle to a depth not exceeding 300 feet and a diameter of 1⅝ inches, yielding cores of ⅞ inch.

The hollow boring rods are revolved by the hand gearing, worked by two men, as shown; the third man is working a pump and forcing a supply of water down inside the rods: this keeps the boring head cool and washes away the dust outside the rods before returning to the surface. Pressure is put on the rods by means of a weighted lever, the number of weights being reduced as the depth of the hole increases. The cores are brought up in 2-foot lengths; the drill will also bore horizontal holes or holes at any desired angle for proving lodes or for tapping water when approaching old workings.

The drill, which is manufactured by Messrs. Schram, of London, has been largely used for prospecting purposes in various parts of the world. The outfit is as follows:—

1 crown set with diamonds.
6 blank crown bits.
200 feet of boring rods with couplings.
1 hand pump with hose, &c.
2 core barrels.
2 core breakers.
2 core lifters.
1 crown chuck.
1 set of tools for setting diamonds, and
Complete set of spanners, etc., necessary for erecting and working the machine.
Price of machine complete, with above outfit, 170*l*.
Price of extra crown set with diamonds, 25*l*.
Price of extra boring rods, 5*s*. 6*d*. per yard.
Weight of machine complete, 12 cwt.
Weight of heaviest piece, 150 lbs.

The rate of drilling varies greatly according to the hardness of the strata passed through. The average speed is from 6 feet to 12 feet in a shift of 8 hours.

Down to a depth of 100 feet two men are sufficient to work the machine, the number of revolutions being from 60 to 70 per minute; beyond that depth, however, three or four men will be required. The machine is supplied with an ingenious apparatus for raising the rods. Instructions are also sent for the re-setting of the diamonds on the spot, so that the boring head need not be sent away for this purpose.

If steam power is available both the drill and the pump can be driven by a small engine instead of by hand. The portability of the machine, however, as a hand-drill is one of its principal features, and if steam is to be used it would be more satisfactory to have the full-sized drill next described.

In boring deep holes by the diamond drill a wrought-iron H-shaped frame is used to carry the machinery necessary for boring, pumping, and raising the rods. A portable engine is used to do all the work, and the system requires only the services of two or three attendants. The power is transmitted from the engine by means of belting and an arrangement of gearing. A set of shear-legs is erected which carries a pulley immediately over the bore-hole, and one end of a chain passes over this pulley, the other being attached to a power-crab, which may be set in motion by the steam engine when desired. By this means the rods are raised and lowered. The machine carrying the cog-wheels and shafting is moved when it becomes necessary to draw the rods. To render this easy a short length of rails is laid down to receive the machine, and facilitate changes in its position.

The boring rods are carried, and receive their rotary motion by means of a vertical hollow rod. Attached to the upright side-frames of the machine are slides, and in these work a cross-head having a rise or fall the length of the slides. It supports and guides the vertical rods, so as to ensure their being perpendicular.

This cross-head also conveys the necessary pressure to the crown whilst rotating, balance weights being attached to it by chains and pulleys.

The pressure is controlled and regulated by means of the weights. The upper part of the rod is slotted, thus allowing of its upward movement through the cog transmitting the power, and on the top is placed a water union, joined up to a force-pump by means of flexible hose and iron pipes. The pump is set in motion by means of a belt and suitable gearing.

The boring tool consists of different parts, and is shown in Fig. 15, G being the crown, F the core-trap, E and D core-tubes, and C a sediment tube.

The crown, G, is a steel ring, having grooves cut in it as shown in the drawing, which enable the water to pass freely under it as the boring proceeds. There are 9 diamonds placed in the crown, 3 being inside, 3 outside on the base, and 3 on the circumference. For large holes more diamonds are fixed in the crown. The diamonds are placed in holes prepared to receive them according to their shape, and the metal is drawn round on every side of each diamond by means of a punch, until only a very small portion of the stone projects beyond the surface of the steel ring.

The crown, G, screws into the core-trap, F, which is a short length of steel tube of special construction, the inside being slightly convex. In it is fitted an expanding ring for securely holding the core whilst the rods are being drawn. There is nothing special in the construction of the core-tubes, E and D; they are made of steel, and threaded at the ends for attachment. One or more lengths of core-tubes may be used, but if only one be attached a core of great length cannot be extracted; it also renders necessary a more frequent change of the rods.

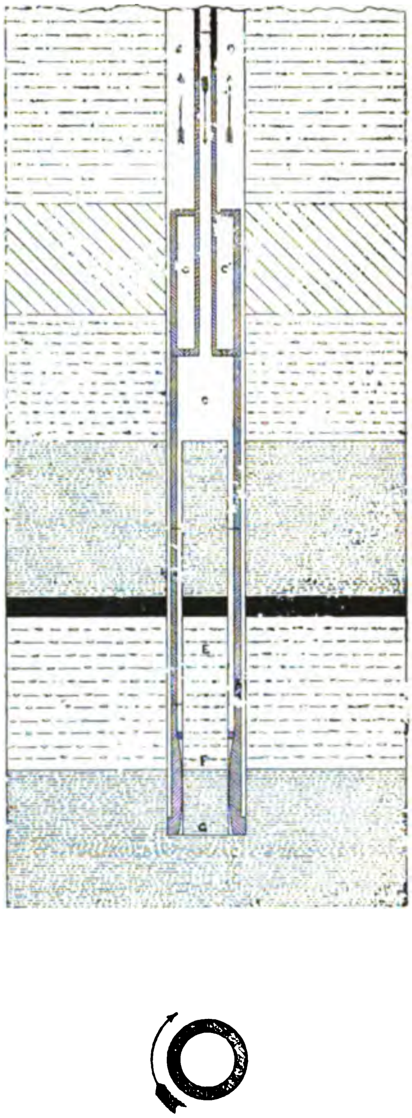


Fig. 15.—Method of Boring with the Diamond Drill.

In hard rock 20 feet is the length of core-tube adopted, and for this length the shear-legs must be 40 feet high.

The top of the core-tube is solid, but is provided with a threaded opening in the centre, into which are screwed the boring rods, A. Above the core-tube is the sediment tube, C, screwed on to it. Its object is to collect the sediment washed from under the crown as the boring proceeds. As shown in the drawing, the crown is slightly larger than the other portions of the boring tool, so as to reduce friction. The manner of procedure is as follows:—

The boring tool, with one length of rod attached, is lowered into the bore-hole by means of the chain over the shear-legs pulley, and is then suspended by clamps placed over the bore-hole until one or more lengths of rods (which consist of ordinary tubes threaded at the ends) are raised by means of the chain, and attached by screwing to the rods held by the clamps. All is then lowered together, and again suspended by clamps. This process is repeated again and again until the crown reaches the bottom of the bore-hole. The machine is then moved forward on the rails into its working position, the cross-head lowered, and the hollow shaft screwed on to the boring rods.

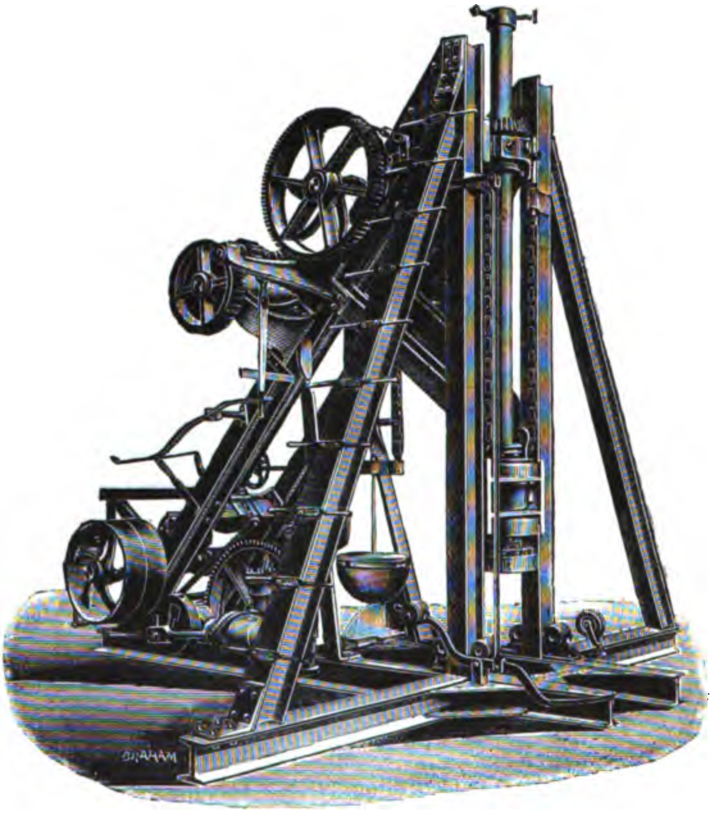


Fig. 16.—The Diamond Rock Drill.

The rods are then given a rapid rotary motion (from 200 to 300 revolutions per minute), and the motion of the rods being transmitted to the crown an annular channel is cut by the diamonds, and as the rods descend the core formed inside the channel enters the core-trap, and afterwards the core-tubes. As the rods are revolved the pump is set in motion, and water is forced down the hollow rods, A, as shown by the arrow, at a pressure which must be sufficient to cool the crown, and keep it clean by sweeping away the eroded material. After passing under the crown the water returns to the surface by the annular space formed between the walls of the bore-hole and the boring rods, as shown by the arrows.

Unless the ground is soft and friable the core formed is solid, and will not break till the act of withdrawing the rods takes place, when it parts at the base and is retained within the core-tube by the expanding ring. Upon reaching the surface these cores are all accurately numbered and safely housed by the attendant in charge, who also enters all notes as to stratifications between the numbers of cores not extracted, and is responsible to his employer for the preservation of an accurate account of the strata.

Fig. 16 shows Appleby and Beaumont's patent diamond drill, which is capable of boring holes up to 16 inches in diameter to depths of 2,000 feet, and of bringing up solid cores 30 feet long weighing more than 3 tons.

The framework of the drill is built of steel girders, and consists of a strong tripod mounted on horizontal girders, provided with all the appliances required to rotate the drill, to counter-weight the rods, and give a uniform pressure on the cutting-head.

The boring rods are made of drawn steel tubes with screwed flush joints, and are caused to rotate by being clipped in a universal chuck to a revolving quill, which has a stroke of 6 feet, and works in the vertical slides attached to the upright side-frames of the machines. The pump is connected with the top of the boring rods by means of a flexible tube. The weight of the rods is counterbalanced so that an even pressure is maintained on the boring crown.

The crown is screwed on to the core tube, and this latter for hard rock has a length of about 20 feet. The core tube is of a slightly less exterior diameter than that of the crown, and may be of much greater diameter than that of the boring rods which are screwed into it. The boring rods are of course hollow, and, when at work, a constant current of water is forced down them in order to keep the crown cool and clear away the borings, which are carried up to the surface as fine sand.

When a certain depth has been bored, varying according to the strata and the length of the core tube, the core clip is used for detaching the solid portion left standing within the annular groove cut by the crown, and the core is then raised to the surface.

The following is a complete specification of the outfit necessary for a machine ready for work, for a boring 2,000 feet deep:—

Boring machine, as shown in the illustration (about 395*l*.).
Portable engine of 10 horse power.
2 engine straps.
6 1¼-inch flexible hoses.
3 pairs of unions.
2 water unions.
150 feet of ¾-inch chain.
1 18-inch top sheave, with spindle and bearings.
2,000 feet of steel boring rods and joints.
300 feet of 6-inch steel lining tubes, with steel joints.
400 „ 5 „ „ „ „
600 „ 4 „ „ „ „
800 „ 3 „ „ „ „
Special connectors—6-in. to 5-in., 5-in. to 4-in., 4-in. to 3-in.
6-inch to 3-inch steel driving shoes.
2 15-foot core tubes.
2 special connectors of boring rods to core tubes.
2 „ „ of rods to 3-inch tubes.

2 6-inch boring crowns
2 5 ,, ,, ,,
2 4 ,, ,, ,, unset.
2 3 ,, ,, ,,
4 2 ,, ,, ,,
4 core clips.
2 6-ton hydraulic jacks.
1 3-ton set of blocks and falls.
2 lifting swivels for rods.
1 set of eccentric clips.
4 patent rod tongs.
2 ,, tongs for 3-inch tubes.
2 ,, ,, 4 ,, ,,
2 ,, ,, 5 ,, ,,
2 ,, ,, 6 ,, ,,
1 complete set of diamond setting tools in lock-up box.
1 5-inch parallel vice.
1 complete set of spanners.
2 patent shifting wrenches.
1 1-gallon oil-can and 2 oil-feeders.
1 set of stoking tools.
1 set of spare valves for the pumps.
1 spare set of pinions in steel.
About 200 carats of carbonates or black diamonds.

This complete machine, engine and outfit, will cost about 1,400*l.*, exclusive of the carbonates, which vary considerably in value, and can only be quoted for from week to week. The price is about 42*s.* per carat.

In boring through surface clays or other soft material, the ordinary method of boring compares favourably with the diamond drill, but in hard strata and for depths exceeding 100 yards, the diamond drill works to great advantage, being more expeditious and not more costly.

The bore-holes may be wholly or partly lined with tubes to prevent the lateral pressure choking the hole, and also to prevent water entering from the strata at the side, which would otherwise represent a certain amount of pressure to be overcome in maintaining the water-flow pressure down the boring rods. The lining tubes in some instances are quite as necessary in some sections of dry borings, in order to prevent the feed-water escaping through porous strata frequently found near the surface and always adding to the power required to maintain a given pressure of water. Occasionally a bore-hole is commenced by the ordinary method, and on reaching a certain point where harder strata are found, it gives way to the diamond drill. Again, a bore-hole is sometimes undertaken by the diamond drill from the bottom of an existing shaft to prove the lower strata there. In some instances borings have been put down by the diamond drill from the bottom of heavily watered sinking pits to the coal workings below, after which the sinkers were enabled to proceed in a dry shaft to complete the sinking. The diamond drill cuts the hardest rock at rates varying from 2 inches to 8 inches per minute.

In bore-holes of small size, unless the seams of coal passed through are extremely hard, the action of the drill grinds them and the specimens appear in the core-box in a pounded condition. If the bore-hole is of somewhat larger size, this will not be so noticeable, and if the specimen of the coal seam is protected by hard rock over it in the core-box as shown in the drawing, a very perfect specimen of the seam may reach the surface. Where, however, the coal is of a soft nature and it forms the top portion of the core entering the core-box, the weight of water in its descent will wash a considerable portion of the coal away, and this may lead to an erroneous section of it. A valve has been placed on top of the core-tube with a view to take the weight of water off the whole core, and possibly some of the Boring Companies adopt this practice now.

The diamond used is in a different state from the gem, and is technically called *carbon*, *carbonate*, or *carbonado*, but the substance is really carbon in an imperfect state of crystallisation, and this rough diamond, while as hard as the ordinary diamond, does not so easily break. It is black or dark grey in colour, and granular in texture; it is supplied only from the mines of Brazil in the district of Bahia, where it was discovered in 1842; and no diamonds of the same class have been found elsewhere, except, it is said, in Borneo. Experiments have been tried with other substances, such as bort, or rough splintery diamond, but none compare with the carbonate in fitness to cut hard rocks. A piece of carbonate the size of a large pea, will cut a hole in sandstone, half a mile deep or more, without sensible abrasion to its surface, and although in harder rocks than sandstone the abrasion is more, it is still very slight. The loss on the crown from the act of drilling half a mile is very slight, but the diamonds get broken from other causes. If a jar breaks one, it causes more jarring and leads to the breakage of others, and as the diamonds are costly, the loss to the Boring Company becomes serious.

A great advantage of the diamond-drill boring is that the hole is kept true and vertical, and this cannot be assured in other systems. Then the cores frequently contain whole and uninjured fossils characteristic of the strata, which but for this evidence would remain uncertain of classification. Thus the sub-committee of geologists connected with the Sub-Wealden bore-hole near Hastings, state that the determination at which they arrived with reference to the position of the strata was "due to the manner in which large cores were brought to the surface by the Diamond Rock-Boring Machine, so that a number of fossils were obtained entire, the species of which could be accurately determined."

The percentage of cores obtained by the diamond drill seems to vary from about 60 to 90 in bore-holes of small size. The boring rods are subject to accident, occasioning delays, such as breakages and jams, but they are not more difficult to deal with than are the breakages inseparable from other modes of boring. When a trial of the diamond was first made to ascertain its suitability for the purpose of boring, it was given a percussive motion, but experience soon showed it to be better adapted for abrasion.

On the introduction of the system the holes drilled were under 2 inches in diameter, and the cores produced were about ¾ of an inch in diameter. Consequently difficulty arose in securing cores in soft strata, although the result in hard rocks was quite satisfactory. On this account and also to give room for lining tubes, the bore-holes have been gradually increased in size. Where they have been of large size, 100 per cent. of cores of the strata have been obtained even with the softest of rocks. For instance at Caerphilly, in South Wales, in a hole put down to a depth of 1,007½ feet in 1874, the cores yielded showed a complete section of the strata passed through, and the samples of coal were satisfactory.

Bore-holes of 26 inches in diameter yielding cores 23½ inches in diameter have been put down by the diamond drill.

A Diamond Rock Boring Co., prospecting for minerals, usually stipulates that the employer must find water and engine power, or, instead of the latter, pay a fixed sum per month, according to the size of the engine, but the company find everything else that is necessary. Boring to any specified depth is not guaranteed, but the company uses its best efforts to reach it. Unless a certain prescribed depth be bored the employer must pay the cost of carriage to and from the boring site or sites, and all damage caused in moving the machinery on and off the ground.

The schedule of prices is in arithmetical progression over certain zones of depth, and only apply to strata of ordinary character.

One of the deepest bore-holes known is that made at Schladebach, Leipzic, to a depth of 5,736 feet. It was commenced 11 inches in diameter in the Trias, and after passing through the Permian entered the Old Red Sandstone, the size of the hole having been reduced to 1·22 inch at the bottom.

The following very interesting particulars of this borehole appear in the *Colliery Guardian* of April 22nd, 1892.

"At the closing *séance* in 1891 of the Société Industrielle de Mulhouse, M. Charles Zundel gave some particulars, gathered from two reports sent to the society by the German Mine Department, of the deepest borehole which has yet been put down—viz., that at Schladebach, near Kötschau in Merseburg, Prussia, which attained the great depth of 1,748 metres, or 956 fathoms.

"The boring was begun in August, 1880, by the Royal Division of Prussian Mines, and finished in the autumn of 1886, having occupied 1,247 actual working days, with a mean daily advance of 1·4 m. (4 feet 7 inches), and at a total cost of 212,304 marks (£10,615), or say, £6 per yard of boring. A 25-horse power portable engine sufficed for the motive power necessary to work the boring rods and also the pumps, a tower 27 m. (89 feet) high, permitting of 20 metres of tube lining being raised or lowered at a time.

"The initial diameter of the hole was 280 mm. (11 inches). After 20 m. of sand, gravel, and marl, passed through by the system of *Schappenbohrung*, or tube boring, the variegated sandstone was struck, which was attacked by the *Hohlfreifallinstrument*, or free-fall apparatus, a cast-steel drill with end in the form of a cross with guides, water flowing down outside the tubular rods and rising up the inside. At the depth of 57 m. (31 fathoms), the lining could not be got to descend further, so an attempt was made to go on without lining; but the *débris* detached from the sides of the hole rendered it necessary to put in a tube lining of 230 mm. (9 inches) diameter. At 164 m. (90 fathoms), strata of gypsum and anhydrite were encountered, and also water charged with sea salt. At the depth of 175 m. (95 fathoms), the free-fall apparatus was abandoned for rotary boring by means of a soft iron crown, of 210 mm. (8¼ inches) diameter, set with borts or carbonate diamonds, producing cylindrical cores of 140 mm. or nearly 6 inches diameter. In the dolomitic limestone, it was found impossible to detach one of these cores; and the crown became fixed so fast that it became necessary to draw up the rods and bore into the core with a small crown, in order to detach it, a work that occupied three weeks.

"After several accidents and difficulties—such as the rods bending and having to be cut and lifted, a crown sticking in the bottom of the hole and having to be perforated and ground up by the free-fall apparatus, or a tool falling by accident into the hole and necessitating the cutting of the rods at great depths—the depth of 1,070 m. (585 fathoms) was reached, the diameter of the hole having been reduced seven times. Between this depth and that of 1,724 m. (943 fathoms), the hole was continued with a diameter of 48 mm. (2 inches), yielding cores of 23 mm. or less than 1 inch diameter. The Old Red Sandstone had been passed through between 327 and 1,630 m., when the Devonian rocks were traversed, without, however, encountering a seam of coal, which might have been met with at that geological depth. A stratum of cuprose schist, struck at 326 m., was not considered rich enough to warrant its being worked.

"Nevertheless, the hole was continued with a purely scientific object, and at the beginning of 1886, a lighter tubing of 33 mm. (1⅜ inches) diameter, was put in, in which boring was carried on with a crown of 31 mm. (1¼ inches) giving cores of 12 mm. (½ inch bare) only. But finally, after successive accidents—such as breakage of the rod and fall of the crown, and breakage of the screw of the core-extracting apparatus—at the depth above mentioned it was found necessary to stop the work, which could only have been continued at a great expenditure of time and money. The following are the rocks passed through, with their depths and thicknesses:—

	Thickness Metres.	Depth Metres.
Vegetable earth	0·6	0·6
Sand	4·27	4·87
Clay	17·76	22·63
Variegated sandstone	141·89	164·52
Gypsum and anhydrite	16·03	180·55
Permian limestone (Zechstein)	46·36	226·91
Gypsum	10·41	237·32
Anhydrite	89·19	326·51
Cuprose schist	0·9	327·41
Old red sandstone	1,302·59	1,630·
Devonian rocks	118·4	1,748·4
	1,748·4	

"Careful thermometric observations were made as the work proceeded, and naturally delayed its progress considerably. From a depth of 1,200 m. they were continued regularly every 30 m., and before the tube lining was put in, so as to avoid any disturbing influence from conduction, while, to counteract that of water-currents, the thermometer was immersed in a fixed column of water between two clay plugs, and left there for sixteen hours. To prevent the glass from being broken by the great pressure, it was enclosed in an iron case; and there were always three superposed thermometers from which a mean of the readings was taken. The observations of the first 1,200 metres were taken subsequently. The observations went to show that the increase of temperature does not diminish with depth, as had before been imagined, but follows a constant arithmetical progression. The last reading taken, viz., at 1,716 m. (938 fathoms) was 45·3° Réaumur, or 56·6° Cent. or 134° F., which shows an increase of 1° Réaumur for every 46·09 m. equivalent to 1° Cent. for every 36·87 m. At this rate potassium, the fusion point of which is 48° Réaumur = 60° Cent. = 140° F., would melt at a depth of 1,845 m. (1,009 fathoms), and grey foundry pig (1.240° Réaumur = 1,550 Cent.) at 56,775 m., or about thirty-five miles, while the greatest temperature of blast furnaces, which, according to Schérer is 2,230° Réaumur (4,037° Cent.) would be attained at a depth of 104,708 m., or a little over fourteen geographical miles.

"M. Zundel concluded his interesting communication by calling upon the Société Industrielle to urge the putting down of a deep borehole in Alsace-Lorraine, or the deepening of two Hasenrain and Dollfus holes in order to prove the existence of coal or bituminous shales yielding petroleum, or even to demonstrate practically, that a source of heat and power exists beneath our feet, that shall maintain warmth in our bodies, and keep our mills and factories going, when all the coal deposits shall be exhausted."

The deepest boring is that made by the Prussian Government at Paruschowitz, near Rybnik, in Upper Silesia, particulars of which were communicated to the Société Industrielle de Mulhouse, by M. Charles Zundel. This borehole, begun on January 26, 1892, was finished on May 17, 1893, after having attained a depth of 6,572·7 feet, or 2,003·34 metres. The surface level of this boring being 498·7 feet higher than that of Schladebach, it only penetrated 338 feet nearer the centre of the earth than the Schladebach boring. The Paruschowitz borehole proved the existence of eighty-three coal-seams, many of them being of considerable thickness, and giving a total thickness of 89·5 m., or 293·6 feet. Begun with a diameter of 12 inches, and lined with a tube $\frac{13}{32}$ inch thick, the boring was put down to a depth of 230 feet, from which point to 351 feet the diameter was reduced to 10 inches. At this depth the blue marls encountered became so compact as to necessitate the use of the diamond drill for the further boring. Under the action of the water injected into the hole, the marl swelled and subjected the lining tubes to such compression that it was found necessary to gradually reduce their diameter. Shifting sand, met with at the depth of about 656 feet, also caused great difficulty.

The greatest difficulty of all, however, was the great weight of the boring-rods as the depth increased. Although a reduction in the weight of these was obtained by substituting steel for iron, yet at the depth of 6,560 feet the total weight of the tools was 13·7 tons. With such a weight and so great a length, ruptures of the rods were frequent, and an accident of this nature finally stopped the work. When the diameter was $2\frac{23}{32}$ inches, and that of the cores brought up $1\frac{25}{32}$ inches, about 4,500 feet of rods fell to the bottom, and became jammed in an unlined portion under the tubing, so that it was impossible to withdraw them. Consequently the borehole was abandoned. During the 399 working days a daily advance of a little over 16 feet was made. The total cost of the boring was £3,761, or 34s. 4d. per yard. Temperature observations made showed 12·1° Cent., or 54° Fahr. at the surface, and at a depth of 6,572·7 feet the temperature reached 69·3° Cent. or 157° Fahr. This gives an average increase of 1° Cent. for 34·14 m., or 1° Fahr. for 63 feet, differing from that observed at Schladebach, where the mean increase was 1° Cent. for a depth of 35·45 m., or 1° Fahr. for every 64 feet 7 inches.

During thirteen years the Prussian Government have made 400 boreholes for exploring purposes, of an aggregate depth of 80 miles, and at a cost of £650,000.

A bore-hole at Sperenberg, Berlin, was made by means of rigid rods to a depth of 4,170 feet, and took $4\frac{3}{4}$ years to accomplish. A boring for salt near Lubtheon, in Mecklenburg, was carried out by a diamond drill to nearly 4,000 feet, and was completed within 6 months. The boring was not only wonderfully successful in the speed with which it was accomplished, but was further distinguished by the hole yielding 100 per cent. of cores, one specimen of rock-salt being over 20 feet long.

No virgin property of a size to require any borings on it should have less than three, and it may be wise to put down more; but in a neighbourhood free from faults and dykes, and proved by winnings all round, none will be necessary. It is natural to let the first be nearest the rise. It must be remembered that the depths as proved by boring through inclined strata do not give the true thicknesses of such, as the bore-hole is perpendicular and the line of stratification does not often form a right angle to it. The true thickness of a stratum is found by multiplying the thickness as proved in the boring by the cosine of angle of dip. Thus, if a stratum was proved to be 1 fathom thick in the boring where the dip was at an angle of 30°, the true thickness would be ·866025 of a fathom or 5·19615 feet. Tables of incline measure may be obtained showing the comparative lengths of the hypothenuse, horizontal, and vertical legs of a right-angled triangle for every degree of the quadrant.

Supposing three bore-holes to be put down on a property at equal distances apart, say 500 yards. No. 1 being nearest the rise and proving the coal at a depth of 15 fathoms, No. 2 fully to the dip of No. 1, as well as can be ascertained, proving the same seam at a depth of 30 fathoms, and No. 3 still fully to the dip of No. 2 as nearly as may be judged, proving the same seam at a depth of 50 fathoms, all the bore-holes being at the same surface level. It would appear

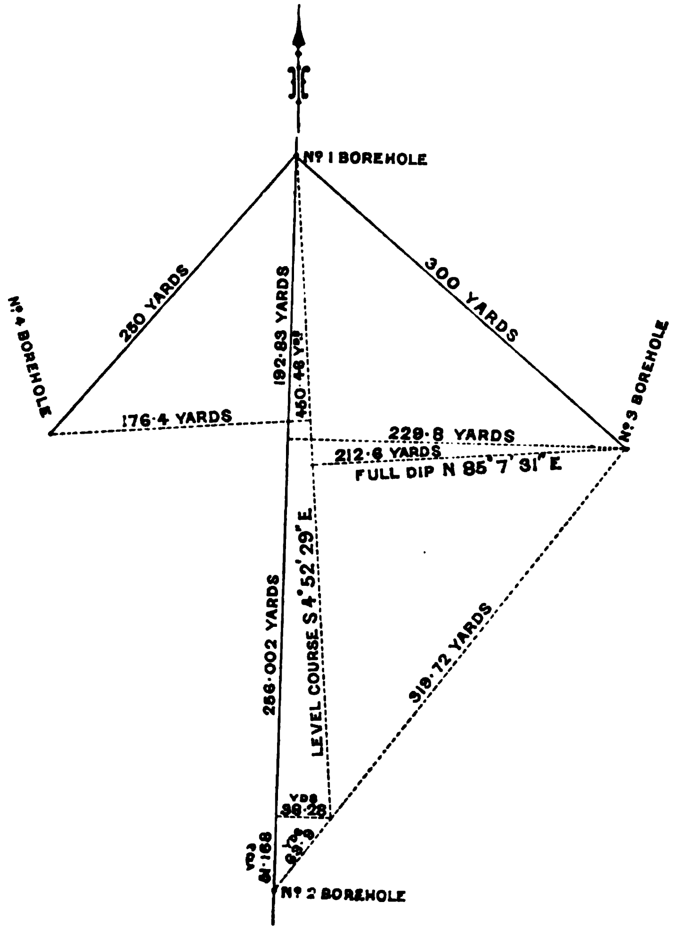


Fig. 17.—Plan showing Position of Bore-holes. Scale 6 chains to 1 inch.

that the dip of the strata from No. 1 to No. 2 bore-hole was 500 ÷ 60 yards − 30 yards $= \frac{500}{30} = 16{\cdot}\dot{6}$, that is, 1 in 16·$\dot{6}$, and from No. 2 to No. 3, 500 ÷ 100 − 60 $= \frac{500}{40} = 12{\cdot}5$, that is, 1 in 12·5, supposing there were no faults or dislocations in the strata to upset the calculation.

Suppose now 4 bore-holes are put down on a property, and there are no other means of forming any idea of the dip. No. 1 proves the coal at a depth of 85 fathoms, No. 2 is due South of No. 1, by the magnetic needle, and proves the coal at 80 fathoms, being 500 yards from No. 1. No. 3 lies S. 50 E. of No. 1, and is 300 yards distant, proving the coal at a depth of 110 fathoms.

No. 4 is S. 40 W. of No. 1, 250 yards distant from No. 1, and proves the coal at 64¼ fathoms, all the bore-holes being at the same surface level. The direction of full dip and its extent may be determined as follow.

By plotting as shown in Fig. 17, or by trigonometry, it will be found that the distance between No. 2 and No. 3 would be 383·62 yards. The difference in level between the seam at No. 2 and No. 3 bore-holes is 220 yards — 160 = 60 yards and $\frac{383\cdot62}{60} = 6\cdot39$, that is the seam dips from No. 2 towards No. 3 at the rate of 1 in 6·39. Now, to find the level course of the seam, follow along the course from No. 2 towards No. 3 until a point is reached 5 fathoms or 10 yards below the seam at No. 2 bore-hole, because the difference in the depth between No. 1 and No. 2 is 85 — 80 = 5 fathoms. To gain 10 yards of fall along a course dipping 1 in 6·39 follow it for a distance of 6·39 × 10 = 63·9 yards. Therefore a line drawn from No. 1 bore-hole to a point in the line connecting No. 2 and No. 3, and 63·9 yards distant from No. 2, gives the level course of the seam, and by means of plotting or from the computed value of the angles, it may be seen that the bearing from Nos. 2 to 3 bore-holes is N. 36° 48′ 9″ E., and that of the level course of the seam S. 4° 52′ 29″ E. from No. 1. The full dip would be at right angles to this or N. 85° 7′ 31″ E. To get the amount of dip, draw a line connecting No. 3 bore-hole with the level course of the seam and at right angles to it. Such a line would measure 212·6 yards, and since the difference of level in the seam between the two ends of it is 110 fathoms — 85 = 25 fathoms, or 50 yards, we have $\frac{212\cdot6}{50} = 4\cdot25$, so that the full dip would be 1 in 4·25 in a direction N. 85° 7′ 31″ E.

In the same way it can be shown that a line drawn from No. 4 bore-hole to join the line representing the level course of the seam and at right angles to it would measure 176·4, and taking the dip at 1 in 4·25 as proved on the other side of the level course $\frac{176\cdot4}{4\cdot25} = 41\cdot5$ yards or 20·75 fathoms, and 85 — 20·75 = 46·25 fathoms as the depth of No. 4 bore-hole to prove the seam, which is the depth given, a reasonable conclusion from which would be that the coal on the property was fairly free from faults. It is only right to say that these calculations are often upset by faults running between the position of the bore-holes, and when these are suspected to be on the property, more bore-holes should be put down before the sinking is decided on.

In coalfields having carboniferous rocks exposed on the surface the outline of the basin may be traced by following the course of outcrop of the upper edge of millstone grit or other strata underlying the carboniferous, if conformable to them. Any estate being within the coal measure or productive area as shown on a geological map would usually be expected to comprise coal seams. If situated to the rise of the lowest seam's outcrop, no coal would be available, although the rocks might belong to the lower members of the carboniferous period.

Where a coalfield is known to exist over a certain extent of country by its coal-bearing portion of the carboniferous rocks outcropping in some places and disappearing in others under Permian or newer rocks, which lie unconformably on them, there is much room for speculation as to the extent of the coalfield. Under such circumstances the search for coal of course becomes far more intricate.

As an instance of the difficulties of such a search may be cited that of the South Staffordshire Coalfield under the "red rocks."* A fault which exists in the neighbourhood of West Bromwich appeared to cut off the coal measures which are exposed to the westward. To the east of this fault the coal measures are covered by "red rocks." About 1839 the Earl of Dartmouth sank the Heath Pits ¾ of a mile or thereabouts to the east of the boundary fault, and these pits passed through Lower Permian rocks overlaying the coal measures and proved thin seams of coal, thus showing that a further productive area existed of unknown length, but ¾ of a mile in width. As the Heath Pits did not find the Staffordshire thick coal, a "heading" was driven eastward, and a boring upwards from the end of the heading proved "red rock," while another downwards from the same point struck a hard rock, afterwards believed to be Silurian. After these failures to find the thick coal, one of the thin seams was followed to the west, towards the old or previously known portion of the coal-field, and this led into the thick coal.

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* See *Colliery Guardian*, May 14th, 1875.

Sir Roderick Murchison was of opinion that the shaft had been sunk upon a line of dislocation, the continuation of the upcast of Silurian rocks of Walsall and Thame Bridge.

Professor Jukes thought there was a sudden rise of Silurian rocks through the coal measures, forming a bank, and that this bank had been favourable to the formation of sandstone and the accumulation of clay, but unfavourable to the formation of coal. Silurian shale having been found in a pit at Langley Mill, Oldbury, Professor Jukes assumed the bank to be continuous for that distance —about three miles.

While opinions differed as to what lay to the east of this bank, none were favourable to the existence of workable coal seams.

Sinkings as indicated below in the newly-found tract of productive coal measures were stated to have proved various thicknesses of red rock (Permian) resting on various thicknesses of coal measures believed to have been denuded.

	Permian over coal measures.	Coal measures over thick coal.
Lewisham Pits	315 feet	520 feet.
Lyng Colliery	550 "	350 "
Heath Pits	806 "	40 "
Bullock's Farm Pits	700 "	330 "
Unitt's Boring at the "Ruck of Stones"	664 "	Abandoned.

It was thought that these sinkings indicated a thickening of Permian beds to the eastward, while the coal measures below would unconformably be still further denuded. Professor Jukes considered it probable that a little further to the east of the Heath Pits, the coal measures would be entirely wanting, and the Permian rest directly on the shale of the Silurian formation.

The boring at the "Ruck of Stones" lies more than a mile east of Bullock's Farm Pits; and the fact that the strata between them have a dip eastward of about 10 degrees, shows that the 700 feet of Permian at Bullock's Farm are wholly below the 664 feet proved at the "Ruck of Stones." From this Professor Jukes estimated that in the neighbourhood of West Bromwich there must be at least a total thickness of 1,500 feet of Permian rocks.

On the assumption that the coal measures had not been subject to denudation and that the rocks to the eastward of the Silurian bank retained a normal thickness, it was argued that in any sinking for thick coal 1,500 feet of Permian and 1,000 feet of upper coal measures might be expected before reaching it.

For many years these opinions of geological authorities had the effect of preventing trials on the doubtful ground, but at last the Sandwell Park Colliery Company, under the direction of Mr. Henry Johnson, made the venture, and as a reward for their enterprise struck the thick coal at 418 yards from the surface about the beginning of 1875.

The red rocks at Bullock's Farm and other West Bromwich sinkings were proved by the Sandwell Park sinkings to be coal measures instead of Permian as previously supposed. The Sandwell Pit is a mile to the eastward of Bullock's Farm. As fixing the position in the pit where the coal measures were struck, fossil plants found at a depth of 110 yards in red measures were stated by authorities on the subject, to be Permian. At a depth of 200 yards a thin seam of coal 7 inches thick was proved. Above the coal was a black shale containing fossil plants, some of which were identical with those found at 110 yards, whilst beneath the coal was a bed of fire-clay containing *Stigmaria*. At 230 yards a second coal seam 6 inches thick was struck, and a third of the same thickness at a depth of 244 yards, while at 418 yards the thick coal, the object of this patient and enterprising search, was attained. The shaft therefore entered the coal measures at a depth of 110 yards or less, showing that there is a greater thickness of coal measures over the thick coal at Sandwell Park Colliery than was proved by the pits previously sunk through the Permian strata.

As another instance of a successful search for coal we may mention that near Dover, in Kent. This discovery is full of interest to the geologist, and as it is within a workable depth it may create a very important industry in the South of England.

Previously, no true coal had been found in England to the south of a line joining Bath and Stamford and continued to Great Yarmouth, though lignites were known to exist in the Wealden strata of Kent, Surrey, and Sussex. An examination of the formations (which are much newer than the carboniferous) prevailing over most of the area indicated would of itself not appear hopeful, yet geologists have reasoned for many years that there was a possibility of finding coal under these newer formations and within a reasonable distance of the surface.

About the year 1855 Mr. Godwin-Austen started this theory, but Sir R. Murchison disputed it. In 1871 the Royal Coal Commission published its Report, including a very elaborate communication from Professor Joseph Prestwich, who had accepted the theory suggested by Mr. Godwin-Austen.

The main feature which gave rise to the theory, is the fact that a great axis of elevation extends from the South of Ireland to Westphalia, a distance of 850 miles. Its existence can be traced at one extremity from Ireland to Frome in Somersetshire. Along the strike the lateral pressure elevated the fractured ends of the strata into ranges of hills, of which there now remain two remnants, viz., the Mendips, in Somersetshire, and the Ardennes, in Belgium. The Ardennes have many features in common with the Mendips. On the northern flanks of both ranges the palæozoic strata are highly inclined as if tilted up from the same disturbing cause, and both are overlaid by newer strata reposing horizontally on them. From Westphalia to the North of France there is a series of coalfields, the more important being those of the Ruhr, Aix-la-Chapelle, Liége and that of Charleroi, Mons, and Valenciennes, whose longer axes succeed one another along the same line of strike. In all these the coal measures are highly inclined on the south against the Mountain Limestone, while on the north they disappear under newer formations. Westward of Valenciennes no palæozoic strata are exposed on the surface, but the coal measures have been proved to pass beneath the chalk and tertiary strata to Enquin within 30 miles of Calais, whilst further west the older rocks subtend the chalk. A boring at Calais, however, proved carboniferous strata at a depth of 1,032 feet from the surface after passing through the chalk.

Passing to a point near the other extremity of the same axis of elevation, a striking resemblance in the geological features is seen. From Milford Haven to Tenby are found old red sandstone and mountain limestone in a contorted condition, resting on which to the northwards is the Pembrokeshire coal-field, considerably disturbed, though not to the same extent as the old red sandstone and mountain limestone; from which it may be inferred that the forces disturbing the strata decreased in proportion to their distance from the line of upheaval. Proceeding eastward along the axis of elevation to the Somersetshire coal-field, the Mendip Hills give evidence of the same tilting and denudation of the older rocks observable in those of the Ardennes range, and on their northern flank are covered by mesozoic strata laid horizontally over the highly inclined coal measures and mountain limestone.

The South Wales coal-field is not hidden by newer strata, but a very large portion of the Somersetshire coal measures is covered by permian, lias, and oolite, and proceeding eastwards these in turn are overlaid by the chalk. The Pennant rock, yielding but few seams of coal, and the associated sandstones, shales, and coal seams above, and also below, with the addition of ironstone, form strong features of resemblance between the coal-field of South Wales and that of Somersetshire.

The similarity in the structure of the rocks of the Mendips and Ardennes, and in the direction of their strike, points to their being due to a common cause, namely, an upheaval of irresistible might and affecting an enormous mass of the earth's crust. The upheaval was too powerful for any opposing force to interrupt, and its continuity under the newer rocks which hide it from view in the South of England, is almost a matter of certainty.

The mountain limestone is continuous throughout the line of elevation, and wherever the older rocks come to the surface, the coal measures show signs of having covered the mountain limestone at the time of disturbance and to have accompanied the latter in its foldings and movements. This fact, taken in conjunction with the unconformability of the permian, enables geologists to fix the age of the axis of elevation, as being after the coal measures and before the permian. The strata of the coal measures were altered from their originally horizontal position by subterranean forces, causing the line of elevation. As the latter proceeded in a slightly irregular, wave-like direction, mostly east and west, the rocks were doubled up or folded into a number of anticlinal and synclinal curves proceeding northwards from the main line of disturbance. The effect of this would be to change what was once a gigantic coal-field into a number of isolated basins separated by ridges of the older rocks, similar to those known to exist. A proof of the series of existing coal-fields having once formed part of a larger, lies in the fact that their edges contain beds of equal thickness with the more central portions. Were they separate coal-fields there would be a thinning out of beds near old lines of shore.

If this reasoning be accurate, then the amount of coal which will be found underlying the newer formations in the South of England will depend entirely on the amount of denudation the coal measures have been subjected to over that area anterior to the deposit of the newer strata. The borings hitherto made point to a southern slope of the surface of the palæozoic rocks in the south of England; the denuded surface can of course give no indication of the dip in the rocks beneath.

Professor Hull shows in a section from Gloucestershire to Oxford that all the newer rocks below the great oolite thin out rapidly to the S. E., the total thickness of the overlying rocks diminishing from 1,880 feet in Gloucestershire to about 600 feet at Oxford. At Burford, in Oxfordshire, coal-measures have been reached at a depth of 1,184 feet.

In 1854–5, a boring at Kentish Town, on the north side of London, passed through tertiary and secondary strata and proved red sandstones, believed by some geologists to be of palæozoic age, at a depth of 1,114 feet. In 1854–7, a boring at Harwich proved lower carboniferous rocks at a depth of 1,026 feet, after passing through tertiary strata and the chalk and gault, the lower greensand, the oolites and the lias being absent. These borings presented the same order of superposition as that at Calais and in other places in the North of France and Belgium.

The borings alluded to were made for water, but in 1872–5, to test the theory of Mr. Godwin-Austen and Prof. Prestwich, a boring was made near Battle, in Sussex, under the direction of the Sub-Wealden Exploration Committee. It passed through 200 feet of Purbeck strata and 1,705 of oolite below, the bottom of which was Oxford clay, and was then abandoned. It proved that the palæozoic rocks did not lie within 1,905 feet of the surface at Battle. In 1886, Sir Edward Watkin, chairman of the Channel Tunnel Company, acting on Professor Boyd-Dawkins' report, which recommended that a bore-hole be made in the neighbourhood of Dover (at the Channel Tunnel works, under the Shakespeare Cliff), gave orders to commence the boring. The site was about 30 miles west of the boring near Calais, which had established the fact that coal measures extended under the newer strata to the French coast line at a depth of 1,092 feet from the surface. After penetrating 500 feet of cretaceous rocks the boring passed through 613 feet of oolitic rocks, when the coal measures were reached, being therefore 1,113 feet below high-water mark and 1,157 feet from the surface. The boring, after passing only 27 feet further, met with a good seam of coal on the 15th February, 1890. This result, whilst very encouraging to all concerned, has by no means cleared up all the points connected with the hypothesis started by Mr. Godwin-Austen in 1855, but it has clearly established the fact that at any rate one coal-field exists in south-eastern England, where it is covered by the newer strata, and that within a workable depth. This alone is no slight reward for the patient toil and thought of the geologist. It will doubtless act as an incentive to other trials along the most likely course of the line of elevation crossing the south of England, between Kent and Somerset, in order to search for other coal-fields or to discover the detached portions of what at one time was one immense coal-field, and is believed now to exist in a more or less denuded state. Indeed, a company was registered on March the 27th, 1890, with a capital of £2,000 in £1 shares, to search for coal and other minerals in Kent, Surrey, Sussex, and elsewhere in Great Britain.

In this country, however, there is little encouragement offered to private enterprise in searches for coal of the importance attached to the search instituted at Dover. If successful, it is the landowners in the immediate vicinity who derive the greatest benefit, whether they promoted the search or contributed towards its cost or not. They may make what use they like of the information obtained (without reference to the bold spirits who pioneered the venture or those who carried it through), whether by fixing a high royalty price, or by sinking shafts themselves to work the coal so found. In France, where the minerals belong to the State, private enterprise meets with the encouragement which is due to it, and many agricultural districts have through this means been converted into busy mining centres.

When the law relating to royalties has been revised, searches of national importance may be expected to proceed more rapidly in this country, but all things considered, it is perhaps not surprising that so many years should elapse between the expression of Mr. Godwin-Austen's theory and the result to which it has eventually led.

The boring for coal near Dover has been continued to a greater depth since the above remarks were written. In a paper read before the Manchester Geological Society on February 2, 1894, Prof. Boyd Dawkins gave details of the Dover boring up to that date. It appears that twelve seams of coal, having an aggregate thickness of 23ft. 5ins., have been proved. The first was a seam 3ft.

6ins. thick, struck at a depth of 1,136ft. 6ins. below high-water mark, and described as "bright and bituminous, with sandstone parting," yielding 2ft. 6ins. of good house-coal. At a depth of 1,200 feet, the second seam occurred, but this was only 6ins. thick. Then at 1,229 feet occurred a 2ft. seam, and another of the same thickness at 1,279 feet, both described as "blazing, with specks of iron-pyrites." At 1,311ft. 9ins. the borer touched a seam of bituminous coal, 1ft. 3ins. thick; at 1,433 another seam, 1ft. thick; and at 1,456ft. a 2ft. 6ins. seam of "good house-coal." The eighth seam, described as clean bright coking coal, was touched at 1,570ft., and found to be 2ft. 3ins. thick; the ninth seam, 2ft. 9ins. thick, also clean bright coal, occurred at 1,763ft. 9ins.; the tenth seam, of "clean, bright coal," 1ft. 8ins. thick, was struck at 1,831ft.; the eleventh, a 1ft. seam of hard coal; and, finally, at a depth of 2,177ft. 6ins., a seam of "bituminous coal" was discovered, having a thickness of 4ft.

Having regard to the opinion of Mr. McMurtrie, of Radstock, and of M. Watteyne, a Belgian engineer, Prof. Boyd Dawkins concludes that "the horizon of the south-eastern coalfield is that of the valuable upper measures of Somersetshire, and of the important middle coal-measures of Belgium and Northern France." At the same time he points out that the beds under Dover have an exceptionally gentle dip. It is notable that in the course of the Dover boring a bed of oolitic iron-stone, 12 feet thick, was discovered, and it is probable that this ore, worked in connexion with the underlying coal, may become of great industrial value. A shaft has been sunk 44 feet from the surface to high-water mark, and the boring is continued from the bottom of the shaft.

It will be useful to compare with the results at Dover the number and thickness of seams, and the thickness of coal measures in the French, Belgian, and Westphalian coal-fields as given by Professor Joseph Prestwich in the following table:—

	WEST OF DOVER.		EAST OF DOVER.		
	South Wales.	Somerset.	Mons.	Liège.	Westphalia.
Number of seams	75	55	110	85	117
Total thickness of workable coal	FEET. 120	FEET. 98	FEET. 230	FEET. 212	FEET. 294
Mean thickness of coal measures	FEET. 11,000	FEET. 8,400	FEET. 9,400	FEET. 7,600	FEET. 7,218

The English coal measures appear to contain fewer seams than the Continental, although they are of considerably greater thickness. This is partly due to the Pennant rock in the Somerset and South Wales coal-fields, where it is from 2,000 to 3,000 feet thick and contains but little coal. In Belgium the Pennant rock is replaced by productive measures. It frequently happens that there are marked changes in the quality and thickness of the coal seams in the same coal-field, as well as in the associated strata. In districts where these changes defy all attempts at co-relation, the seams are often known by different names at different points in the same coal-field.

The difference shown in the particulars of the coal-fields in the above table, does not therefore prevent our associating the Dover field with them, or exclude the probability that it was at one time continuous with them. This probability, indeed, is strengthened by the coal already proved in the boring, the thin seams of which correspond with those of the North of France, Belgium, and Somersetshire, which belong to the true carboniferous series. Thicker seams will in all probability be met with at greater depths as in the lower series of the Somersetshire coal-field, and also in the department of the Nord. That the Dover field will prove of great value there can be little doubt, even though the smallest number of seams given in Professor Prestwich's table may never be proved and the total coal area may be very limited. The development of this field both in the east and west is awaited with great interest.

The length of the coal-field of Northern France from east to west is about 94 miles. Its width is very irregular, varying from about 11 miles near Douchy to about half a mile at Fléchinelle, Pas de Calais. This coal-field is everywhere masked by tertiary or cretaceous strata. In 1890, it employed 29,000 persons, and produced nearly 8,000,000 tons of coal.

The Somersetshire and Gloucestershire coal-field extends from the north side of the Mendip Hills to Cromhall, near Wickwar, a distance of 26 miles. Its greatest width, from Ashton near Bristol to Twerton near Bath, is 12 miles; gradually diminishing toward the north until at Cromhall it is only 1 mile. Four-fifths of the whole area are covered by the New Red Sandstone, Lias and Oolites. The official returns for the year 1890 for the production of this area, show a total of 1,425,071 tons of coal, beside some fireclay, &c., and 7,982 persons employed.

The greatest length of the South Wales coal-field is 89 miles, from Abersychan on the east to St. Brides Bay on the west. Its width is irregular, but it nowhere exceeds 21 miles, and the extent is taken approximately at 1,000 square miles, 846 of these being exposed, 153 lying beneath the sea, and one square mile covered by newer formations. The quantity of coal worked in this area during 1890, according to official returns, was 29,310,295 tons, beside fireclay, ironstone, &c., and the number of persons employed above and below ground, 109,404.

The importance of the discovery at Dover is emphasized by a consideration of these facts. No doubt, a considerable area of the coal-field lies under the sea between Dover and Calais, the minerals of which under the foreshore are the property of the Crown. Interposed between the bed of the sea and the coal measures are impervious strata, of sufficient thickness to protect any coal workings which may be carried on. At Whitehaven, coal has been worked for two miles under the sea with 300 yards of cover below the ocean bed. This distance might be indefinitely extended but for the cost of haulage, and the still greater difficulty of proper ventilation. As it is not practicable to sink shafts at sea, computations relating to under-sea coal are usually limited to 3 miles from the shore. But in the case of the Dover coal-field, unless thicker seams are met with, the working of under-sea coal will be at a greater cost per ton than in most of the other coal-fields in this country. This no doubt will be duly considered in fixing terms which will create a precedent in the district. No insuperable difficulties are likely to be met in sinking shafts for coal through the cretaceous and oolitic strata overlying the coal measures, for chalk of a similar character has been successfully pierced in the Belgian and French sinkings. In these cases, considerable difficulties arose at times owing to large quantities of water in the thick tertiary and cretaceous sands. Again, in Somersetshire, although similar difficulties were encountered in the sinking of shafts through oolitic and triassic strata, the coal measures were successfully reached and pierced to the required depths. The lower greensands of the London basin are known to contain large quantities of water, which would be very difficult to deal with in a sinking, should that member of the cretaceous system be found on any site chosen for a colliery. It is probable, therefore, that working the Folkestone and Dover field will necessitate dealing with heavily watered strata in the overlying formations; these, although adding to the cost of winning, are not insuperable obstacles, and will not deter the development of the coal-field, provided the seams are commercially favourable, and landlord's terms reasonable.

The foregoing observations have reference only to the probabilities of finding coal on the north side of the great axis of elevation, with which we are more particularly interested, but there is the still further question as to the possibility of finding coal in carboniferous rocks on the south of the great ridge. From the fact of the Mendips being raised after the deposition of the coal measures, the latter in all probability follow on the south side conformably to the beds of mountain limestone.

This has exercised the minds of several men, and led to trials by shafts and borings at positions calculated to prove coal, none of which, however, have met with success. They established the fact that the newer formations attain a greatly increased thickness as compared with those on the north side. To reach the coal measures, shafts must be sunk to a great depth. Then it has been held by some that the culm measures of Devonshire point to a rapid deterioration of the coal measures in that direction, so that the productive measures, if in existence at all on the south of the Mendips, are only so to a limited extent. So far, therefore, as our knowledge extends at present, the search for coal is more likely to be successful on the north than on the south of the line of elevation.

The most recent deep boring in England is one at Culford, about four miles north of Bury St. Edmunds, in Suffolk. The boring was begun in chalk, and it is notable that palæozoic rocks were struck, immediately below the lower greensand, at the exceptionally moderate depth of 637½ feet from the surface. Mr. Whitaker and Mr. Jukes-Browne have described the boring in the *Quarterly Journal of the Geological Society* for August, 1894. The age of the palæozoic rock is undetermined, but they regard it as undoubtedly older than the coal-measures.

Search for coal under the newer rocks of East Anglia is being undertaken at the present time by the Eastern Counties Coal-boring and Development Association, Limited. According to the first annual report, issued in June, 1895, it appears that on the advice of certain geological experts (Mr. W. Whitaker and Mr. T. V. Holmes,) a site for the first trial-boring was selected on the Crepping Hall estate, in the parish of Stutton, on the Suffolk side of the River Stour. On November 26, 1894, the boring was commenced by Vivian's Boring and Exploration Company, Limited, of Whitehaven. The total depth bored up to June 8, 1895, was 938 feet; and the boring was then in the Grey Chalk. The result of this undertaking is awaited with much interest.

CHAPTER III.

MINERAL LEASES AND OTHER HOLDINGS.*

Ownership of Minerals in the British Isles—Severance of Mineral and Surface Ownership—Freehold Properties—Copyhold Tenure—Minerals worked by Owners or Leased to Others—Take-notes or Licences to Search—Arranging a Mining Lease—Effect of Severance of Ownership in Working—Responsibility for Surface Damage and for Local Rates and Taxes—Theoretical Perfect Lease—Terms of Leases in Northumberland and Durham—Royalties—Annual Certain Rent—Short Workings—Second Royalty Rent under Copyhold Land—Surface Wayleaves—Underground Wayleave—Under-Sea Coal—Outstroke—Instroke Rent—Shaft Rent—Aircourse and Watercourse Easements—Buildings Reverting to Lessor at Termination of Lease—Free Houses and Coal for Miners in Northumberland and Durham—Building Leases for Cottages—Arbitration—Farms included in Mining Lease—Renewal of Lease—Reduction of Rent in Bad Times—Joint-ownership of Minerals—Terms of Leases in West Yorkshire; in South Yorkshire; in Derbyshire, Nottinghamshire and Leicestershire; in the Cannock Chase District; in West Cumberland; in Lancashire; in Scotland; in South Wales; in Somersetshire; in North Wales—Forest of Dean Coal-field—Crown Grants in the Forest—Free Miners—Gaveller of the Forest of Dean—Certain Rents and Royalties in the Forest—Sale of Gales by Free Miners—Forfeiture of Gales—Ancient Mining Customs in Derbyshire—Licences to Search, and Leases from the Crown, in Isle of Man—Terms of Leases in Kilkenny and Tyrone Coalfields—State Minerals in Spain—Leases from Concessionaires—State Minerals in France—Coal and Iron in Belgium—State Control of Coal and Iron in Germany—Ownership of Lands in United States—Experimental lease of Lead and Copper Mines by Federal Government—Mexican Grants—Patents Granted by United States Government—Effect of the Discovery of Gold and Silver in California—Boundaries of Mineral Grants—Price of Coal Lands on the Public Domain—Wayleaves over Public Land—Mining Laws of Separate States—Classification of Coal Mines in the United States—Minerals in Newfoundland—Royalties on Crown Lands in British Columbia—Ownership and Working of Minerals in New South Wales—Crown Licences and Leases in Queensland—in South Australia—in Victoria—in Western Australia—in New Zealand—in Tasmania—Ownership and Working of Minerals in India.

In the British Isles, the minerals, except gold and silver, belong to the owner of the surface, unless other arrangements have been made, involving a mineral reservation. The mineral reservation may include the power of working on paying surface damage. Again, a mining company may have purchased outright plots of minerals from an owner who retains possession of the surface. Under the Enclosure Act of 1845, and its later amendment and extension, the Lord of the Manor has the minerals under what at one time was common or waste land, while the surface belongs to other owners. The rights of the proprietors of the minerals vary according to the terms of each Enclosure Act. Manorial rights may be transferred by sale and purchase.

In the Isle of Man the Crown owns all the minerals, except those under an extremely limited area.† To the Crown also belong the under-sea minerals along the foreshore, except where the owner of the adjoining land has obtained a title to the foreshore by some ancient grant or prescription.

The Crown, of course, owns the minerals under Crown lands in the same way as a private owner, and in the County of Durham some of these are leased on terms prevailing in the district.

In Wales the Crown is Lord of the Manor over many thousands of acres, beside owning unenclosed land.

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* The information contained in this chapter has been largely obtained from the published evidence given before the Royal Commissioners on Royalties, 1890–1.

† In 1765 the British Crown purchased the Island from the Athole family, the Duke of Athole being then Lord of the Isle.

The Ecclesiastical Commissioners, representing a Department of the State, own considerable areas of land, as do also, to a smaller extent, other public bodies and corporations, such as universities and colleges; municipal corporations; charities and hospitals; and the Board of Admiralty, who possess the forfeited estates granted to Greenwich Hospital, the minerals under which are leased by them on terms that are usual in the district, or that are within their power of granting.

All gold and silver belong to the Crown, but copper, tin, lead, and the other base metals are not mines Royal. In early times the Crown claimed all minerals.

Only one gold mine is now in active operation in the United Kingdom, viz. the Gwynfynydd or Morgan Gold Mine, near Dolgelly in North Wales, belonging to the Morgan Gold Mining Company.

The following are the royalties chargeable by the Crown in the British Isles :—

If lease granted to owner	$\frac{1}{30}$th	of gross	produce.
,, ,, ,, others	$\frac{1}{20}$th	,,	,,
,, under Crown lands	$\frac{1}{15}$th	,,	,,

These rates have not been invariably the same, and may possibly undergo further alterations.

The Crown does not, as a rule, give licenses to search for gold on private lands unless a gold miner has first obtained a license from that owner to work the baser metals which belong to him. There seems to be some doubt as to the right of the Crown to enter upon the lands of private owners, without their permission and to erect machinery thereon or divert the water courses, or if the right exists it is never exercised. The gold found in Wales is associated with the baser metals running in the same veins, so that it may be impracticable to work one but not both. If a mine is worked exclusively or mainly for the gold or silver obtained it is of course a mine Royal, and that even though copper or other base metal may have been the original object. The gold and silver may be claimed by the Crown by virtue of the right of pre-emption of the ore brought to the surface. The lessee of the mine would exercise his previous right of mining ore and bringing it to the surface, but as long as the base metals were present in merchantable quantities, they would be paid for by the Crown at certain fixed prices, and possession of the rest claimed. When not present in merchantable quantities there can be no pretence that the mine is not worked primarily for gold or silver, but it may sometimes be extremely difficult to decide whether the base metals are sufficiently plentiful to constitute merchantable quantities, owing to fluctuations in the market. If the Crown does not exercise its right of pre-emption, the precious metals together with the baser ones are taken possession of by the lessee of the mine. It is believed that the right of pre-emption has never been exercised.

The veins carrying metallic ores may continually change their character as to quality and yield.

Owing to the dual ownership set up on private lands, the gold miner has practically two royalties to pay. In some of the Australian quartz no baser metal of any market value is found, and the quartz is crushed for the sake of the gold only.

There are no mines which can be classed as silver mines working in this country. It is true that silver is obtained, but the mines yielding it do not work chiefly or solely for silver. It is contained in varying degrees of richness with lead ore and obtained almost exclusively from this source. In Crown leases for lead mines, the royalties charged are determined by the amount of silver in the lead, but in other cases the yield of silver is so small that the Crown foregoes its claims.

Freehold properties may be inherited or acquired otherwise. The former are either inheritances absolute, called fee simple, or inheritances limited which are either qualified or base fees, or fees conditional. Fee simple is the largest estate or interest which is possible in landed property; the owner is free to dispose of it to whom he pleases in his lifetime by deed, or by will and if he dies intestate it descends to his heirs.

A qualified or base fee has some qualification or limit annexed which may determine the tenure of the estate.

A conditional fee restrains to some particular heirs exclusive of others. A person who holds land which by deed or will has been tied up so as to be descendible to the heirs of his body is called a tenant in fee tail and his estate a fee tail.

A freehold not of inheritance is an estate held by the owner for his own life only, or during the life of some other person, or until the occurrence of some uncertain event.

Copyhold tenure is a survival of the feudal ages, and originated from the king in olden times granting large estates to his lords, who thereupon let out to tenants parts of their lands in return for services rendered. In the course of time money payments took the place of those services. At the present time the title to copyhold property is altogether constituted by custom. It is supposed in law, to be held at the will of the lord of the manor; the mines and minerals belong to him; without his license the property cannot be let on lease for more than a year, unless there is a special custom of the manor to the contrary.

Copyhold tenure has certain liabilities peculiar to itself in the shape of heriots and fines; it is also subject to quit-rents. A heriot is the render of the best beast or other chattel (according to the custom) to the lord on the death of a tenant. Fines may be due on the death of a tenant, and on the alienation of the land: sometimes they are fixed by custom and are alluded to as "fine certain," sometimes arbitrary and alluded to as such. Where it is arbitrary the lord of the manor is allowed to inflict what fine he thinks fit upon the descent or alienation of the land, but in practice it has been decided that double the annual improved value of the property, after deducting the quit rent, is a reasonable sum; and no more is allowed to be taken unless under very exceptional circumstances. A fine is the sum payable to the lord of the manor upon any sale, mortgage, or other transfer of the property, or by the heir to whom the property descends on the death of the owner without will. In negotiating these transfers, beside the fines, stewards' fees have to be paid, which in some manors are very extortionate.

Quit rent, in law, is a small rent payable by tenants of manors which leaves them free of other charges.

In many copyhold manors there are special customs not generally known, and applicable only to a particular manor.

The owner of the land in fee-simple who received the property intact, and who has not disposed of any interest he has, claims everything lying under it in a perpendicular direction from the surface to any depth.

A mining company with a large capital may acquire by purchase the whole of the surface with the minerals they intend working, but the heavy outlay which this necessitates, and the length of time which must elapse before a dividend can be earned, preclude the general adoption of this course.

Minerals are in some instances worked by the hereditary or other owners for their own profit, but usually they are leased from the owners (the lessors) to others (the lessees) whose interest it is to produce them skilfully and economically, and then dispose of them so as to yield a profitable return on the capital.

On an unproved estate, or where there is much room for speculation as to the existence of minerals in paying quantities, a *take-note*, or license to search, is first granted by the owner to an applicant. Usually take-notes are for a period of from one to three years, and occasionally for five years. A nominal rent is payable half-yearly, and a provision is inserted whereby the prospector, if successful, is entitled to a lease upon terms usually granted by the landowner or customary in the district. Take-notes, licenses, or search-notes are used more in metalliferous than in coal-mining districts.

A permanent mining lease is a very important document, and should be studied with the utmost care by those who will afterwards be bound by its covenants. The lessee will of course calculate the area he may require, and will consider such questions as the quality of the coal and its disposal by rail or otherwise, its proximity to the markets, whether it is compact and accessible or otherwise, whether capable of extension if desired, and whether there is any severance of ownership as between the surface and minerals on any of the properties; or any joint ownership of the minerals.

The negotiations between lessor and lessee are usually made with the assistance of skilled lawyers and mining engineers, or trained and experienced agents representing both parties. Before completion, several interviews may be necessary extending over months, and even years, as all possible contingencies have to be considered.

Without such caution, speculators and others may easily enter into bargains which they will afterwards regret.

Where there is a severance of ownership as between the surface and the minerals, the lessee's powers of working may be more restricted than in cases of working the mines under freehold lands. Owners of copyhold lands are entitled to surface support, unless it is customary to provide for compensation in case of subsidence. The custom must, if necessary, be proved. Failing this, copyholders have power to obtain an injunction to prevent the extension of a working which has caused subsidence.

In 1888 an important case was tried to test the mining rights in the county of Durham. It was the action *Shafto* v. *Bolckow, Vaughan and Co.*, tried before Mr. Justice Kekewich in the Chancery Division of the High Court of Justice. Mr. Shafto owned a residential estate, Whitworth Park, and Messrs. Bolckow, Vaughan and Co., under leases granted to them in the years 1867 and 1868 from the Ecclesiastical Commissioners of England, were working coal under Mr. Shafto's copyhold land. His Lordship, in giving judgment, said there was no evidence of the custom claimed by the Ecclesiastical Commissioners as lord of the manor to cause subsidence to the surface, and to injure buildings on paying compensation to the copyholder.

Other actions of this sort have within late years been decided on the same lines.

In some districts where custom can be proved, the lessors of minerals under manors where the surface belongs to other owners, claim the right to work the minerals on paying reasonable compensation for damage to surface-owners, and exercise this right.

Special or particular Acts of Parliament may exercise an influence on the development of the minerals in different parts of the country. For instance, there is to the north of Wakefield a tract of about 2,300 acres, which is subject to the provisions of the Wakefield Enclosure Act, 1793. The Duke of Leeds owns the minerals as Lord of the Manor; the surface being allotted out has changed ownership from time to time. The mine worker here is, contrary to usual custom, entitled to work under the provisions of the 1793 Act without payment of compensation for such surface-damage as results from subsidence. The surface-owner gets no compensation for this from anyone. Where a mine-owner damages the surface by sinking shafts, by rubbish tips, or similar works, the surface-owner is then entitled to compensation. That compensation, however, is not received from the mine-owner, but from the owners of the allotments, the Act providing that a rate shall be made for that purpose.

Probably similar Acts are operating in other parts of the country.

Under the usual conditions of mining, the lessees are responsible for all damage done to the surface which results from their mining operations, and must pay proper compensation for it whether buildings are erected after the lease is granted or not. The lease sometimes stipulates that proper shaft pillars are to be left, and barriers of 40 yards or other stated width next the boundaries of the property, and, where necessary, pillars of sufficient size to support surface buildings. Lessees generally have to pay all local rates and taxes.

Theoretically, a perfect lease is one in which the lessee works out the whole of the coal during the term, and which becomes exhausted at, not before, its expiration. This, of course, is a matter for careful calculation on the part of the lessee, that he may be able to exhaust the taking by the expiration of the period, make a fair profit during its existence, and get back his outlay.

Leases are granted in Northumberland and Durham for 21, 42, or 63 years, depending upon the extent of the property and the prospect of exhaustion within the stated period. This is the custom in these counties. In entailed estates the term may be other than what is customary, so as to accord with the powers which the proprietor at the time being possesses.

In these counties the lessee has usually the power to surrender at the end of any year, upon giving a clear year's notice and paying the accrued rent; in some cases, however, it is only every third year, and in others only every fifth year.

Although lessees may have erected very costly buildings, plant, &c., and sunk deep shafts at great expense which can only be recouped by a profitable working of the colliery for a considerable period, power of surrender may become of importance in case of exhaustion of the coal, or of absolute destruction of the colliery by fire or water. Lessees will sometimes work a colliery in depressed times at a slight loss rather than close it, as the re-opening of underground roadways after standing idle is a costly matter. Again, where the colliery is being worked year after year at a loss, without, in the opinion of the lessees, the slightest hope of more prosperous times setting in to change the loss into a profit, lessees would exercise their powers of surrender, and abandon the mine in order to prevent things from going from bad to worse.

A colliery may be so situated as to make it unwise for lessees to exercise this right. For instance, pits may be sunk on a taking in which the coal turns out unworkable and worthless owing to faults, but in order to work coal from a conterminous property under the same management, to existing pits, the lessees would sometimes retain their lease, and pay the dead-rents without working the coal, rather than sink new shafts and remove the machinery to the adjoining property.

Usually the lessee only has power of surrender, but in cases where the lease enables either party to terminate the engagement at a stated break or breaks by giving the other notice, a lessor may take advantage of a time when rents are overpaid to give notice to the lessee. Such action on the part of a lessor might cause great monetary loss to the lessee, and should the lessee have expended money on workmen's cottages the loss would be so much greater.

A covenant in the lease prevents the lessee from assigning, or subletting, or parting with possession, without the written consent of the lessor. In some instances provision is made that such consent shall not be unreasonably withheld, and that no pecuniary consideration will be exacted for giving consent.

If a lessee sublets any portion of his taking, it is almost certain to be at an increased royalty rent, which he may have been able to obtain in a more flourishing state of trade than existed at the date of his lease.

The lease is subject to forfeiture if the lessee neglects to pay his rent within 40 or 60 days of its becoming due, or on any breach of covenant, or on his becoming bankrupt when the lessor may re-enter, but this provision is frequently modified by others.

A royalty or lordship on the tonnage is paid, ranging in Northumberland and Durham from 2½*d.* to 10*d.*, the average being about 5*d.* The difference in prices fixed for royalties arises from the leases having been granted at times when trade was depressed, prosperous, or unduly inflated; or from greater difficulties presented in working some mines than others; or from variety in the quality of the coal-seams to be worked; or from the difference in the depth at which they lie; or from the geographical position of the property under which the seams are to be worked. In an entirely undeveloped district, without railways or other suitable means of transport for the coal, a lessee would probably succeed in getting an unusually low royalty price fixed. If the coal to be worked is subject to a wayleave rent, it influences the royalty fixed. At the same colliery the royalty rent may vary on the different seams worked, in order to meet the increased difficulties and cost of working thin seams or those having other disadvantages such as a bad roof and floor. Where, however, there are a number of competitors for a property, the lessor of course obtains a high royalty rent.

Fireclay and ironstone where worked are subject to royalties in precisely the same way as coal.

In some leases the royalty is paid on all coal raised, in others a certain percentage is allowed to be deducted for colliery consumption. In Monmouthshire, if a lessee has power to gob any portion of the small coal, his tonnage royalties are made proportionately higher. In Northumberland and Durham leases the rent is "tentale" instead of on the tonnage, a ten sometimes consisting of 48·583 tons and sometimes of 50 tons. In these counties as the small coal is of much less value than the round, it is subject to a separate tentale rent which varies from a half to a quarter of the tentale rent on the large coal.

In order to act as an incentive to diligent and energetic working of the minerals, a *certain annual rent* is fixed, which is termed dead-rent—so that in periods of depressed trade, when the usual yearly sale is not reached, or the production is reduced owing to unexpected difficulties, so that the royalty does not amount to the certain rent, that rent is paid to the owner and there remains what are termed "short workings." These are the deficiencies for which the fixed rent necessitates payment in advance, and may go on from year to year. On the other hand, if the quantity of coal worked in any year exceeds the standard fixed to make up the certain rent, the excess is called "over workings." Usually power is given in the lease to make up "shorts" during the whole term of the lease, that is, the short workings of any year or years may be made up in any subsequent year or years of the term, though in some instances the shorts may only be liquidated during triennial, septennial or other fixed periods. If quantities in excess of the dead-rent are worked in any one period into which the lease is divided, they are not allowed for in the next, but a fresh start is made for each period. If there are overpaid dead-rents at the expiration of the lease the lessor retains them.

The certain rent reserved in leases is determined by the number of acres in the property or the probable output of the colliery, and is imposed as a guarantee that the lessee will work the minerals under the lands leased and not keep them locked up. The surface area to be leased and the probable thickness of the seam over that area enable a calculation to be made as to the probable quantity of available coal. This, after certain deductions for loss and waste in working, is divided by the number of years for which the lease is granted, and so gives the expected or probable annual output, which at a certain rate per ton fixes the dead-rent; or, the surface area is divided by the number of years for which the lease is drawn, and this gives the annual acreage which will probably be worked and in some instances forms the basis for fixing the dead-rent. The fixed rent should only be a certain proportion of what would probably be payable to the landlord for royalty, say about $\frac{2}{3}$rds.

In many modern leases a sliding scale has been adopted for the payment of royalty instead of a fixed unalterable tonnage rate. The amount of royalty rents then payable depends upon the selling price of coal at the pit's mouth, being usually from about $\frac{1}{12}$th to $\frac{1}{16}$th part of such selling price on every ton sold or raised. A minimum tonnage royalty is usually reserved, so that when the selling price descends below a certain point, the scale does not apply, or does not descend further with it. The object of a sliding scale is to give relief to the lessee during times of trade depression, and in prosperous times an increased rent to the lessor, who has had to bear his share in the adverse circumstances.

In general no royalty or fixed rent is charged during the first few years of the term for which the lease is granted, a reasonable time being allowed for the sinking of shafts and opening the workings. In some instances a very low minimum rent becomes payable a year or so after the lease commences, and gradually increases till at the fifth year or so it has reached the full dead-rent and continues the same throughout the remainder of the lease.

A dead-rent may be fixed so high as to ensure during the term granted, that the whole of the coal whether worked or not is paid for. What remains unworked becomes unconditionally the property of the royalty owner, who may sell it again, and this cannot be regarded as other than a hardship to the previous lessee. If, from unavoidable causes, the coal has not been worked by him, it appears only fair that the lessee should be compensated for what mineral remains unworked, or that he should be allowed an extension of time to take away that which he has paid for, on his paying for the use of surface land he must continue to occupy.

In Northumberland and Durham, in order to get an accurate account of the coal raised from the different royalties, books of account are kept in which are entered the actual weights of the coal worked from day to day as sent up the shafts, the plans of the workings serving to show which roads or stalls send out coal from any particular area belonging to a royalty owner.

In many districts, instead of this being done, the royalty rent is based on the acreage worked, and periodically such area is calculated from the working plans. Again, in other districts the royalty rent is charged on the tonnage, no account being kept of the coal sent up from any particular area, but an estimate is made of these from the acreage shown on the working plan and the known thickness and yield of the coal-seam.

A rental paid on the acreage may certainly have a tendency to prevent waste in working, as it is to the interest of the lessee to extract all the coal and leave behind no stumps or unnecessary barriers. At the same time there may be instances in which it is a hardship for him to pay on a considerable area left either for the support of a continually increasing number of valuable buildings, or through its being unworkable to profit owing to its inferior or faulty character.

Surface subsidence increases in proportion to the thickness of the coal extracted, and also in proportion to its proximity to the surface, and for these reasons where there are two or three seams being worked at the same time at any shallow colliery, a covenant in the lease often stipulates that the uppermost seam shall be first worked and kept in advance so that different seams would not be worked simultaneously over the same area. By a provision in the lease of an acreage royalty to that effect, the lessee may be exempted from payment for such areas of coal as are not workable to profit, and for any left for the protection of the surface. Where large blocks of coal have to be left for any purpose they always cause additional expense to the lessee in opening out the mine again, and where of frequent occurrence may seriously hamper the working. Where the coal is worked by the lessor himself it becomes a question whether he will leave blocks of coal to support the surface buildings or run the risk of injury from subsidence for the sake of the profit from the working of the coal.

Instances occur in which what are practically a second royalty and a second wayleave rent occur. In the county of Durham a large area of coal lies beneath lands of copyhold tenure. By law copyholders are entitled to surface support and are not compelled to give their consent to the coal being worked, or the non-manorial coal being conveyed under their copyhold land, even if compensation for damage is paid, unless the custom so to work can be proved. An agreement is in force in this county by which the copyholder, beside receiving compensation for actual damages done to the surface, is paid also one penny per ton on all coal worked from under the copyholds, which is equivalent to a second royalty and wayleave rent to be paid by either the lessor or lessee.

Besides providing for payment of the royalty and certain rents, a lease may have clauses dealing with surface wayleave, underground wayleave, outstroke, instroke, shaft, air, and water privileges, which may be subject to payment.

In some districts, the necessities of colliery owners as to surface wayleaves are met by the large railway companies, who wherever it is likely, or it can be guaranteed that a large quantity of coal will be worked regularly, obtain parliamentary powers to make a branch line for its removal. In others, great difficulties have occasionally arisen in consequence of owners prohibiting any coal but their own being carried over their lands. Again, where connection to a railway or shipping place can only be made by going through intervening lands, the owners may demand exorbitant or prohibitory terms. In Monmouthshire under an old Railway and Canal Act, 1792, there is a provision by means of which any mine or ironwork owner having a colliery within 8 miles of the Monmouthshire Canal, had power to make a railway to connect with the Company's Canal in case the Company refused to do so. In Glamorganshire, similar powers were given to mineral owners within 4 miles of the Glamorganshire Canal in 1790. This provision became inoperative in 1799, when a Standing Order of the House of Commons was passed requiring the usual notices to be given in the case of all railways proposed to be constructed as a means of transport to canals.

In Northumberland and Durham, these wayleaves sprang into existence very extensively at the commencement of this century before the development of public railways, there being no canals in these counties. The object of all the private wayleave railways then made was to place the collieries in communication with the sea or a navigable river, so that the produce of the collieries might be exported. The place of shipment was sometimes far away from the colliery, necessitating the construction of a long-distance private railway. This of course, would often pass over many properties outside that on which the colliery itself existed, and the owners might or might not be interested in the minerals under their surface areas. In later times, short branch wayleave railways were made connecting the collieries with a public railway, or a long-distance main private railway, a river, or the sea. Fig. 18 shows a case where a private branch line connects the colliery with a public railway, which may be used to carry the coal to staithes at the river or elsewhere. The owners of such a branch line would pay a wayleave rent for that portion of it outside the mining lease. Had the public railway company constructed that branch railway under parliamentary powers, the land would have been purchased by them, and so no wayleave rents would be paid by the Colliery Company, but a tonnage rate to the Railway

Company who made and worked it. It may have been inconvenient for a lessee to purchase all the land necessary for the construction of a railway before developing his colliery; moreover, he had no power to compel a sale of the land if he desired it, and therefore pays a certain tonnage rate for the privilege of conveying coals over these railways. The area of land occupied per mile of wayleave on a private railway having only a single line of rails laid, is usually about five acres. The annual wayleave charge on the land occupied generally exceeds, and sometimes largely exceeds, the full fee-simple agricultural value of the land.

For coals brought from an adjoining royalty, as from B, C, D, E, F, or G, Fig. 18, and raised at the shafts on the parent property A, over which the railway is laid, a tonnage payment is made to the owner of the parent property. If the wayleave is over lands unconnected with the minerals leased, as would be the case between the parent property A, and the public railway in Fig. 18, the payment is also per ton, but in this case there is a fixed certain annual rent and power to make up short leadings in a manner similar to the short workings of coal, or the wayleave charge may be fixed irrespective of the quantity carried, being then from £150 to £250 per mile per annum, railways less than a mile being charged nearly the same as those of a mile long. This fixed payment may become a serious strain on the lessee of a colliery in times of depressed trade when short time and a low output prevail. Beside the tonnage or annual payment for the coal carried, a surface wayleave contains a provision for payment of the land occupied estimated on the acreage at double its annual agricultural value, and in addition a stipulation for the lessee to maintain in efficient repair bridges, arches, culverts, drains, &c., and a liability to compensate

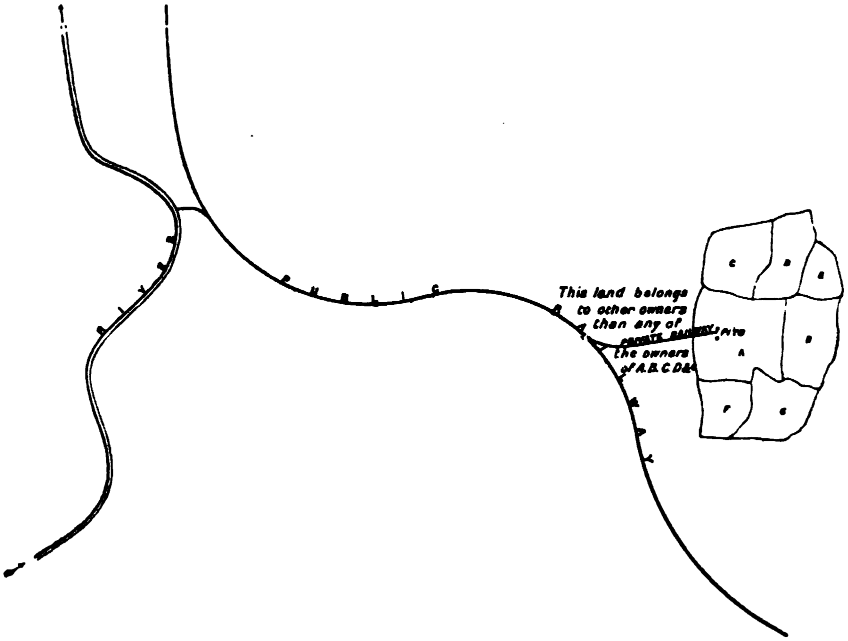
PUBLIC RAILWAY
This land belongs to other owners than any of the owners of A.B.C.D.&c.
PRIVATE RAILWAY
Pits
C
D
E
B
A
F
G

Fig. 18.—Sketch showing possible position of Land held on a Mining Lease with respect to Railways, &c.

the occupying tenant for damage to crops, &c. Very rarely a surface wayleave varies with the selling price of the produce carried.

Underground wayleave is a right on payment of a tonnage or acreage rate to the owner of the parent or other property expressed in the lease, to convey coal from an adjoining or outlying property to the shaft or elsewhere. If it is not distinctly expressed in the lease, that the lessee has power to bring other coal through a property, he is prohibited from doing so. Possibly, an owner may refuse a wayleave right, because by so doing his own coal would, as a consequence, be more largely in demand. The coal obtained in a mine may pass under lands belonging to one or more royalty owners in their outward journey before reaching the parent property. Take, for instance, the circumstances shown in Fig. 18. The whole of the colliery taking consists of the properties A, B, C, D, E, F, G, of which A is the parent property. No wayleave is charged for coal obtained from the property A on which the shafts are sunk, but a rent is payable for that which comes to the shafts from the adjoining properties B, C, D, F, G, and from the outlying property E, which is thus brought into competition with the coal obtained from the parent property. All coal from these would be subject to the payment of wayleave rent to the owner of A. As the coal from E cannot reach the property A without also passing through either D or B, the coal coming from that area would be subject to the payment of two wayleave rents, one to the owner of A, and one to either the owner of D or B, or perhaps both. The geographical position of E is not so favourable as that of the other properties, and may therefore be leased by the owner at a royalty as much lower than the royalty charged by the owner of A as would pay the necessary easements. Similarly the royalty owners of C, D, B, F, G, although better situated than the owner of E, because they adjoin the parent property A, may receive a royalty as much lower than the royalty charged by the owner of A as would pay the easements required for them. It is to the lessee's interest to have the shafts sunk on the larger of two takings, as in that case he has no easement rents to pay upon the larger part of his coal working. Moreover, the owner may insist on having the shafts sunk on the larger property, in order that he may avoid the risk of having a small portion of his property run over and then deserted by the lessee surrendering his lease.

Where the coal to be obtained is from a property which is not extensive enough to admit of shafts being sunk on it, and the wayleaves are high owing to the number of properties the coal must necessarily pass through before reaching the shaft, the royalty is usually correspondingly low. Many of these small areas which are much intermixed in some districts, are only rendered workable at all by their proximity to larger ones. Occasionally, the owner of a small area, or a grasping landlord's agent, tries to exact extortionate and prohibitory terms for royalty or wayleave, and so causes difficulties to the lessee, who consequently has to divert underground roads at considerable expense in order to avoid passing through that property. In other cases, the owner of a small area may be quite at the mercy of a lessee whose taking surrounds the small area. In such a case the lessor is obliged to let the lessee work his coal on the best terms he can get, otherwise it would soon be surrounded with the broken mine and quite isolated from any colliery that could work it.

A taking in a district of large owners under one landlord, is to be preferred to one made up of small detached lands belonging to many owners and surrounded by small owners. Mining properties of the former description have usually no difficulties with respect to wayleaves, and each lessee has access to a canal or railway on his own taking. Moreover, the continually advancing working face of the coal bears no relation whatever to physical features on the surface. In a large total area, therefore, belonging to a number of small owners, it may take many years before any coal can be obtained from those areas furthest from the shafts, the consequence of which is that immense sums of overpaid rents are paid by the lessees, to their serious disadvantage if not actual loss. In some cases these large sums of overpaid rents have accumulated through lessees taking a large number of properties with no immediate intention of working, but rather to prevent competition from others taking them. Consequently they lie dormant for many years before they are worked, during which overpaid rents keep on increasing.

When the coal in the original royalty on which the pits are sunk is nearly exhausted, the lessee of a single property, who has erected valuable works and is pressed hard by reason of a diminished coal area on which the dead-rent continues payable, may seek to utilise his capital by taking adjoining properties. As a result the lessor of the original royalty on his part tries to obtain the best terms he can exact for wayleave rent, if this item has been entirely omitted from his lease.

The coal under the foreshore of the sea and under the foreshore of tidal navigable rivers usually belongs to the Crown, and the under-sea coal can only be brought to the surface by means of shafts situated on property adjoining or near the foreshore, the foreshore being that portion of a river or ocean bed between high and low tides.* From their favourable geographical position, the owners of this property are able to demand a somewhat high wayleave rate on the under-sea coal, as much as 1¾*d.* per ton being paid in Northumberland and Durham, where it is considered a matter of arrangement that the Crown takes ⅔rds and the landowner ⅓rd of the gross rent of 5*d.* per ton paid by the lessee. Before taking it, however, the terms of the wayleave are well known by the lessee, and he urges this as an argument for reducing the royalty payable on the under-sea, or similarly unfavourably placed coal to as low a figure as possible so as to enable him to work it.

Practically, the royalty owner who is unfavourably situated generally has to pay the wayleaves demanded by the owner, who holds the key of the position. If the circumstances of the lessees are such that they urgently require the coal, hoping thereby to obtain a return on their capital outlay, and the owner of the unfavourably placed royalty remains firm in his demand for a full royalty value, the whole of the wayleave burden may have to be borne by the lessees, more especially if the coal underlying the property is accessible to the workings of two or more colliery companies who become competitors for any particular area.

Outstroke is the privilege of breaking the barrier and working and conveying underground the coal from an adjoining royalty. A lessor granting a lease of his property may insist on an unbroken barrier of coal being left next its boundaries. Another whose lessee may be anxious for a larger taking than the lessor owns, grants this outstroke privilege, which gives facilities for the lessee to work the adjoining coal and bring it through to that on which the shafts are sunk. For this privilege a rent in addition to the wayleave rent is sometimes paid.

Instroke rent is a charge in addition to the ordinary royalty rent made by the owner for the privilege granted to the lessee of going underground into his workings from an adjoining property, and is made on all coal extracted therefrom and raised at a shaft on other property. This rent is not usually charged by lessors of small properties, nor always by owners of large ones. Where it is charged, it is upon the plea that the lessee is not put to any expense in sinking shafts on a property of considerable extent.

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* Ancient grants from the Crown in some instances conferred privileges of an extensive description on the lord of a manor upon the seashore, such as the exclusive right to take sand, stone, and seaweed, besides wreck of the sea and fishery rights. The seashore, therefore, may be claimed as a portion of a conterminous manor. Where the claim cannot be proved by adducing an actual grant, or by evidence of acts of ownership, the seashore and the minerals thereunder belong to the Crown.

A shaft rent is paid for the privilege of raising coals at a shaft which have been obtained from an adjoining or outlying property, or is charged by a lessor to his lessee for the use of shafts sunk before his lease was granted.

This rent is not invariably demanded where these privileges are given.

Aircourse and watercourse rents are paid for the privileges of conveying air or water to or from an adjoining property and inwards or outwards through another property along roads which may have to be carefully preserved, thus deferring the working of a certain portion of coal next them.

Aircourse and watercourse easements on the various properties of one lessee may be very useful, but should always be kept distinct from each other. Many owners object to watercourse facilities through their properties on account of the danger of flooding. Each colliery usually deals with its own water as a whole, and for this reason barriers of coal are left by the lessee next the boundaries of his takings. If this were not done there would be no protection from water to coal workings in a property lying to the dip of another, so that although the lessor of the colliery property situated to the rise may not consider barriers next boundaries to be necessary, the owner of the coal situated to the dip would naturally expect and insist on having barriers left, so as to prevent the water from flowing to his coal. In South Staffordshire the difficulties of each colliery pumping its own water have led to a scheme being carried out by the South Staffordshire Mines Drainage Board under an Act of Parliament for dealing with the district as a whole and pumping the water at common pumping stations, for which purpose the barriers between collieries are cut through.

In fiery districts, the ventilation of each colliery is much better kept separate from that of others, and clauses have been inserted in leases in order to ensure this. By this means the damage resulting from an explosion is confined to the colliery in which it occurs.

Usually, in England and Wales all buildings necessary for the working of a colliery are erected by the lessee. A clause is usually inserted in the lease which enables the lessee to take possession of lands for building or other colliery purposes, such as a spoil bank for tipping rubbish on, by paying double the agricultural rent for the surface damage. At the end of the lease he must restore the soil or pay its value. The buildings, such as engine and boiler houses, coke ovens, offices, workshops, stables, store rooms, &c., are for the most part placed close to the pit's mouth, and being constructed of brick or stone, revert unconditionally to the royalty owner at the termination of the lease, as do also the shafts and permanent underground works for keeping the mines open. If the minerals are all worked out, these buildings, &c., may be of no value whatever to the lessor on his obtaining possession of them.

The lessor allows the lessee to take the clay, stone, and sand necessary for the building from the land without payment. The lessee endeavours to erect such structures as will last the term of lease. At its expiration the machinery still belongs to the lessee, but the lessor has the right to purchase, and if no agreement is come to between the two interested parties as to the value, it is settled by arbitration. Engine-beds are usually removable by the lessee. The buildings are often not kept in an efficient state of repair towards the end of the lease, and at its termination may be further dilapidated by the removal of the lessee's machinery, so that they are then not worth much.

In Northumberland and Durham each married workman is allowed a house, rent free, and has an allowance of free coal, paying *6d.* per fortnight for the leading only, but these privileges are considered part of his wages. In order that these houses may be near the work, they have in many cases been built on the property leased, and then form part of the surface buildings which revert to the lessor.

If, however, other land can be obtained for the purpose near at hand, it is of course usually preferred. This land may belong to the same or another owner, who grants a building lease for a much longer term of years than the colliery lease, from which it is quite distinct, and thus the cottages remain the property of the lessees subject to the payment of ground rent after the pit buildings have reverted to the royalty owners. In some districts the workmen's houses are erected upon land bought for the purpose by the lessees. Occasionally cottages are built by the lessors, the lessees paying an annual rent for them for a term of years concurrent with the mining lease. Excepting in Northumberland and Durham, it is customary for the workmen to pay rent for the houses they occupy.

In Scotland it is usual, though by no means general, for the lessees to have power to remove all buildings at the end of the term unless the owner chooses to take them at a valuation, and in some rare cases this is so in England; the lessees, however, in that case are not allowed to get the clay, stone and sand for the buildings free of charge, and very possibly the landlord when negotiating the lease fixes a higher rent to meet this arrangement.

In cases where the surface and mines ownership are severed, the buildings would not revert to the lessor at the expiration of the term, unless some special arrangement were made to that effect.

Where a property is leased on which the shafts are already sunk and all the machinery belongs to the lessor, a higher royalty is sometimes charged in accordance with their value. In districts where coal mining has been carried on for many years, perhaps centuries, as is the case in Northumberland and Durham, nearly all the original leases have expired, and the royalties paid under existing leases include the value of shafts and buildings in the lessor's possession.

There is generally a clause in the lease by which the lessee is obliged to work the coal in an approved and economical manner and in accordance with the views of a mining engineer appointed by the landlord, who inspects the mine periodically.

An arbitration clause provides for the settlement of disputes which may arise between the parties respecting the covenants or agreements contained in the lease. Each party has power to appoint an arbitrator, but if either fails to do so after a stipulated number of days the other party's arbitrator acts alone, and his award is binding. If each party appoints an arbitrator, then the two arbitrators before proceeding to the matter in dispute appoint a third. If the third arbitrator is not appointed within a stated number of days by the others, he is appointed by a Judge of any of the Superior Courts of Law or Equity, and the award of the sole arbitrator or of the two arbitrators to be first appointed without the co-operation or interference of the third or of any two of the three arbitrators is final and binding on all the parties. For the thorough investigation of the matter in dispute, witnesses are examined upon oath by the arbitrators, and documents in their custody bearing upon the subject may be demanded.

In Yorkshire, and also in some other districts, a large farm is sometimes taken in connection with the colliery leased, and where the land is good this may be of benefit; but in districts where the land is poor and it is thrust on the lessees to cultivate it may be a considerable burden. The taking of the farm should be optional, and distinct from the mining lease, although running concurrently with it.

If toward the termination of a lease the coal is not all worked out, the lessee, if desirous of continuing to work the colliery, approaches the lessor for a renewal of the lease. The lessor, although not obliged to renew, generally meets him in a reasonable spirit, so that the lease is renewed on terms satisfactory to both. No doubt hardships have been inflicted on lessees under such circumstances, but these cases are by no means common. When conducting the negotiations overpaid certain or dead-rents are sometimes taken into consideration, and carried forward into the new lease, at others they are absolutely forfeited. It would not be usual to carry shorts forward in case minimum rents had been charged in the expired lease. On the renewal of a lease, a rent is sometimes charged for the shafts and surface works which have now become the property of the lessor, so that the lessee pays for improvements which he has himself carried out during his previous lease.

A lessee may have property of his own on which he sinks his shafts and erects all his machinery, although most of his coal may be obtained from the properties of other owners. Where this is so, it may assist him in obtaining better terms for the renewal of his lease, than he otherwise would get.

The life of a colliery is sometimes prolonged by a lessee not renewing his lease, but by his leasing coal seams below that exhausted, and within the same property, so that the same shafts and plant are used for raising the coal.

In times of extreme trade depression, appeals have been made by lessees to the royalty owners for a reduction of rent, which in some cases have been granted. In some districts there has been difficulty in obtaining relief owing to the fact of estates being vested in trustees whose legal obligations deter them from giving relief in cases where the lessor acting for himself would probably have acceded to the request.

Difficulties in working the coal may arise from joint-ownership of the minerals. For instance, a lease may be held of the coal under common land which has been enclosed and belongs jointly to two or three owners as lord of the manor in well-known proportions which are not partitioned off. In case mining difficulties should prevent the coal from being worked out during the term of lease, and render its renewal desirable, the joint-owners may not be unanimous in their opinion as to carrying the shorts forward or on other items submitted for their consideration. Again, they may take different views respecting concessions which are asked for during times of great trade depression.

In West Yorkshire the terms of leases vary from 21 to 40 years. The rent is paid at a price ranging from £50 to £300 per acre per annum, or at a fixed sum per foot thick and varying from £20 to £40 per acre per annum, a fixed minimum rent being stipulated for, and no allowance being made for free coal for colliery use. There is usually power within the period of the lease to make up short workings. An acreage rent is payable to the lessor on whose land the pits are sunk for coal worked from adjoining properties and raised at the pits referred to. This shaft-rent includes underground wayleave which is not separated from it. Surface wayleaves are paid generally by the acre, but sometimes by the ton.

In South Yorkshire the terms of leases vary from 30 to 60 years. The rent is paid on the acreage worked, according to quality and thickness. The rent for the Barnsley Bed varies from £120 to £375 per acre, for the Silkstone seam it varies from £25 to £60 per foot thick per acre. There is always power to make up short workings by subsequent workings within the term of the lease. Surface wayleaves are but little known. Underground wayleaves are subject to a payment per acre.

In Derbyshire, Nottinghamshire, and Leicestershire, the terms of leases are from 21 to 60 years, the rental being paid chiefly by the acreage, though sometimes by the foot thick per acre, and sometimes by the ton. Generally power is given to make up fixed rents paid in advance by subsequent working within the term of the lease. In some instances the coal for colliery use is free from rent where there is a tonnage royalty.

In the Cannock Chase district the terms of leases are from 31 to 60 years, the rental being paid on the tonnage of 2,640 lbs. The dead-rent is fixed at from 20*s*. to 60*s*. per acre on large areas, and power is given to make up short workings within the period of the lease. The coals actually consumed for colliery purposes and those allowed to workmen are free from rent.

In West Cumberland, the terms of leases vary from 21 to 63 years, there being no power of assignment. The royalty is paid by the tonnage mostly, but sometimes by a per-centage on the net sale, then being from $\frac{1}{7}$th to $\frac{1}{12}$th. The dead-rent is fixed at from 10*s*. to £2 per acre. There is power to make up short workings, sometimes over three subsequent years, sometimes over seven, and sometimes over the whole term. Usually five per cent. of the coal raised is free from royalty rent as being allowed for colliery consumption.

In Lancashire the terms of leases vary from 20 to 50 years. The royalty rent is for the most part paid by the acre, which is either statute measure or a Cheshire acre of 10,240 square yards. It varies from £50 to £120 for ordinary coal per Cheshire acre per foot thick. For cannel it is as much as £200 per foot thick per Cheshire acre, but varies according to circumstances. It is charged on all coal obtained, large or small. There are a few leases under a tonnage royalty. There is no power of surrender in the majority of leases. Overpaid rents are usually allowed to be made up by subsequent workings in any year or years within the term of the lease, but sometimes it is only so in specified breaks of two, four, or six years, or in quinquennial or septennial periods. Surface and underground wayleaves are usually paid by the acre per foot thick of mine worked, and range from £5 to £10 per Cheshire acre. The general wayleave is "*under, over, or through*." Shaft rents are not generally paid. In a few cases where shafts were in existence at the time of granting a lease, an annual charge of from £20 to £50 has been made for them. There is no deduction made from the rent in respect of coal used for colliery consumption.

In Scotland, the usual terms of leases are from 19 to 31 years, a very few being for longer periods. The rental is mostly paid on the *triping* ton—*i.e.*, on the whole output of large and small (dross) coal as raised—the royalties being much higher for the valuable cannel than for other kinds of coal. Some of the former are charged from 1*s*. 2*d*. to 1*s*. 4*d*. per ton for royalty. In some cases the royalty charges for large and small coal are distinct, the latter being much lower than the former. Coal used for colliery purposes and by the workmen is allowed rent free, exact quantities being ascertained. Usually power is given to make up short workings within the term of the lease by breaks of from two to five years ; in some instances they may be made up any year throughout the lease ; occasionally there is no power whatever to make them up. Houses and buildings are erected by the lessees, which may or may not revert to the landlord according to the conditions of the lease. There is usually no wayleave on the surface where one underground, this one being considered sufficient. Unless the lessee breaks the conditions of the lease, or fails to pay his rent within a specified time, the lessor has no power to anticipate the natural termination of the lease, but the lessee can abandon his lease at any time on proving that the minerals are unworkable to profit, and there is a clause in the lease which enables him to abandon it on giving proper notice at the third, fifth, or other year without assigning a cause.

In the South Wales coalfield, leases are usually granted for a term of 60 years, some of the earlier ones having been granted for 99 years. In the old leases there is power to surrender upon the exhaustion or unprofitableness of the coal. In modern leases, the lessees have power to surrender on giving twelve months' notice at the end of certain fixed periods such as the third, sixth, ninth year, and so on throughout the term of the lease. There is power of assignment subject to the lessor's assent, which is not to be unreasonably withheld. The dead-rent is fixed at from £1 to £2 per acre and up to £5 in exceptional instances.

The royalty rent is paid on the tonnage, the ton consisting either of 21 cwts. of 120 lbs. = 2,520 lbs., or the imperial ton of 2,240 lbs. The average royalty paid on the steam coals is about 8*d.* per ton for large, which forms about 85 per cent. of the yield, and 4*d.* per ton for small, the remaining 15 per cent. The royalty on house coal is higher than on the steam coal, being as much as 1*s.* 6*d.* per ton in exceptional cases for all coal obtained, large and small mixed. Where a sliding scale has been adopted it is usually from $\frac{1}{10}$th to $\frac{1}{12}$th of the selling price of the coal at the pit's mouth, a minimum royalty of 6*d.* per ton being usually reserved.

In many of the early leases granted when the coalfield was undeveloped, a light dead-rent was provided for and no royalty. A portion of these properties has since been sublet on terms which pay the dead-rent, and the remainder is in the hands of lessees, practically royalty rent free.

In some cases short workings may be made up throughout the whole term of the lease, in some they may be made up for any year or years within three to five years of any year in which the certain rent has exceeded the amount of royalties, in others a provision in the lease divides the term into fixed periods of three to five years, within which period only the recovery of over-paid rents is possible.

No shaft rents are charged, but surface and underground wayleaves are, the former usually being light owing to the proximity of a railway to the shafts. The latter, including waterleave, airleave, and shaftage, are charged only on coal and ironstone, and not upon rubbish or materials.

Coal for colliery consumption is free from royalty, sometimes the exact quantity used being allowed, in others the practice is to limit the quantity to from 5 to 10 per cent. of the output.

The whole of the colliery consumption is apportioned to the several properties under lease, according to the quantities worked.

In the Somersetshire coalfield, leases are granted for terms varying from 30 to 40 years. The usual fixed rental is from £1 to £2 10*s.* 0*d.* per acre. In some instances the royalties are charged by the ton, being from 5*d.* to 9*d.* per ton, the average being about 6*d.* In some cases the royalties are paid on the selling price of the coal at the pit's mouth, varying from $\frac{1}{10}$th to $\frac{1}{30}$th and averaging about $\frac{1}{16}$th. Short workings may be made up in some instances within periods of 3 years, in others of 6 years, and in some cases every year throughout the whole period of the term. The lease may be surrendered on the exhaustion of the coal, or when it becomes unworkable to profit, or by notices varying from one to two years.

The coal used for colliery consumption and workmen's coal is usually free from royalty. In some leases the amount allowed to be deducted for colliery consumption is restricted to 10 per cent. of the output, and the workmen's coal is charged at about half the usual royalties.

In North Wales, the usual term of lease is 42 years, excepting in cases where the proprietors' powers are limited to 21 years. The dead-rent is very variable. The royalty payable varies from £15 to £35 per statute acre per foot thick, the average being about £25. In a few instances a royalty is fixed by sliding scale, varying from $\frac{1}{12}$th to $\frac{1}{20}$th of the selling price of the mineral at the pit's mouth.

Short workings may be made up to recoup over-paid dead-rents, sometimes within fixed periods of three years or more, and sometimes throughout the whole period of the lease. In leases of old date there is no power of surrender until the whole of the minerals are worked out, whether being worked to a profit or not, and there is no power of assignment without the written consent of the landlord. In modern leases there is no power to surrender unless the minerals are exhausted, or unworkable to profit, and no power to assign without the consent of the landlord, such consent not to be unreasonably withheld. The colliery consumption is mostly allowed free from royalty, and is sometimes restricted to 10 or 12½ per cent. of the output.

In the Forest of Dean the minerals belong to the Crown, and the nature of the tenure under which they are worked is unique. This coalfield forms a more perfect basin than any other in England. There are eight seams over 2 feet thick, and some not very important clay ironstones. In the Carboniferous Limestone there are brown hæmatite iron ores as well. A group of Sandstones about 143 yards thick divides the measures into two series, and are much quarried for the excellent stone yielded. The upper series extends over a much smaller area than the lower; the latter are much more heavily watered than the former, owing to the exposed outcrop over the more extensive area. There are 26,000 acres of forest woodland, the property of the Crown, and the total area of coal measures within the outcrop of the Trenchard Delf, or lowest seam, is about 19,000 acres. The division into two groups of seams by the series of Sandstones, which are by some thought to correspond with the Pennant of the Somersetshire and South Wales coalfields, has suggested to eminent geologists the probability of the Forest of Dean, the Somersetshire and South Wales coalfields having at one time formed one large coalfield which was subjected to great disturbance, followed by a long period of denudation.

In the Forest of Dean the Crown grants licences to work the minerals to men called Free-miners, and to them only.

Free-miners are all male persons above the age of 21, who have been born within the hundred of St. Briavels, and have worked a year and a day in a coal or iron mine. In order to become a free-miner the applicant must fill in a prescribed form and produce a certificate of birth or baptism to the Deputy-Gaveller. He is then entered in the Register of Free-miners, and is supplied with a certificate of registration signed by the deputy-gaveller giving his number, name, age, and date of registration.

Technically the Gaveller of the Forest of Dean is the First Commissioner of Woods and Forests, and by Act of Parliament his deputy must be some person skilled in mining.

The free-miners have worked the mines in the Forest of Dean for centuries, under a licence from the Crown at a payment of what is meant to be a fifth of the profits. There are records showing that coal mining was carried on in the Forest as far back as 1282. Formerly the proceedings were regulated by a court or jury of free-miners, and documents over 200 years old are now in existence, containing a record of their deliberations. The powers of the free-miners' jury were insufficient to prevent confusion and disputes arising from trespasses on one another's coal. Consequently in 1838 a Commission was appointed, by whom three separate awards were made, one for coal, one for ironstone, and a third for quarries. The award was signed in 1841, and its provisions have been in force ever since.

By an old custom "every free-miner, duly qualified, claims the right to demand of the King's gaveller a gale—that is, a spot of ground chosen by himself for sinking a mine—and then, provided it does not interfere with the working of any other mine, the gaveller considers himself obliged to go, receiving a fee of 5*s*., and inserts the name of the free-miner in the gale book."

This custom still continues in force, subject to modifications introduced by Act 1 & 2 Vict. c. 43, entitled "An Act for regulating the opening and working of mines and quarries in the Forest of Dean and Hundred of St. Briavels, in the county of Gloucester."

The free-miner lodges his application for a gale on a printed form at the gaveller's office. If the gale applied for is not already granted, or in charge,—that is, does not belong to any party other than the Crown—the application is entered into the book kept for the purpose at the gaveller's office; if the gale should be in charge or owned, the application is not entered but filed. If the free-miner thinks his application is the first one on the book for the gale he applied for, he sends its number to the gaveller, begging that it may be granted to him. If he proves to be the first applicant and no objection arises, instructions are sent from the Office of Woods and Forests to the gaveller's office to prepare the approbation paper, which contains the terms upon which the gale will be granted. These terms, in the case of coal mines, allow two years in which to open the mine, during which no royalty and no dead-rent are payable, and afterwards no royalty is payable on coal consumed by colliery engines or for other necessary colliery purposes. In the case of iron mines, a period of four years is allowed for opening the mine, during which no royalty or dead-rent is payable.

The terms vary, but usually include payment of a certain annual dead-rent of about £20 to the Crown, for which 1,600 tons may be worked free of any other rent, being at the rate of 3*d*. per ton; and for any quantity beyond 1,600 tons or "overs" 3*d*. a ton. It is provided also that the gale shall be opened according to the rules and regulations then in force or be forfeited. The approbation paper, together with the draft grant, is returned to the Office of Woods and Forests for approval. When approved, the grant is advertised for two consecutive weeks in three local papers.

After the second week's advertisements, a clear week must elapse after which the gale is granted to the grantee, galee, or free-miner, a fee of 5*s*. being payable for the parchment. Thus, from the time of application to the receipt of the grant, five or six weeks must elapse. In the case of a quarry free-miner, nearly the same routine is gone through, excepting that he is not entitled to apply for coal or iron gales.

No free-miner can have more than three gales granted to him. If one or more allotted be forfeited, the number may be made up again to three, if the miner is otherwise entitled to it.

A gale of coal and a gale of iron are not granted to the same man, but are kept distinct from each other. Also the upper series of coal seams are kept distinct from the lower, separate gales being granted for each.

The area of the gale is at the discretion of the gaveller and deputy-gaveller, and depends upon the depth at which the minerals lie, the probable capital required in sinking and for plant, &c. The intention is, that the smallest gale shall be large enough to justify the expenditure of capital for the purposes named. One of the deepest gales granted is about 2,000 acres in extent, whilst some of the shallow ones are only about 100 acres.

As to royalties, the maximum is the right of the Crown to put in a fifth man to work the coal for the Crown's profit after the mine has been opened; that is, assuming the gale to be a partnership property, the Crown would have the right to put in one partner to four, but would bear no part of the expenditure incurred in winning the mines. There is then this peculiarity in the partnership that the Crown pays no share of winning, but is entitled to receive one-fifth of the profits after the minerals are won.

The value of this fifth share is reduced to cash by the gaveller or his deputy, and if the dead-rent and royalty are fixed by them at what the free-miner thinks excessive figures, he can go to arbitration with the Crown, and both are bound to accept the decision of the arbitrator.

The royalty is only fixed for 21 years for each gale from any assessment, and at the expiration of that time it is open to the gaveller on the one hand, or the free-miner, or the person who has succeeded to his interest on the other, to object or express a desire to alter the rent; if the interested parties cannot agree, the matter is reterred to arbitration.

In an arbitration case of an unopened mine, it would devolve on the Crown to prove to the arbitrator what, in its judgment, it will cost to sink the pits, to erect the machinery, &c.; what afterwards will be the probable quantity worked, and the probable profit realised in order to ascertain the probable value of the Crown's fifth proportion.

The licence once granted to the free-miner, unless forfeited, is granted in perpetuity, and is in the nature of a freehold; it is transmissible by will or by sale to one who is not a free-miner.

A capitalist may therefore buy a gale or as many of them as he likes. A free-miner, who seldom has much capital, may be able to work a crop gale successfully, but to win the coal in the deep gales requires a considerable outlay, and if the coal is 500 or 600 yards from the surface, the miner has to part with most of his interest in order to induce any capitalist to find the money for sinking the pits. In many instances, the free-miners merely sell the grant they have obtained to some speculator, who retains it in the hope of finding some one else to whom he may sell his interest at an enhanced price.

The following advertisement appeared in the *Dean Forest Mercury* of Feb. 5th, 1892, and explains the way in which a gale becomes subject to forfeiture:—

"V. R.—Dean Forest Mines.—*Notice to Free-miners.*—Whereas, the Persons entitled to or in possession of the Gale of Coal called Nag's Head Colliery have desisted from working the same for a space exceeding Five Years at one time after the vein of Coal has been gained, contrary to the rules and regulations contained in the Second Schedule to the Dean Forest Mining Commissioners' Report of Coal Mines, dated the eighth day of March, one thousand eight hundred and forty one, whereby the said Gale has become liable to Forfeiture, I Hereby Give Notice, that unless the working of the said Gale is *bonâ fide* resumed on or before Wednesday, the tenth day of February, 1892, it is my intention, soon after Five o'clock on that day, to declare the Gale Forfeited to Her Majesty.

"I Further Give Notice, that in the event of the Forfeiture of the said Gale, applications for a grant of a Gale of a Licence to get the Coal in the said Gale will be received at the Gaveller's Office, at Coleford, at and after Ten o'clock a.m., on Thursday, the Eleventh day of February next.

"Geo. Culley, Gaveller of Dean Forest.

"Office of Woods, &c., 22nd January, 1892."

The deputy-gaveller's surveyor makes all the surveys, and keeps up all the plans of every one of the collieries and iron mines in the Forest of Dean.

In two districts of Derbyshire, very curious ancient mining customs have existed from time immemorial, and are now confirmed by Act of Parliament. The districts are Wirksworth and High Peak, in both of which lead is mined. The lead in them belongs to the Crown through the Duchy of Lancaster.

Some of these mines are believed to have been worked since the time of the Romans, whose inscriptions have been found on pigs of lead in several places at Matlock and Cromford.

In the wapentake or hundred of Wirksworth, by an Act of Parliament passed in 1852, an equal right is given to every one, strangers and inhabitants alike, to search for lead ore upon nearly all lands, whether cultivated or moorland, without the permission of the owner or tenant, and without paying compensation for damages. The same Act contains clauses enabling the miners to obtain grants of mines on easy terms from the barmaster, and subject to the tenure of workmanship they maintain a complete title to them. The appointment of barmaster is made direct from the Crown. His duties are to superintend the measuring of the ore (in measures peculiar to the neighbourhood), to choose the grand jurymen and small jurymen, to summon them when required, and to perform other business, such as setting out, giving away, and regulating the working of the mines.

The Wirksworth district is leased by the Duchy of Lancaster, but it is merely on nominal terms, the lessee receiving no pecuniary benefit from it at all, for the dues payable go direct to the Duchy of Lancaster. The dues are called "lot" and "cope." In the Statute the lot is defined as consisting of a thirteenth part, but as a matter of fact a twenty-fifth only has been taken for a long period. The "cope" payable is 6*d*. per load. These dues are received from the miners upon the gross produce when it is washed, cleaned, dressed, made merchantable and fit to sell for smelting. Under the Act fees are payable by the miner to the barmaster.

If on a search being made ore is found, the searcher applies to the barmaster, who can then set out a mine. The proportion of surface set out by the barmaster is in his discretion, and that of the two grand jurymen he selects for the purpose. The grand jurymen are practical miners, who consider each case on its merits, and set out the area they think reasonable for the commencement of operations. A large area would not be set out in advance, but sufficient to meet the probable requirements for two or three years, after which, if the necessity arise, a further allotment may be marked out. The miner has a right to the water or to the highway, and the surface allotted gives him sufficient room for buddling the ore and making all other surface arrangements. This has reference to the surface only. There is no restriction underground within the wapentake, and the miner can follow the lode as far as he likes. Besides setting out the mine, the grand jurymen inspect it in case of dispute, their practical knowledge of the veins worked being valuable. In case of a dispute on a point of law, of title, of trespass, &c., the small jury, consisting of mine owners who are a more highly educated class, act as arbitrators. A steward is appointed as judge, and the jurymen give their opinion on certain points laid before them. The judge decides and gives sentence in accordance with the opinion of the jury. The two juries and the court are recognised by law, and are established by Act of Parliament. The law for the most part merely confirmed ancient customs, it being thought that a special Court so established would be more suitable to deal with mining customs than any ordinary one. The Act does not enforce the keeping of plans of the workings, so that, unfortunately, there is no record of old workings, consequently much money is wasted in driving long exploring places.

If within a reasonable time after allotment, a person does not work his mine, anyone else may apply for it to the barmaster. The duty of the barmaster then is to give notice to the owner as written in the barmaster's books, that if the mine is not worked within three weeks or a valid excuse given, it will be assigned to another.

Many of the mines are very small, the veins not extending far, so that a working man may hold the title of a limited area, though, of course, as the ore becomes exhausted, it is more difficult to make a living from this source. Other circumstances favourable to the small capitalist are the shallowness of the workings, and their situation above the water-level, requiring no machinery for pumping water out. If the mine is of considerable area and depth, and copiously watered, large capital and costly machinery will be requisite. In some cases of this kind the owner starts a company, in others this is done by a number of neighbouring owners who work the whole as one mine.

Any mine once acquired is regarded as a freehold, the owner having the rights of sale, &c., and of transmission by will as in the case of ordinary freeholds.

Mines in private property in other parts of Derbyshire, are contracted for quite differently from those of the Wapentake.

The High Peak Mining Customs were confirmed by an Act of Parliament passed in the year 1851, called the "High Peak Mining Customs and Mineral Courts Act, 1851." This Act applies to seven small liberties of the King's Field, viz., Monyash, Taddington, Upper Haddon, Bradwell, Castleton, Hucklow, Winster, and the Out Liberties. In addition, there are private liberties within the High Peak district, but these are not subject to the High Peak Act.

The same mining customs exist over all other properties—excepting private ones—as those already described at Wirksworth. There is a difference, however, with regard to payment of the dues. In the King's Field in the High Peak, the lot consists of one-thirteenth under the Act of 1851, and the cope payable is 4*d.* per load, instead of 6*d.* Although one-thirteenth is claimable for lot, as a matter of fact, one-twentieth is taken in five out of the liberties, one-thirtieth at Castleton, and one-twenty-sixth in Winster. The Duke of Devonshire leases the King's Field from the Duchy of Lancaster, paying a certain rent for it and then farming it out. The lot and the cope are paid to the Duke of Devonshire. Besides these two dues, the miner has to pay tithes, so that the royalties of lot and tithe make about one-thirteenth throughout the district.

The royalties are taken in kind. After measuring the ore the barmaster gives the miner a ticket. All the ore then goes to the lead merchant, who will not pay the miner until he has the barmaster's note showing the proportion to be deducted on his account. It is thus taken in kind, the barmaster selling it to the merchant, and paying the cost of its transit.

In the Isle of Man, the minerals with trifling exceptions, amounting to about a thousand acres, belong to the Crown. There is no coal worked in the island, and it would appear from the absence of the usual deposits found over the carboniferous limestone in English coalfields, that if the coal measures at one time existed here, they were afterwards removed by denudation. Lead mines which yield lead, blende, and silver, however, have been known, and have rendered the Isle famous from early times. Some of these are successfully worked now, but others pay their shareholders no dividend. The mineral rights of the Crown are managed at the Office of Woods and Forests. Any person wishing to prospect there, first applies for a licence to search for and develop the minerals. That licence enables him to enter into the common Crown lands or those of the customary freeholder. For so doing he is only required to pay compensation for damage to the surface. The amount payable may be arranged with the surface holder, or in case of disagreement, settled by arbitration as assessed under the direction of a judge or deemster, by a jury of four men. The leases are granted for 31 years at royalties of from $\frac{1}{10}$th to $\frac{1}{20}$th of the selling price of the ore, whether the mines are working to profit or not, and subject to certain annual dead-rents.

In Ireland the only coalfield of value is the Kilkenny, producing anthracite coals, used chiefly for malting and household purposes. The leases granted are from 21 to 31 years, according to arrangement, and there is a certain annual dead-rent varying from £100 to £200, and a royalty of from $\frac{1}{12}$th to $\frac{1}{16}$th on the selling price of coal.

In some cases, the coal is not leased, but worked by a contractor, who pays a higher rent of $\frac{1}{8}$th or $\frac{1}{10}$th of the selling price, but in such cases the landlord pays all the taxes.

The Tyrone coalfield has not been much developed hitherto, and the same may be said for another much smaller coalfield near Ballycastle in the county of Antrim.

In Spain the minerals belong to the State, which theoretically takes possession of all, and then divides them into three groups.

The first class include all minerals of an earthy nature, silicious stones, slates, granites, basalt, limestones, sulphate of lime, sands, marls and clay, and generally all material used for *construction.*

The second class comprises alluvial deposits whether metalliferous or not, bog iron ore and lake ore, emery, ochres, rubbish heaps and slag heaps, and metalliferous ground proceeding from old workings, peats, pyritous earth, magnesian earth, saltpetre, phosphate of lime, fluor spar, steatite and china clay.

In the third class are the metalliferous substances, anthracite, coal, lignite, asphalte, pitch, bitumen, petroleum, mineral oil, graphite, saline substances and mineral waters.

The minerals of the first class are granted to the owner of the surface, as also are those of the second class; if, however, he refuses to work them, they are made the subject of a concession precisely the same as the third class, but preference to work this second class is given to the owner of the soil.

A person desiring to work minerals presents a petition to the civil governor of the province, together with a full topographical description of the surface. The minimum area granted is about 2¼ acres, and consists of a square under the surface, whose side is 100 metres or 109·36 yards. This is called a *pertenencia,* and the applicant in his petition defines the number of pertenencias he claims and describes their boundaries. He also describes the minerals he desires to work, gives a name to the mine, and sends these particulars to the governor. The governor forwards them to the engineer of the province to see whether the ground is free. The applicant is not obliged to prove that there are minerals on the ground, but merely makes application for it. If the ground is free, the governor publishes the petition in the official gazette of the province on the third day, and calls upon any opposing parties to lodge their protest within 60 days. Should there be no protests within this time the governor orders the mine to be marked out by the engineer of the province. This is done within four months, and the fact that such and such mines are going to be marked out is announced in the official gazette of the province about eight days before it takes place. Notices are sent to all the neighbouring landowners, so as to give them the opportunity of being present, or of sending others to make any representations they may think proper.

If no protest is presented within 10 days, the governor declares that the petition is well founded, and orders the title to be made out. Thus within about seven months from the date of petitioning the applicant may secure his title, but if there is any protest, the matter may be delayed for a long period, the delay varying according to the nature of the conflicting claims. The admissible grounds of protest are prior claims, infringement of boundaries or want of free space, or technical faults in the claims, so that a petition in which the description of the mine is not well given, or which contains technical faults, may be set aside. The protest is lodged with the governor of the province, who gives his decision thereon, subject to appeal to the Minister of Public Works. If the person protesting is not satisfied with the minister's decision, he again appeals to the Contentious Department of the Council of State, called the *Contencioso.*

The concession is granted in perpetuity, subject to the annual payment of a surface tax or rent to the Government, and in the case of underground mining, conditionally upon the concessionaire contributing his quota to the general mines drainage scheme of the district.

The surface tax for all minerals except iron ore is 10 francs per hectare, and as a hectare is about 2¼ acres, this is equivalent to about 3*s.* 7*d.* per acre. In the case of iron ore it is 4 francs per hectare, equal to about 1*s.* 5*d.* per acre.

The Government has power to compel any group of miners to associate for the purpose of draining their mines; and the refusal of any one to do so, or to pay the surface tax, renders him liable to forfeit his concession. When the title is once granted without opposition, the concession cannot be forfeited in any other way. There is no condition compelling the concessionaire to prove that mineral exists, and no compulsory provision for its working, and whether the right to search is used or not it remains with the concessionaire against all other applicants.

A concession is granted to one person only; it is indivisible; only one person petitions and treats with the Government. He may sell his rights or transfer them entirely to a company, which will be recognised as an individual entity.

If the concessionaire only sells part of his rights, or leases them, the purchaser or lessee is not recognised by the Government. In such case it would be the duty of the original concessionaire to defend the rights over the whole of the property. The law requires all transfers to be entered in a public registry, where full particulars are kept, showing all titles to the mines, and whether a concessionaire has parted with his grant, or mortgaged it, or any part of it, and to whom.

The minimum acreage granted by the Government is a square 100 metres; there is no fixed maximum. Similarly there is no restriction in the depth to which minerals may be worked. A concessionaire is entitled to work all the minerals reserved from the ownership of the soil within the limits of his concession.

To obtain a concession a man has to deposit £3 up to ten *pertenencias*, and subsequently, a few francs per annum; so that, including about £2 for a stamp on getting the title, a mine of considerable size can be obtained for about £20.

One peculiarity is noticeable, viz., that the concession gives no grant of surface. The concessionaire has the right, however, of forcing the owner to sell such surface land as may be necessary for his work and offices, buildings, pits, machinery, and any roads he may wish to make within his claim, and so on. By law he must first attempt an amicable settlement with the landowner. If this fails he petitions the civil governor again in the same way as for his concession, but under the law of expropriation for the ground, sending with the petition a plan of the land that he proposes to take. The petition goes through certain formalities, and if the landowner does not oppose it, the concessionaire soon gets the land. If the landowner contests the claim it may take a long time, because valuers have to be appointed on both sides. If the valuers do not agree, the judge of the District Court appoints a third person to arbitrate, and his valuation is binding on both parties. Land bought at valuation costs something more than the agricultural value, and the Government add 3 per cent. at the end for expenses and taxation. The fees in connection with the valuation are very light.

The concessionaire has power to obtain wayleave facilities underground through another's concession, or on the surface over someone else's land, which power is similar to that exercised in the case of the occupation of the land. If an amicable arrangement cannot be made with the neighbouring mine owners, or surface owners, the concessionaire can ask for a decree of public utility. If the public utility of the passage for his mineral through another man's mine is greater than that of the mine through which it would pass, the right of passage is obtained, valuers are again appointed, and compensation given.

The workings cannot legally be carried within a distance of 40 metres of a building, but most of the concessions are on common land.

A miner is obliged by law to allow another miner to drain or ventilate through his mine, the principle of the law being to encourage the development of the mining industry, by giving every facility for the working of the mines when a concession has once been given.

In the case of injury to the surface through subsidence, the concessionaire must buy the land or give compensation. If a friendly settlement cannot be arrived at, he has power to purchase the land, by a certain process of the law.

As a concessionaire can so readily transfer his property to others, a large number of the mines are not worked by the concessionaires themselves. When leases are granted the terms are usually from 10 to 15 years and provision is made for the payment of an annual royalty of so much per ton, an annual minimum dead-rent, power to surrender the mine on its becoming exhausted; the exhaustion to be proved by a technical committee composed of mining engineers of the province who have to declare that the mineral remaining is not workable at a profit. Some leases have no terms granted for making up shorts, but usually from one to five years are given. Any form the lease may take is purely a matter of bargaining between the lessor and lessee, and the Government has no part whatever in such negotiations.

The State levies a special tax of two per cent. on the value of iron ore at the quarry or mine mouth, which is paid by the lessee, and is not recoverable from the concessionaire.

In France all the minerals except a small quantity reached from the surface by opencast, or which are of an alluvial character, belong to or are at the disposition of the State. Only the surface owner may search for them on his own land unless another searches with the owner's consent or with authorization from the Government. The search is generally limited to two years, and is prohibited within a certain distance of buildings and enclosed spaces. A surface owner must give notice to the prefect of the department before commencing open workings. In the case of a mine, however, neither he, nor any other, can work without first obtaining a concession, which is given to the discoverer or another applicant at the discretion of the Government. Geological and other considerations influence the Government in deciding the limits of the concession, which may be bounded by imaginary lines crossing several surface properties. Concessions vary in area from 8 to 17,442 hectares. A concession is indivisible and is granted in perpetuity for the mineral specified. It may be forfeited under certain circumstances. Two or more concessions may be amalgamated by consent of the State. Land outside the concession is obtained for a railway, &c., on the plea of public utility; within by paying compensation. No underground wayleave questions arise. The rents paid by the concessionaire are a fixed and proportional royalty to the surface owner, which are heavy in the St. Etienne coal-field, and a fixed and proportional royalty to the State. The State fixed royalty is ·10 franc per hectare, payable annually whether the concession is worked or not, and the proportional royalty, including an extra tax for special purposes, is 5·5 per cent. on what is practically the profits of the mine.

In Belgium and Germany the mineral law is for the most part the same as that of France, but in Belgium the owner or combined owners of the surface, has a preferential right, subject to satisfying the claims of the discoverer, to a concession. In Germany the concession is granted to the discoverer of the minerals. Here the royalties paid to the Government are 2 per cent. on the annual profits obtained from the coal sold.

In the United States of America a broad distinction exists between public lands which belong to the Republic and are under the care of the Central Government, and private lands held by individual owners. If this distinction is not kept clearly in view, confusion and misunderstanding will arise. Lands which were private property when the Federal Government acquired its general rights are subject not to that Government, but to the laws of the State in which they are situated.

The United States Government possesses all public lands, some being in the States and some in Territories which have not yet been formed into States. Concessions have been made to the older States, by liberal grants of land sometimes comprising all the public lands within their area. But this course has not been adopted with the newer and less thickly populated States, where the Federal Government retains possession of most of the public lands.

No sovereignty is claimed by the central Government in the minerals. Before the Declaration of Independence (July 4, 1776) the thirteen original States were mostly colonies of Great Britain, and had received by royal grant, in one form or another, a delegated sovereignty in the minerals. The rights conceded by the English Crown in the original grants to the colonies were subject to certain royalties payable to the Crown. The colonies were virtually sub-lessees under the sovereignty grant. When their independence was declared and recognised by treaty Sept. 13, 1783, the colonies succeeded to all the rights of sovereignty which Great Britain previously enjoyed. Any rights not afterwards ceded by the original States to the Federal Government were reserved to the States, and as the right of sovereignty in the minerals was not so ceded, it has been claimed since that time by the original States, and although the laws authorising it are mostly dead letters the right has to some extent been exercised. Great acquisitions of territory have taken place since 1783. Large portions of this territory, in which the original thirteen States had conflicting claims, were presented by them absolutely to the central Government, including of course all sovereignty in the minerals. These lands were at the time occupied by Indians and their forests infested with wild beasts. After incorporation with the United States, Congress authorised the Secretary of the Treasury to lease lead and copper mines, and for forty years the experiment was tried, the copper mines being situated in what is now the State of Michigan. These experiments proved unsuccessful, and in the year 1847 an Act was passed by Congress, abandoning this practice and authorizing the sale of mineral land, the surface and all beneath being sold together.

Further extensions of territory followed at the end of the war with Mexico by purchase under the treaty then made. California, Arizona, and New Mexico were added. The Mexican law was the old Spanish law under which the Government never alienated the ownership of any of the precious metals. Any grants made previously by Mexico to private persons were simply for agricultural purposes. These grants were of large extent and defined by natural boundaries. The discovery of gold in California followed closely on the Mexican war, and the "gold fever" spread rapidly, bringing a large influx of population from all parts of the world. Previously mining had not been carried on in California, so that the Mexican mining laws had obtained no footing there, and the comparatively insignificant original population of California had little influence in framing the laws and customs. A great deal of trouble followed through the grants having been defined by natural boundaries and not actual survey; this gave rise to fraud, as the natural boundaries could be so regarded as to exclude or include certain gold lands and gulches. The Government had the grants tested, the boundaries properly fixed by survey, under the authority of Congress, after which regular patents were granted for those areas constituting them private property. These patents were deeds of fee simple, and although they contained no clause as to reservations, it became a question whether they were subject to the reservation contained in the original grant from Mexico. In cases which were brought before the High Court of California, and afterwards carried to the Supreme Court of the United States, it was decided that every grant of land from the United States Government includes all that is on and under the soil, unless otherwise specified.

Since the unsuccessful experiment of leasing lead and copper mines in the Mississippi valley, the Federal Government has simply fixed the terms of sale for its mineral lands. and legalised certain peculiar methods of mineral searches and working on public lands without purchase. This legislation arose from the peculiar circumstances following the discovery of gold and silver in 1847 in California. The country was over-run by large numbers of adventurers, and was without any court, or resident Government officers, and without means of communication with the central authority by railroad or telegraph. Mass meetings of the miners were held and laws agreed upon and in this way order was maintained. This, however, did not alter the fact that they were all trespassers on the public lands of the United States Government, a state of things in which the Government itself acquiesced, so that its rights were in abeyance. This continued from 1847 to 1866. In 1847 possessory titles in the public domain of the United States sprang into existence. A statute was then passed embodying the peculiar principle that so long as the Government rights remained in abeyance, all subordinate rights should be dealt with as if there were no Government right. Any mining operations on public land were regulated by local custom or law, and until 1866 it was held in cases of disputed trespass both by Local and Superior Courts, that until the United States superior title was asserted the occupier who conformed to local law had all the remedies of a fee-simple owner.

The Federal Mining Act was passed in 1866, legalising all operations on public lands according to local customs, and also providing for the sale of the mines with a patent.

The advantage of purchase from the Government over a possessory title is obvious, and is accentuated in the case of mines becoming very valuable or requiring large capital for development. In this case a capitalist might well hesitate to invest his money in land held only on a possessory title, because this was entirely dependent on local law which might be passed one day and repealed the next: whereas a Government title would afford absolute security.

Other circumstances tended to the same conclusion. Occasionally the raids of Indians would necessitate a stampede of the miners, who, in the hope of a safe return, passed a stay-law, which prevented the appropriation of their property by others. Under this law districts have been locked up for a considerable time, and when re-opened conflicting claims have arisen, owing to the fact that some owners have not returned with the rest, and their mines have been claimed by others.

In 1872 an Act was passed by which it became still more advantageous to take title by patent. That Act provides for a permanent and accessible official record of the boundaries of properties, an immense advantage to the owners and others. The same Act makes the sale of mineral lands as simple as in the case of agricultural land, with one important exception. The grant of a mineral location which is made upon the edge of a lode or vein differs from the absolute title of the fee simple, inasmuch as the grantee has an extra lateral right. The two end lines of his claim must be parallel, and he has no right to follow the vein beyond vertical planes drawn through those two lines, but with regard to the lateral boundaries, the grantee has a right to follow outside these to any depth whatever vein may be outcropping within that surface. He is unable, however, to trespass upon his neighbour's surface, and is liable in damages if he injures it. Every grantee is of course subject to the exercise of the same lateral rights on the part of his neighbour. These lateral rights are referred to as lode claims.

A placer mining location or a tract located for alluvial mining, and granted by the Government, has its boundaries defined on all sides by vertical lines drawn through the boundaries of the surface. The land is sold as mineral land out-and-out in the case of placer claims, or with the peculiarity in the class known as lode claims of having the lateral right of the grantee or patentee added to his fee simple, and the lateral right of his possible neighbour subtracted therefrom.

Coal lands on the public domain are sold by the Government by the acre, surface and mineral going together; the price if situated within 15 miles of a railway is 20 dollars per acre, if beyond that distance 10 dollars per acre. The Government will sell the land on the completion of its surveys, but a man may enter upon possession beforehand, with preferential right of purchase. It claims no seigniorage or royalty.

The Act of 1872 to a great extent extinguished the vagaries of local customs by laying down regulations to be observed by local Governments. One clause insists on the performance of a certain amount of work annually on every mine held by a possessory title. The discoverer of a vein on the public land locates it, and if no one else has a prior claim he marks out the area of the land he wishes to hold. By doing a certain amount of work every year upon that claim, he for an indefinite period may retain possession of it without purchasing. The amount of work must not be less than that prescribed by the Government; more may be required if the local customs happen to be onerous.

The Government having once sold mineral land does not afterwards hamper its working. No law regulates the method of working or the payment of taxes, or interferes with the freedom of the owner of the property to use his surface or underground right as he may think fit. He is in the same position as the owner of land under English law. One very important provision, however, has been made by the Government with respect to wayleaves on public lands. First, there is absolute freedom of way across the public land for mining ditches in which water is conveyed, and also for mining roads, tunnels, or anything necessary to mining operations. Secondly, there is a right-of-way easement, which does not include the freehold. In this case compensation for the land is determined as in other cases where private property is acquired by the State for public use. If A. is carrying on mining operations and can only get out by crossing the claim of an adjoining neighbour B., he may do so whether B. is willing or not. If, however, the crossing is done so as to injure B.'s mining operations, a jury assesses the damages, or the courts take charge of the case. The Government makes mining claims and lands subject to all rights of way already existing over the public land, and also subject to rights of way, easement, or drainage enacted by the local Legislature; these are invariable reservations from the grant. If an easement is obtained by legal process compensation will be accorded in proportion to the damage sustained and not in proportion to the benefit accruing to the party seeking the right of way.

All questions affecting the safety of miners and other local regulations are left to the individual States.

On a mine being developed and proving valuable the high price which it will command in the market is equivalent to a capitalised royalty. Of this Government of course receives no portion There are cases on the public lands where a regular tonnage or per centage royalty is paid. This may arise from the first adventurers having lost money; the mine may have proved poor, even if rich for a time. The owners not being able or willing to find sufficient funds to carry it on, let it on lease at a royalty rent.

In the Eastern States there are no laws to govern wayleaves or easements between neighbouring owners; questions relating to these are settled by common law, on the basis of freedom of contract. In these States two classes of iron and coal mines exist, viz., those worked by proprietors who pay no royalty, and those worked upon a system of mining leases under royalty, the grantor being the owner of both the surface and the minerals, who does not wish to be burdened with the management both of the land and the coal working. The system of leases is very similar to the English. The leases in the case of iron mines are for terms of 20 years or upwards, and in coal mines usually from 25 to 50 years and occasionally 99. They are subject to dead or certain rent with power to make up short workings for any year throughout the term. The powers of renewal at the expiration of the term, wayleave rights, and right of sale or of re-letting are matters of arrangement between the parties. The royalties are sometimes paid according to a sliding scale which is regulated by the selling price of the coal.

It is not customary for the proprietors to provide houses for their workmen rent free, though a lower rent is usual to their workmen than would be the case to others.

The coal mines of the country are divided into three classes, viz., the anthracite, the bituminous coking coal, and a third class consisting of all other coals. The anthracite and the bituminous coals are subject to royalties which vary in proportion to the value of the coal yielded, and the facilities for working it. The lignites are found for the most part in the Rocky Mountains, and are not subject to royalty rents. They are worked by adventurers who take up the land, which is sold cheaply by the Government.

In some States under the general railway laws a right of way across intervening lands may be obtained by proceedings in condemnation if amicable arrangements fail. In this case damages are assessed on the basis of injury done, and not of benefit accruing to the promoters. The roads authorised by condemnation are always assumed to be public carriers; they must carry the freights brought to them. Private wayleaves are obtained by negotiation and in almost all cases by the payment of a lump sum.

The Government reserves all public lands officially reported to contain minerals. Other land is sold to agricultural applicants at from \$1¼ to \$2½ per acre. Mineral lands are sold at \$2½ to \$5. Coal lands are sold at \$20 per acre if within 15 miles of a railway, and \$10 per acre otherwise. No agricultural claim can be made on mineral land, but if a tract should be purchased as agricultural land and mineral be afterwards unexpectedly found on it, the agricultural claimant is entitled to the mineral interest. He becomes absolute owner and may work the mineral himself, lease it to others, or keep it locked up. No agricultural land is sold by the United States Government as a rule, until the surveys are completed, but in the case of mineral land that survey is not made until an application is received from a purchaser, who is charged by the Government officials with the cost of the official survey. This forms the principal expense in acquiring the land, the price per acre being so small. In addition to this the Government requires proof of an expenditure of 500 dollars under the possessory title before it will complete the sale, as an indication of the discovery of a lode. The maximum size of a mining claim is about 20 acres, 2 roods, and 26 perches, being 1,500 feet by 600. Placer locations are more extensive, running up to 120 acres. A small capital, therefore, is sufficient to purchase a mining claim.

In the colony of Newfoundland, the minerals belong to the Crown. Any person may search without licence, and for a fee of \$50 may obtain a temporary lease, which gives the exclusive right of further search for a year over one square mile. If the search proves successful within that year a further lease is granted for a period of 5 years, the only condition being that a specified sum per annum shall be expended in *bonâ fide* prospecting or operating the mine. The aggregate amount is $6,000, being $800 during each of the first four years, and $2,800 in the fifth year. Failing this, the lease is forfeited and the land reverts to the Crown. These conditions being fulfilled, a grant is made in fee of the minerals (excepting gold which is reserved to the Crown) contained within the one square mile, and also a freehold grant of 50 acres within the same area, the Crown reserving all surface rights over the remainder. When the property is granted in fee to an individual, it remains his whether he works the minerals or not, and he has the same right as a freeholder in this country. The Government reservation of the greater portion of the surface does not take effect until the purchaser has had ample time to complete his search and verify his discoveries. The greater portion of the island is not yet colonized, and all the minerals known are in unoccupied land.

When a grant is made the minerals vest absolutely in the grantee, who has the power to sell or let. Copper is the chief mineral worked, though lead and silver are also mined. Coal has been found, but is not worked owing to difficulties in that locality respecting the French treaty rights, which make the title doubtful.

No local or imperial taxes are levied on the mines, the principle of the legislation being to encourage the investment of capital in the mineral wealth of the colony.

In British Columbia, which now includes Vancouver's Island, there is a royalty charged on Crown lands of five cents (about $2\frac{1}{2}$*d.*) per ton upon coal raised. The coal belongs to the surface owner, subject to this royalty. Minerals other than coal are searched for by free miners who take up claims. The largest claim measures 1,500 feet by 600. Within his claim-limits all veins and lodes belong to the miner holding the licence. He may obtain a Crown grant to include all minerals within the boundaries of his concession, and have power to run drains through unoccupied mining lands subject to paying compensation.

In the colony of New South Wales, the Government possesses all unlet minerals, but a great many of those which have been worked belong to private owners. The grants made before 1861, under Orders in Council, contained various reservations, and were not of a uniform character; some contained a reservation of timber for bridges, some of roads, and some of all minerals. About 1854 the Governor issued a proclamation cancelling the reservation of minerals. A large company called the A. A. Company, having a grant of a million acres, thus became possessed of very valuable coal seams and other minerals which existed under a considerable portion, though not the whole of their grant. This property forms a part of the Newcastle coalfield, the most valuable in the colony. In a Land Act passed in 1861, two conditions were laid down for granting lands, (1) required the payment of £1 per acre and residence for a specified term of years, but conveyed no interest in the minerals; (2) the other, under the Mineral Conditional Purchase Clause, provided that on payment of £2 per acre, and expenditure of £2 per acre in working minerals other than gold, the land should be granted without reservation of the minerals. This clause did not long remain in force, as it was prejudicial to the public interest. Under its operation a number of private people obtained possession of mineral lands.

In 1884 the Crown Lands Act was passed, by which all grants of land for settlement contain a reservation of the minerals to the Crown, the interest of grantees being restricted entirely to the surface. By it the Crown also reserves all rights to give authority to work the minerals. "Minerals" according to the Act, "include coal, kerosene, shale, and any of the following metals or any ore containing them, viz., gold, silver, copper, tin, iron, antimony, cinnabar, galena, nickel, cobalt, platinum, bismuth, and manganese, and any other substance which may from time to time be declared a mineral within the meaning of this Act by proclamation of the Governor published in the Gazette."

Owners of the soil, including the minerals, are much in the position of a private owner in England, having power to sell or lease the coal. Companies have been formed to take advantage of such powers. Where the coal is leased, a tonnage royalty is paid, and a minimum annual dead-rent. The properties are usually large, and seldom give rise to questions of wayleave. The law of subsidence of the surface is the same as in England, and as some of the seams are very thick their working in some places causes injury to buildings, &c., more especially as the pits are generally shallow. The local taxes are paid by the lessees. On the coal becoming exhausted or unworkable to profit there is power to surrender the lease, the question of its being unworkable to profit being determined by arbitration. The lessor has usually power at the termination of the lease to purchase the fixed machinery and rolling stock at a valuation.

The Crown Lands Act of 1884 divides the surface and the minerals into two departments, one the Crown Lands Department, dealing with the surface settlement, the other, the Mines Department, with the mining. It provides that "The Governor shall, notwithstanding the provisions of the Mining Act of 1874, impose a royalty of not less than 6*d.* per ton on coal raised from land which may be hereafter leased; and such royalty shall be in addition to or in substitution of any rent payable by such lessee under the said Act, but shall not affect or prejudice any other condition of the lease."

It is in the discretion of the Minister for Mines whether in addition to this royalty the rent of the land shall be paid or not. The object of this discretionary power to charge royalty and rent is to prevent the taking up of large tracts of land for trading purposes. If a lessee does not work the minerals the Government can insist on payment of the yearly rental. If the land is of a rough description, and the lessee undertakes to spend money for its improvement, the royalty alone is required. The rent is £1 per acre per annum for gold-bearing lands, and for other minerals, 5*s.* per acre. The Crown does not work the minerals, but gives the right to private persons under certain rules and regulations. A man may by paying £1 obtain a miner's right, and for another £1, a mineral licence. Provided with these, he can enter upon any Crown lands in the colony to prospect. If successful, he can make application for a lease, stating what minerals he wishes to work. He is permitted to continue operations during the official survey, and in three, six, or nine months, he receives his lease, and as long as he complies with its conditions, retains absolute possession of the minerals. If there are two or more applications, the Mines Minister deals with them as he thinks right. In the case of a gold lease, the area is from 1 to 25 acres for reefing gold, but much smaller areas for alluvial gold. In the case of any other mineral the largest grant is for 640 acres. More than one block is seldom granted to the same person, the desire being to prevent the taking of land for speculative purposes. Any number of persons may apply collectively and have a corresponding acreage allotted them; thus six may have 3,840 acres.

The lease is for a term of 15 years, with the right of renewal for 15 years, the intention being to extend the lease indefinitely so long as its conditions are fulfilled. No compensation is allowed to the lessee for his expenditure of capital at the termination of the lease. A lessee has the right to sell his interests, subject, of course, to the payment of royalty due to the Crown.

No royalty is charged except on gold and coal; for other minerals a fixed rent is charged.

At the expiration of the term of 15 years, the royalty of 6*d.* remains the same on renewal, but the rent of 5*s.* per acre on coal may be increased.

A Mines Regulation Act, passed in 1874, is in force throughout the colony, which applies equally to mines held under the Crown, and to those under private individuals. Government inspectors are appointed to enforce the operation of the Act.

The Crown reserves in its leases all easement rights which it grants in case of need, to the lessees of adjoining mines. Thus a lessee may obtain shaftage, way-leave, ventilation, or watercourse privileges, if the Mines Minister is satisfied with the urgency of the need. No lessee can dispose of his interest, or mortgage it, without the approval of the Crown.

The Crown retains the power at any time to take from the lessee land which is necessary for a public purpose.

In Crown leases of coal under sea or tidal water, the lessee is compelled to carry on his working by pillar and stall method in order to leave support for the surface.

Forfeiture of the lease may result from breach of covenant, arrears of rent, or using the land for purposes other than those specified in the lease.

If an outlet is required from a mine to a port or railway station, a private bill which provides for payment of all proper claims, is passed by Parliament on the plea of public utility.

Crown lands conceded with full mineral rights, may be resumed by the Crown on certain conditions somewhat similar to those below mentioned, prevailing in South Australia and New Zealand.

In Queensland, the Government has power to mark off mining districts, within which licences are granted for working all minerals other than gold. The licences are for 21 years, at a rent of 10*s.* an acre. Under the Coal Mining Law of 1886, a licence is granted for one year, allotting land up to 640 acres at 6*d.* per acre, within which a search may be made for coal. This is renewable on the same terms for another year. If the licensee discovers coal, he may obtain a lease for 320 acres at a rent of 6*d.* per acre, and a royalty of 3*d.* per ton of coal raised during the first ten years, and then 6*d.* per ton on the output for the remainder of the lease. A law of 1881 regulates labour, ventilation, shaftage, fencing, machinery, inspection, and safety-lamps.

In South Australia, leases for minerals, other than gold, are granted by the Crown over an area of 80 acres (formerly 640 acres) for a term of 99 years at 1*s.* per acre, and 2½ per cent. royalty on the net profits. Licences to search are also granted preparatory to a lease as in Queensland. In the Northern Territory a different law prevails to suit local conditions. There, by a law of 1888, private lands may be resumed for mining purposes by the Government provisionally for six months; after that period the resumption ceases, unless it is previously made absolute. If absolute, the owner receives payment of the purchase money and compensation for the loss of his land, but the price paid does not include anything for the minerals. Royalties received by the Crown from the workers of those mines are paid over to those who, but for the resumption, would be exercising the mining rights, less a commission of 2½ per cent. retained by the Government.

In Victoria, there is a Mining Act of 1890. Here, as in Queensland and South Australia, a licence to search upon Crown lands for minerals other than gold is granted. For the purposes of cutting races, making dams and reservoirs, and diverting waters, licences are also given on Crown lands and on lands leased to private persons, compensation being paid. Mineral leases on Crown lands are granted for 30 years to the extent of 640 acres with a right to cut races and make dams and reservoirs. Rents and royalties are fixed by the Governor. The Mines Act lays down regulations for labour and machinery, and the safety of miners. If pumping machinery for general drainage is erected by one person, and neighbouring mines are benefited, the owners must contribute to the cost.

In Western Australia, the regulations are approved by the British Colonial Secretary, so that they are not under a local Act. Licences to search for minerals are granted. Leases for 7 years are given of not less than 20 or more than 200 acres, and upon these, Crown grants may follow for not less than 20 acres if the grantee has erected plant and other machinery and carried out improvements to the value of £3 per acre. By a new regulation of 1890, the owners of land, beneath which the minerals are reserved to the Crown, may obtain a permit to mine on payment of a royalty of *6d.* per ton on coal, *2s.* per ounce on gold, and on other minerals 2½ per cent. on the value at the pit's mouth.

In New Zealand, there is a separate law for coal mines. On Crown lands leases are granted for 30 years upon 640 acres, at a dead-rent of from *1s.* to *5s.* per acre, with a royalty of from *3d.* to *1s.* per ton. The term may be extended to 99 years. Similar regulations prevail as to safety and inspection to those in Victoria. Another law of 1886 refers to other minerals. As in South Australia, land which has been alienated by the Crown, may be resumed for mining purposes, including any which may be required for water-races, dams, watercourses, &c. The owner receives full compensation for the land and also for any mineral value. Mining districts are formed within which, and even beyond, licences to search are granted. Mining leases for 21 years are given at *10s.* per acre. Power to grant licences for water-races, dams, and reservoirs, upon private as well as Crown lands, may be given by the Mining Warden, subject to proper notice in the case of a private owner. Compensation must be paid him before the work is commenced.

In Tasmania, mineral leases are granted for 21 years of areas not less than 20 acres. Coal, shale, limestone, slate, or freestone leases are for an area not exceeding 320 acres, and for other minerals not exceeding 80 acres. The rent payable for coal and limestone is not less than *2s. 6d.* per acre, and for other minerals not less than *5s.* per acre. Races or other necessary works may be made, or water taken from streams belonging to private owners on payment of compensation. The Settled Land Act of England, 1882, is adopted, whereby a tenant for life of settled land may grant mining leases and put into operation the other powers of mining conferred by the English Act.

The law in India with respect to the ownership of mineral property differs in different parts of the country. In what are called the permanently settled districts—viz. parts of Bengal, the North-west Provinces and Madras—the minerals belong to the surface owner who pays a moderate fixed land tax, the amount of which was settled in 1793, with an undertaking that it should never be increased.

With the exception of these permanently settled tracts and a portion of Madras occupied by petty proprietors, the minerals in all other parts of British Territory belong to the State.

The petty proprietors referred to own the surface and the minerals beneath, but they are liable to pay an extra land tax of five rupees an acre per annum on any land on which they work minerals.

By the ancient law in Native States, the Native Rajah is recognised as owner of the minerals.

The highest royalty which has been paid on State lands for coal is four annas per ton (the anna being $\frac{1}{16}$th of the rupee, the nominal worth of which is *2s.*)

The Government gives a short term rent-free during the opening of the mines. A small dead-rent of two rupees per acre is generally charged, but not as an addition to the royalty. A little coal is raised in Burmah on Government concessions, but the principal coal-pits are worked on private property.

Private owners when not working the coal usually sell their lands outright to colliery proprietors, or give a practically perpetual lease of the land, including the minerals, at a dead-rent per acre, but subject to no royalty charge. The Serampore coalfield, belonging to the East India Railway Company, is so leased with rights of transfer, the charge being ten rupees an acre. A premium of ten years' rent was also charged on the grant of that lease. The Bengal Coal Co., the largest private colliery proprietors in Bengal, bought their land and minerals in fee simple.

The Karharbari coalfield belonged to a Native Chief; under the expropriation law (which enables the Crown to take up land for public purposes) the Government of India took it up and paid a full price of £95,000 for it. It was then let at a dead-rent equal to 5 per cent. on the cost to the East India Railway Co., which was then guaranteed by Government, but is now Crown property.

There are only two cases of coal being worked in the Native States, viz. Umaria in the Rewa territory, and Singareni, in the Nizam's territory. Umaria is being worked by the Government with a view to making it over subsequently to a private company. It pays the Rewa treasury on behalf of the Rajah, who is a minor, a royalty of eight annas per ton, which is double the highest royalty charged by the Government on its own lands. In Singareni the Nizam's Government agreed with a private company who were to pay a royalty of eight annas per ton, for any output to 100,000 tons in the year, on any surplus quantity a royalty of one rupee per ton. This royalty is charged on all the output, whereas the Government royalty is usually charged only on large coal.

Most of the coal in India is raised from shallow shafts not exceeding 100 feet, the deepest being about 300 feet. Some coal is worked at the outcrop like quarries.

The following table shows the number of collieries which were working in India in 1891 :—

In Bengal.	In Madras.	In Punjab.	In the Central Provinces	In Assam.	In Central India.	In the Nizam's territory.	In Beluchistan.	Total.
77	1	1	2	3	1	1	1	87

Their total output amounted to 2,328,577 tons. In 1881 the annual output from Indian collieries was 997,730 tons.

An adventurer wishing to obtain a concession in a known coalfield belonging to the State would be required to prove his financial ability to work the coal properly, and would have to pay the cost of a survey unless the coalfield had already been surveyed.

The terms on which concessions are made, vary according to circumstances. In an altogether untried coalfield at a distance from a railway or navigable river, a larger area, and a longer term free of rent would be given. There is a reluctance on the part of the Government to grant a monopoly of any coalfield to an individual or to a company. The competition is keener in proved and accessible coalfields and much smaller areas are granted. The ordinary terms of a concession provide for a dead-rent of two rupees an acre, which merges in a royalty of four annas per ton of marketable coal. No royalty to be charged for the first two years of the term. Proper books showing the output must be kept by the concessionaire and produced for inspection. The grant is for either 20 or 30 years, and is subject to withdrawal after two years, if a specified output is not reached, or at any subsequent time if it is not maintained. The concessionaire is bound to comply with the Government's requiremcnts as to plans and method of working. Provision may be made to prevent any claim by the concessionaire for damages resulting from any subsequent alteration in thc mining laws. There is no specified power of renewal, but in the event of non-renewal the concessionaire is allowed a given time in which to sell or otherwise dispose of his plant. There is no power of subletting or transferring the concession without the consent of the Government; such consent is, however, usually given in case the transferee is a British subject having means to carry on the mine. Where the surface is waste land and in the hands of the Government, complete power is given to the concessionaire over the surface, but where it is occupied or subject to easements, the concessionaire must pay all damages resulting from his works to the surface occupier or owner. In order to convey the produce of his mine over land belonging to some one else, the concessionaire may arrange the terms of compensation amicably by private negotiation with the surface owner. Failing this, the Government may, if necessary, take action under the expropriation law which would throw the full compensation awarded under the law upon the concessionaire. Ordinary cesses or rates are levied on coal mines for local purposes; neither of the two cesses at present levied in Bengal may exceed one-half anna in the rupee, or about 3 per cent. of the net annual profits of the mine.

The Government are considering the passing of a law for the regulation and inspection of mines.

There is no fixed rule as to terms of concessions or as to the rights given to a concessionaire, so that intending applicants are at a disadvantage as compared with those in European countries, where full information is given in regard to these matters.

CHAPTER IV.

SHAFT-SINKING.

Forms of Shafts—Mode of keeping them truly Vertical during the Sinking—Circumstances calling for consideration in Selecting their Sites—The Tools and Appliances used in Sinking—Timbering as it proceeds—Walling—Tubbing—Machine Drills to Expedite Sinking—Piling through Quicksands—Sinking through Quicksands by hollow Cylinders of Cast-iron—Poetsch's Freezing System of Sinking through Quicksands—Gobert System—The Kind-Chaudron System of Sinking—Explosives used for Blasting in Sinking—Deep Shafts.

SHAFTS may be either rectangular, circular, polygonal or elliptical. In the rectangular and polygonal forms, care must be taken to suspend a plumb line from each angle during sinking. In a circular shaft four plumb-lines should be suspended at the extremities of the two diameters, crossing each other at right angles, and in an elliptical shaft four plumb-lines should also be suspended at the extremities of the major and minor axes. Constant care will be required with this plumbing to keep the shaft truly vertical. The circular is the most secure form, though there may be reasons, such as putting in pumps, why an elliptical or rectangular form may be desirable.

The size of the shaft is determined by a careful consideration of all the circumstances. Before the introduction of boring with the diamond drill, trial shafts were frequently sunk in order to prove the strata. These were made about 4 feet in diameter, which just allowed space for two or three sinkers to work in them at the same time, and room for winding and pumping. Trial-shafts are now seldom sunk, as the preliminary borings, which may be of large size, convey full information of the strata. Before the sinking of shafts to form a colliery winning is commenced, the terms of the lease, or the nature of the holding and the extent of the taking are known. The thickness of the seams proposed to be worked, taken in conjunction with the area of the property, form the basis of calculations of the probable yield of minerals. This yield, divided by the number of years for which the lease is granted and by the probable number of working days in a year, gives the average daily output which should be aimed at. These calculations are good as far as they go, but they are generally upset by subsequent events, such as strikes or lock-outs, the taking of additional conterminous royalties, or the working of thinner seams not at first contemplated. Nevertheless it is usual to make them. The fewer the number of shafts meeting the required conditions of output and ventilation, the more economical is the working of the coal, and if it is practicable to deal with the desired output with only one pair of shafts, no more should be sunk on a taking of ordinary size, especially if the depth to the seams of coal is great.

To comply with the Coal Mines Regulation Act, 1887, the minimum number of shafts is two, and these must not be nearer to each other than 15 yards. Otherwise they may be placed where most convenient, either at a long distance apart or close together to form companion shafts. The latter arrangement is generally more convenient and less costly, because the surface works are concentrated in one establishment, and supervision is so much the easier. Moreover, if the leading shaft be to the dip, the sinking of it usually drains the strata to the rise, so that the second shaft may be sunk comparatively dry. Should the continuity of the strata between them be broken, pumping may still be confined to one shaft by boring a hole downward from the bottom of the second shaft, and connecting it by means of a drift with the leading shaft at different points as the sinking proceeds.

The lower strata of a deep colliery are mostly dry, and it is frequently found that after the shafts are tubbed through the water-bearing strata no further pumping is required. If tubbing is not resorted to, in consequence of seams too near the water-bearing strata having to be worked, the pumping may be more advantageously met by a third or pumping shaft sunk only through the water-bearing strata, and used solely for raising water.

Shafts must be fairly large where a daily output of 1,000 tons is to be landed, and if much more than this is aimed at, and the winding is from a great depth, possibly more than two shafts may have to be sunk on the property. If the sales are assured, a large output is more profitable than a small one: an output below a certain point results in the colliery working at a loss instead of at a profit. Very large outputs from a single shaft are only possible when there are the natural advantages of thick seams which can be easily worked, and then the shaft must be of large area, and, if deep, must be well equipped for rapid winding with cages constructed to take large trams. If a mechanical ventilator be employed both the downcast and the upcast shafts may with advantage be used for winding from a different seam of coal. Either of the shafts may be used for pumping water, but if only one shaft is used to wind, the pumps should be placed in the other, so that accidents to winding or to pumping appliances may interfere as little as possible with the operations in the other shaft.

Very large takings are pierced at different points by such shafts as are thought necessary by the engineer in charge, who allots to each pair of shafts the area that can be most economically dealt with, having regard to the exhaustion of the coal within the period of the lease, the commercial value, and possibly the divisions made by large faults or by old workings.

In South Wales, where favourable conditions prevail, a daily output of 1,000 tons is in many instances obtained, and in rare ones even 1,500 tons have been raised up a single shaft. Here the thickness of the seams allows of the use of large trams, having a carrying capacity of from 30 cwt. to 2 tons. Single-decked cages are for the most part used so as to facilitate the changing of the trams in them. In order to receive two trams ranged one behind the other the cage must be of great length. This of course necessitates large shafts, which are frequently from 18 to 20 feet in diameter in the clear. These large shafts also admit of the workings being well ventilated, which is a matter of great importance in the working of fiery seams, and requires to be kept in view as much as the question of output.

Where thin seams such as those at Radstock are worked, it is found impossible to raise large quantities of coal. In that case faults add to the difficulties of working these seams, and shafts rarely if ever exceed 10 feet in diameter. Such shafts under the prevailing circumstances meet all the demands made on them, whether for winding, pumping, or ventilation.

Deep shafts frequently take years to sink, but larger shafts, although more expensive, are not usually longer in sinking, because of the larger number of men and machinery that can be employed. It may safely be assumed that shafts varying from 10 to 20 feet in diameter will meet all the requirements of the different districts in this country.

In all new winnings two shafts are required by law, and their site is a matter requiring much thought. Other things being equal, that site which gives the largest amount of the field to be worked to the rise is preferable. The underground conveyance is thus rendered easier, and water in the workings will naturally gravitate to the shafts. If the seams of coal to be worked are lying in a horizontal or nearly horizontal position, there is an advantage in placing the main shaft in the centre of a property of moderate size so as to command an equal area of working in each direction. The most distant point intended to be worked from one shaft must not be so far away as to render the underground conveyance difficult. Underground considerations usually give place to those of the surface, although this may entail pumping water out of dip workings, and hauling large quantities of coal up hill, and the sinking of shafts on the rise side of large faults. As a rule, the quantity of water obtained in the workings of deep collieries is very slight, and not likely to be troublesome in proceeding to the dip. The evidence obtained with respect to the dip of the seams and the direction and size of faults in the neighbourhood, and the vicinity of old workings, is often very unsatisfactory, whereas the surface conditions are of course well known. These, therefore, form a trustworthy guide to the mining engineer, who will leave the underground difficulties to be dealt with as they arise. If, however, the position which is best from a surface point of view, does not accord with that suggested by what is known of the mineral conditions, then the site must be chosen after a careful balancing of probable advantages and disadvantages. Neglect of this may afterwards cause greater working costs which no future engineering skill can rectify. A site which will suit a communication with the railway, tramway, canal, or whatever means of transport there is for the coal, is desirable. The necessity for an ample supply of pure water for the boilers and for general purposes must be borne in mind, and as rubbish must be landed during the sinking of the shafts, and possibly afterwards when the coal is worked, a convenient place must be arranged to tip it into, and the right secured to form a spoil bank. Provision must also be made for the disposal of water from the mine. If more than one royalty be leased, the winning of the different royalties, and the question of wayleave and outstroke, may affect the position chosen. It will be only after well considering and balancing these matters that a proper conclusion as to the site can be arrived at.

The tools used in sinking are *hacks*, or *picks*, for loosening the rocks and chipping back the sides. *Shovels* for filling the loosened rock into the kibble. *Wedges* for forcing out the rock by driving them into the joints. *Sledges* are heavy hammers with long handles for use with both hands in driving in the drill, breaking up large pieces of rock, or for striking the wedges. A *hammer* has a short handle, is much lighter than the sledge, and is used with one hand for hitting the head of a drill held in the other. *Drills* are usually made of cast steel, and are bars of different lengths having a cutting and a striking end. A *jumper* is a long drill used with two hands, and without a striker; the operator raising it and letting it fall with considerable force in the hole he is drilling. A *scraper* is a tool for removing the dirt which accumulates in a bore-hole while being drilled. A *swab-stick* is a deal rod bruised at one end, sometimes used for cleaning out the drill-hole. The *bull* is a round bar of iron, with an eye in it, for forcing clay into the interstices of the rock to keep water out of the bore-hole. *Stemmers* or *rammers* for tamping the bore-hole after the charge has been put in. *Cartridges* are cases containing the explosive compound to be inserted in the bore-hole. The *fuse* is the means by which the cartridge in the bore-hole is fired. It must allow time after being fired and before reaching the cartridge for the men to be drawn away. *Kibbles*, *barrels*, *hoppits*, *buckets*, or *bowks* are large iron barrel-shaped receptacles for the transference of the loosened rock from the pit bottom to the surface. The sinkers also "ride" on them.

Fig. 19 (B) shows the usual form of mining tipping kibble. It is self-righting, and the catch can be made self-acting or otherwise as desired. The strong iron bow is attached to two side trunnions placed a little below the centre of gravity on the outside of the kibble to allow of easy tipping. Special care must be taken to provide a proper means of fastening the bow of the kibble, and to keep it in perfect working order. When the spring catch gripping the bow is pulled back, the kibble is free to swing round on the bow, a light pull or push only being necessary for the discharge of the contents which takes place whilst the kibble remains suspended. The kibble is made of wrought-iron, and the catch prevents it from tilting during the winding.

A kibble not intended to have its contents discharged in this way, but for transport to a rubbish tip after reaching the surface, is made as shown in Fig. 19 (c), but without the valve, and sometimes with chain attachment to the sides, as in Fig. 19 (A). Care must be taken in loading the kibbles only to place the broken material to within a few inches of the top. This is a matter of supreme importance to the sinkers in the pit bottom, to whom the falling of an ounce or two of stone or other substance may mean serious injury or instant death. Tipping kibbles were formerly used for winding water, and were made with a smaller bow fixed to the sides as well as with the larger one. The contents were discharged on the surface by the banksman who released the catch and pulled the kibble over by means of the smaller bow. This kind of kibble is unsuitable for winding a large quantity of water, because it does not fill easily, the water having to flow into it over the top. It is sometimes used, however, when there is only a small amount of water in the shaft bottom. The water is then baled into the tipping kibble and wound to the surface. But baling is a tedious process, and therefore costly. Sometimes these tipping kibbles are made with an ordinary chain link placed so as to slide over one link of the bow and long enough to reach to a short vertical pin riveted to the inside of the kibble. Over this pin the link is placed to hold the kibble firmly in place. When the link is lifted the kibble turns over and empties itself. Tipping kibbles, whatever their form, need only to be disconnected from the rope at the bottom of the shaft. The other form of kibble must be taken from the rope and replaced by another at the surface as well as at the bottom of the shaft. The *water-barrel* is much the same as the kibble, but it has a valve in the bottom, and is used for sending the water to the surface, where the banksman pulls a handle communicating with the valve which allows the water to run out; or in another form of water-kibble the valve-spindle projects below the level of the kibble, so that when the engineman lowers it on to the runner the valve is opened by the action, and the water runs out.

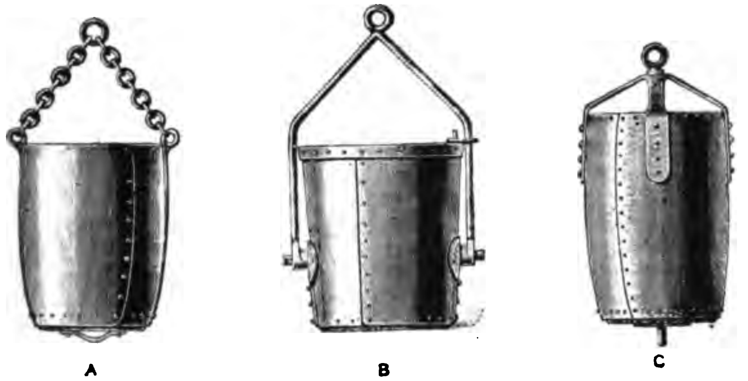

Fig. 19.—A, Kibble for Winding. B, Tipping Kibble or Bowk. C, Barrel for Winding Water. with Valve at Bottom.

Fig. 19 (c) shows a water-barrel which is made of wrought-iron with a water-tight valve in the bottom moved by a central spindle extending beneath to allow the water to run out. The barrel is lowered into the water in the shaft, and if this is of sufficient depth the barrel fills through the valve, which is lifted by the pressure of the water. When the barrel is full, the valve drops into its place and retains the water. The barrel is then raised to the surface, the travelling platform is pushed forward over the mouth of the shaft, and as the barrel is lowered on to it, the projection of the spindle presses lightly on the platform, whilst the kibble is kept in a vertical position by the engineman; this raises the valve and allows the water to be freely discharged, and shutes and pipes lead it away. The barrel is made to hold from 100 to 600 gallons of water. The ***spring hook*** is the means of attaching the kibble to the rope, and is shown in Fig. 20.

A pit to be finished 15 feet in diameter must be marked out 17 feet 6 inches to allow for timbering and walling. The sinking of a shaft is most frequently done by contract, and terms of agreement are drawn up between the contractor and manager, or owner, but sometimes the sinking is done on days' wages under the charge of a master-sinker. If more than thirty persons are employed in the whole of the shifts in the twenty-four hours there must be a certificated manager to comply with the Mines Act. The first few yards may be sunk by means of a windlass, after which a steam-engine of suitable size must be got. Two large balks of timber are placed across the pit, which may be afterwards timbered over, except a portion in the centre for the kibbles to pass up and down. On coming to the surface the engineman draws the kibble well up above the level of this opening, and the "banksman," or attendant, pushes the "runner" (which is a wide trolley-shaped carriage running on rails) into position, thereby covering the pit top and protecting the sinkers, whilst the engineman lowers the kibble on to it. It is then detached, an empty kibble which has been standing ready on the runner is attached to the rope, the signal given to the engineman to lift, and this being done the "runner" is pushed out with the full kibble on it, leaving the opening clear for the descent of the empty kibble. It is usual only to use one kibble in a sinking pit on account of the danger from collision which would arise if two kibbles were used. The travelling platform or runner is particularly useful for water-winding, because it receives the water discharged from the barrels and directs it to any desired channel, but it is not so convenient in use for the débris which is brought to the surface; for, although it forms an effectual cover over the shaft after the kibble reaches the surface, it is removed before the kibble descends, and the pit mouth is then left unprotected until the re-appearance of the kibble at the surface.

Two hinged counterbalanced doors are now sometimes used to close the shaft opening. These doors are hinged so that they lift upwards in two even divisions towards opposite sides, and when open they form a fence at two sides of the rectangular opening, the other two being guarded by fixed boarding. When the doors are closed they effectually seal the opening. Rails are fastened to their upper side to form a continuation of the tramway leading to the tipping ground.

In the ordinary arrangement these doors have to be lifted separately, but Mr. W. Galloway has designed an improvement by means of which both doors may be opened at the same time. This is effected by an arrangement of levers and counterbalances worked by a handle and connecting links. The handle may be moved wholly or partly back, the counterbalances retaining the doors in any position into which they are moved by the handle. If only débris is to be wound up the shaft, no travelling platform is required, but if water is wound it may be still usefully employed to receive the barrel after the trap doors are closed, or it may be preferable to discharge the water as well as the material into tipping-waggons.

The tipping kibble is generally used in connection with some form of tipping waggon (see Fig. 21), which on the balanced doors being closed is pushed underneath to receive the contents of the kibble. When the kibble has been emptied, the waggon is removed, the doors are opened, and the kibble lowered into the shaft.

Tipping waggons are constructed in various forms to suit various purposes, such as the single end tip, the double side tip, and the all-round tip waggon; the general construction is the same, but they are so arranged that the contents may be discharged as described.

The body of the double side-tipping waggon shown in Fig. 21 is made of steel, and is carried upon four pairs of trunnions. A large tipping angle is thus obtained, and a free discharge of the material ensured. The sides and bottoms are formed with one plate, which is bent to further assist in the discharge. The trunnions are firmly riveted through the end plate of the body, which is strengthened by an additional inside and outside trunnion plate. Each end of the body is made with one plate, which is flanged by hydraulic machinery, and is riveted firmly to the sides. A strong half-round welded ring is riveted round the top, and holds it rigidly together. The body as thus constructed is very strong and durable. The under-frame is of channel steel, with steel bowed buffer ends, and steel angle stays across the underframe; the trunnion supports are of angle steel; the draw-bar passes throughout the length of the under-frame, forming a central stay; the axles are of steel and the wheels of chilled iron or cast steel. Waggons with round buffer ends are far superior for light railway work to those built with corner buffers, as the liability to derailment on curves is greatly lessened. This may be of no importance in tip waggons for a sinking pit, and other buffers, if preferred, can be substituted. The trunnions are placed at such a distance apart as to ensure steady running, thus dispensing with safety chains whilst still maintaining an easy tip.

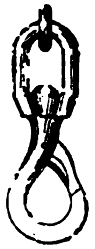

Fig. 20.—Spring Hook.

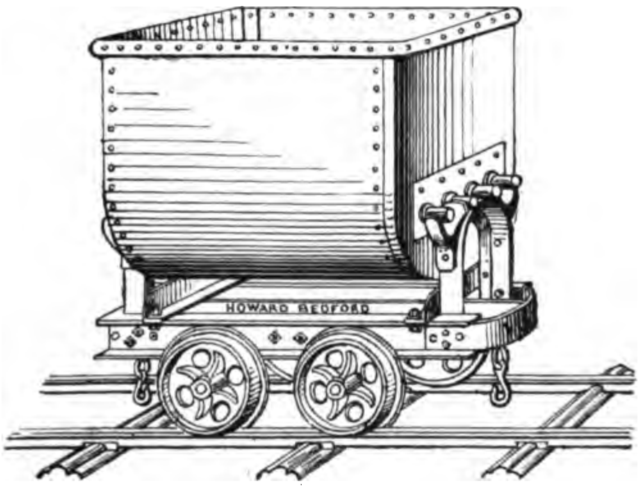

Fig. 21.—Double Side-Tipping Mining Waggon.

The usual capacities of these waggons are 10 and 16 cubic feet on gauges of 18 inches to 24.

After the first 6 feet of sinking has been done, which will most likely be through soil or clay, *curbs* or *cribs* must be put in. These are segments of wood (see Fig. 22) cut out to the circle of the pit, and are generally 6 inches square and made of oak or elm, the former being preferable. The joints must radiate truly from the centre of the shaft, and cleats, as shown in the sketches, secured to them at the surface in order that the joints may be brought into proper contact when the curb is fixed in the pit. Sometimes a scarfed joint is made between anything but pleasant to those riding in a sinking shaft, and may possibly cause an accident through giddiness.

If the shaft be fitted with wooden or iron conductors as the sinking proceeds, these may be used to guide a slide carrier or rider. It consists of a horizontal cross-bar, the ends of which fit the guides. This is carried by inclined struts to a circular piece fitting loosely over the winding rope. A buffer-catch is secured to the rope at its lower extremity, which fits into the circular portion of the slide carrier, and raises it in its ascent.

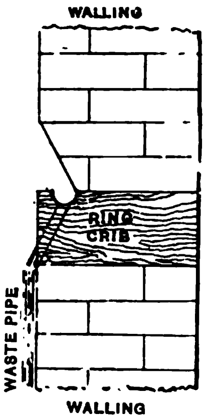

Fig. 23.—Ring Crib.

The guides at their lower ends have stoppers, so as to arrest the carrier in its descent, and hold it there whilst the buffer-catch becomes disengaged from the rider, and the kibble is free to descend without guides to the bottom of the shaft. Similarly it ascends unguided again until it reaches the carrier, which is afterwards lifted up with it.

Where wooden or iron guides are not available, wire ropes may be used as guides for the carrier. The guide ropes may be secured in the shaft to a balk stretched across the shaft; the balk also serves to arrest the further descent of the carrier.

The guide ropes are wound on the drums of steam or hand crabs on the surface, and pass over pulleys at the shaft there, so as to admit of their being easily lowered; they are long enough to reach the expected depth of the sinking.

The objection to this system is that the balk in the shaft requires frequent change. Mr. W. Galloway has patented a great improvement of this system, in which the ends of the guide ropes are secured to the cradle, beyond which the kibble descends unguided through a door in the cradle. In some instances a single guide rope has been used in deepening shafts already at work, where the circumstances were such as to make it undesirable or impossible for sinking and walling to proceed simultaneously while coal was being drawn from a higher level. The guide rope is wound on the drum of the capstan engine at the surface, and uncoiled to lengthen it as the sinking proceeds. During the walling it is used to raise and lower the cradle. While sinking is in progress the cradle is removed and a heavy weight attached to the lower end of the rope, which is lowered almost to the pit bottom. The rope thus forms a guide for the kibble all the way. One end of a horizontal bar is made with an eye to run freely up and down the guide rope, while the other is connected to a bar below the winding rope, so that the kibble in its ascent and descent is continually kept an equal distance from the guide rope. This very largely prevents oscillation of the kibble, which, however, cannot be run in the shaft so steadily with a single

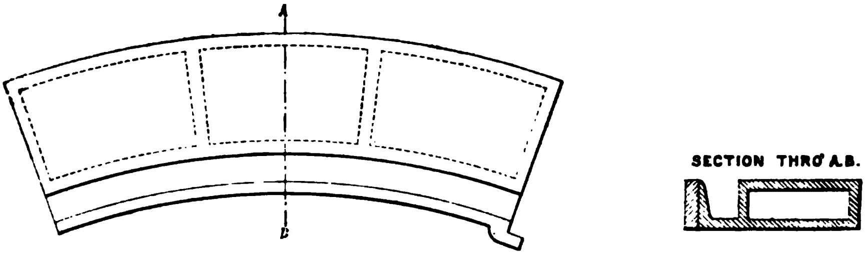

Fig. 24.—Combined Wedging and Ring Crib.

as with a double-rope guide. There are objections, too, to weights suspended in sinking shafts, as they must occasionally be raised above the sinkers working in the bottom.

A *ring-crib* (Fig. 23) is frequently used in wet pits. It consists of a crib hollowed out in the shape of a gutter, and is built into the shaft, the first two or three courses of brickwork upon it being inset or shorn back, so as to leave a portion of the crib in which the channel is cut exposed as shown in the sketch. The water which trickles down the sides of the shaft runs into this ring-crib, and from thence it is conveyed down the shaft by means of a "waste pipe," one end of which is let obliquely through a hole in the crib to the bottom of the channel, and the other is placed over the cistern, from which the pump takes its water. Sometimes a wedging crib is used as a "ring-crib" (see Fig. 24).

Frequently large quantities of water are found in the hard rocks, and if circumstances admit, these should be tubbed back with cast-iron tubbing. If, however, the bed of rock on which we are able to place the bottom part of this tubbing is near a seam of coal, or even if it is some distance away, but in communication with it by fissures, or the rocks between are not impervious to the passage of water, there will be no advantage derived from the tubbing, because the water will find its way down to the seam, and remain as much a burden to the colliery as if the tubbing had not been put in. Assuming an impervious bed of rock to be available, and that there is a large quantity of water below the walling, the sinking would be continued from the last wedging crib at the reduced diameter of 15 feet for a few feet, so as to leave a support under the wedging crib, and then gradually enlarged to a size suitable to take the tubbing, and on approaching the impervious rock alluded to, the shaft should be again reduced to a diameter of 15 feet, and the sinking carried a few feet, say 6, into it, to serve as a sump, and allow of the water kibbles being used whilst efforts are directed to the tubbing operations. Explosives should be avoided in sinking past the point where the wedging curb will be fixed.

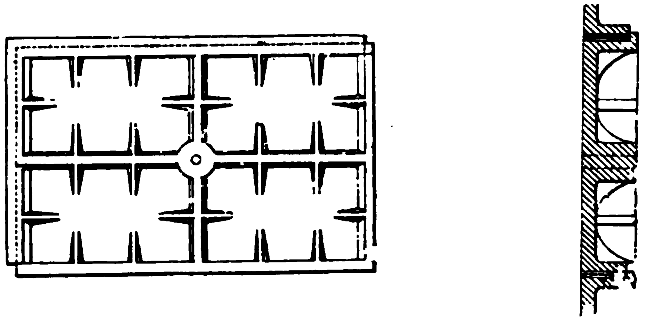
(A) Elevation. (B) Cross Section.

Fig. 25.—Segment of Metal Tubbing.

A bed should be carefully prepared at the commencement of the impervious rock for the wedging crib, or cribs (for often two are laid), which is somewhat similar to the walling curb, but of cast iron about 6 inches deep, and 13 inches in the bed. The pit is shorn back so as to admit of the wedging crib being placed, and also leave a small annular space round it; the bed for the reception of the wedging curb must be dressed with hacks perfectly smooth and level. The curb is then laid and securely wedged, the space behind having been filled with fir sheathing, and behind that again moss or oakum. Sometimes both the single and double cribs are provided with escape valves to release the air as it escapes from the back of the tubbing. The segments of *metal tubbing* (see Fig. 25), having first of all been tested by sounding them all over with a hammer and punch on the surface, are next to be proceeded with. These segments are cast with a smooth inner surface, and are flanged so as to fit into each other, a hole being left in the centre of each to allow the water to run out whilst building them up, and also for convenience in sending down the pit. Pitch-pine sheathing is first laid on the wedging crib, and then the segments are fitted round it, 10 or 12 forming the circle, and they are usually 2 or 3 feet high. The ground is shorn back near the crib where required as the tubbing is built up. The thickness of metal in the tubbing will depend upon the height the tubbing has to be carried, and varies from $\frac{3}{4}$ of an inch to $1\frac{1}{2}$ inches. Mr. Greenwell gives the following formula for estimating the thickness of metal tubbing, the height of the segment being 2 feet:—

Let $x =$ the required thickness in feet.
$P =$ the pressure or vertical depth in feet.
$D =$ the diameter of the pit, also in feet.

$$\text{Then } x = \cdot 03 + \frac{\mathrm{P} \times \mathrm{D}}{50{,}000};$$

so that if we had 60 fathoms of water-bearing strata to tub through in a 15-foot pit, we have

$$\cdot 03 + \frac{360 \times 15}{50{,}000} = \cdot 138 \text{ of a foot} = 1{\cdot}656 \text{ inch.}$$

In practice the thickness of tubbing is generally reduced every few feet upwards. Sometimes it is necessary to cover the tubbing with tar or wood lining, to help preserve it; if the shaft is afterwards to become an upcast, ventilated by a furnace, a brick lining over the tubbing will be necessary. But this would not be placed until the building of the tubbing in the shaft was quite completed. A second course of segments is now proceeded with, a sheathing of pitch pine having been laid all round on the top of the first, to allow of wedging when all is built up, and care must be taken in the building to break the joints of the tubbing. This method is continued until approaching the rock left to support the wedging crib above, a part of which must be shorn back to allow of the tubbing fitting in truly under the wedging crib. All the vertical joints in going upwards should have strips of wood laid behind them an inch thick, and about 6 inches broad, and a wedge-shaped piece driven behind, to force the segments well together at the joints, and the space behind the tubbing should be filled up with concrete. When all the tubbing has been thus placed in position, the important operation of wedging the joints is commenced, from the bottom upwards, leaving the centre holes till last. The plugging of these is upwards also, and requires skill and care if much water is coming through them.

Sometimes the tubbing, when carried above the water-bearing strata, is left open-topped, but this does not allow of such good wedging as the close-topped tubbing. Sometimes it is necessary with close-topped tubbing to put a pipe into one of the upper segments, and either allow it to remain open, and the water to run constantly down the pit, or to continue the pipe up the pit above the level of the water behind the tubbing. This allows a vent for the air from behind, which, when no provision was made for its removal, has been known to do considerable damage. The segments should have proper pieces cast on them to which to fasten the buntons, when the pit is fitted up with guides.

It has been stated that the Shireoaks pits have more tubbing than any other pits in England, viz. 170 yards. The pressure at the bottom is about 196 lbs. per square inch.

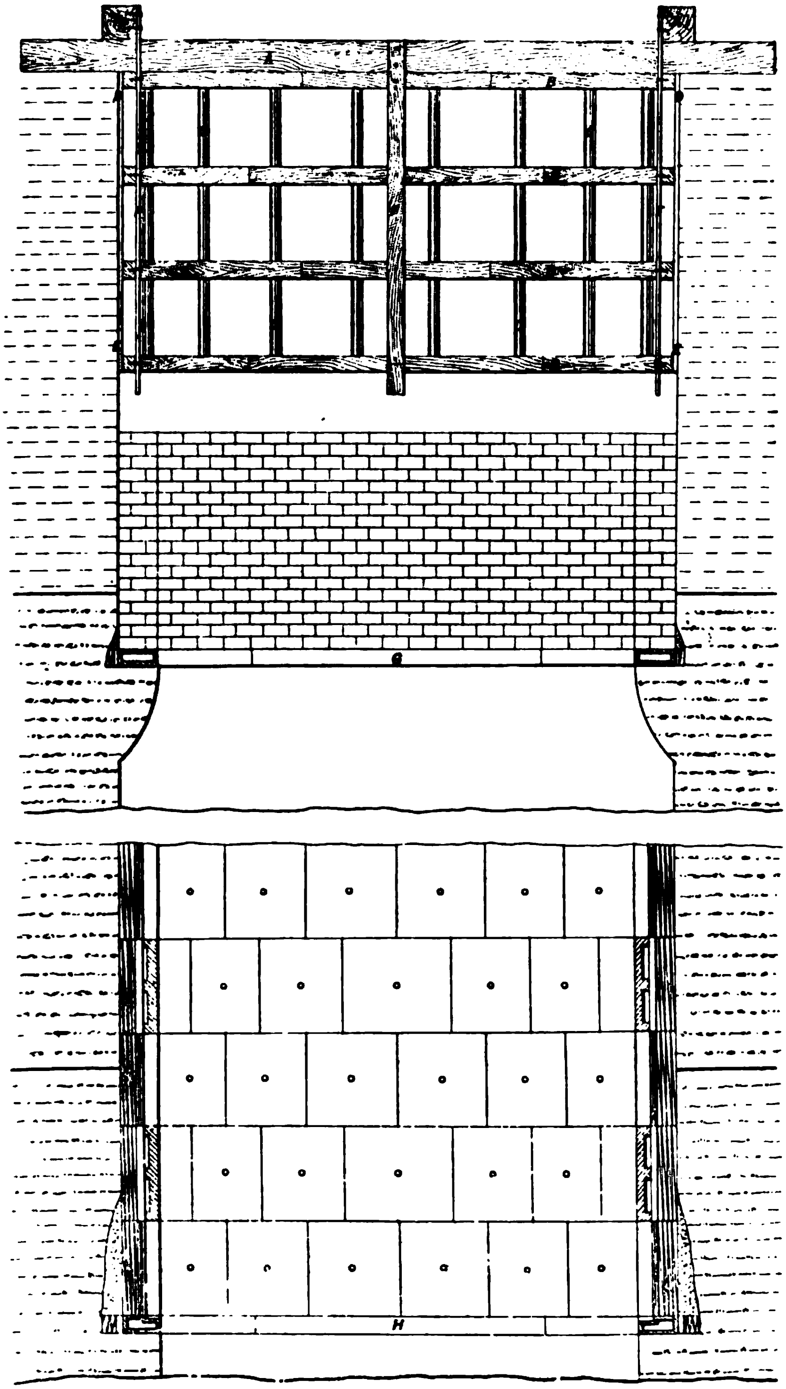

Fig. 26.—Timber, Walling, and Tubbing in a Sinking Shaft.

A. Balks laid across the top of the shaft. B. Timbering curbs. C. Punch props. D E. Backing deals, shown in section but otherwise omitted for the sake of clearness. F. Stringing deals. G. Walling curb, with the walling shown above it. H. Hollow cast-iron wedging curb with cast-iron tubbing resting on it.

It is often a matter of the utmost importance to those connected with a new sinking, that the shafts should be sunk as expeditiously as possible. Where the seam required to be worked lies at a great depth from the surface, and the shafts are to be of large size, some kind of machine-rock drill, worked by compressed air, will be necessary to bore the shot-holes in the bottom rapidly. At the Harris's Navigation Pits in South Wales, different forms of machine drills were used to expedite the work. The shafts forming the colliery are each 17 feet in diameter inside the walling, and are sunk to a depth of 760 yards.

A large diamond-boring machine was at first used in hard rock, the action of the drill being similar to that of the diamond-drill when used for prospecting purposes, and described in Chapter II. of this work. In this case the machine consisted of 8 drills, each drill being complete in itself and driven by compressed air, while the whole was attached to four beams fixed to a centre-piece. The engine was attached to the frame and lowered down the shaft, the motive power being conveyed down a pipe, the bottom of which was a flexible hose. The supply-water for the drills was taken down the pit in a pipe of smaller diameter than the drills, and was raised by the ordinary means of removing water from the pit bottom. The drills were capable of adjustment on the beams, and could be placed obliquely to any extent parallel with the face of the beams, each, however, remaining under separate control so as to be worked or stopped as required. About 30 holes were drilled, 8 proceeding simultaneously in the bottom from 3 to 5 feet deep, and also as an experiment deep holes from 15 to 30 feet were bored and blasted in sections. Whether it gave the most effective shot-holes or not, it was necessary to keep the long holes vertical, or their lower extremities passed beyond the line of the shaft. The short holes made frequent changes necessary, and resulted in the loss of diamonds in the drill crowns, causing the process to be abandoned. The Ingersoll rock-drill, which is fully described in Chapter XVI. of this work, was then applied to bore the shot-holes, and this it did very effectively.

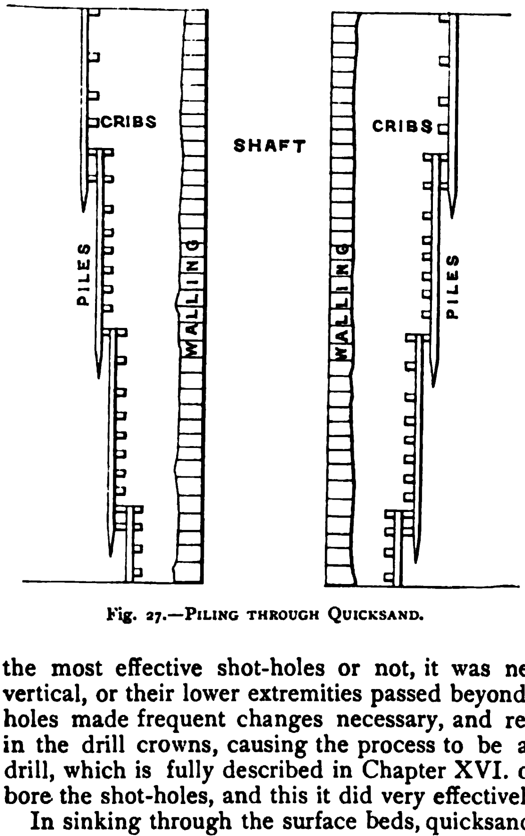

Fig. 27.—Piling through Quicksand.

In sinking through the surface beds, quicksands are occasionally met with which tax the energy and patience of those engaged in the sinking operations. The old method of sinking through quicksands was by piling, which requires the shaft at the commencement of the quicksand to be very large in diameter where there is a considerable depth of sand.

It is shown in Fig. 27 and is carried out as follows:—The piles are, say, 15 feet long, 6 inches wide, 3 inches thick, pointed and shod with iron, their edges being bevelled to allow a true fit into each other; the head should be hooped for the purpose of preserving it in driving. The curbs are 6 inches square, the piles 3 inches thick, so that each course of piling put in will reduce the size of pit

18 inches. With 15 feet length of piles, a fresh course is required every 12 feet, so that the depth of the quicksand in feet divided by 12 and the result multiplied by $1\frac{1}{2}$, gives the reduction in size. The reduction in a quicksand of 84 feet would be $\frac{84}{12} = 7$, and $7 \times 1\frac{1}{2} = 10\frac{1}{2}$ feet; if the diameter of the pit is to be 17 feet 6 inches, to allow of a 15-foot net size, it would require to be 10 feet 6 inches + 17 feet 6 inches = 28 feet in diameter at the top of the quicksand.

It will be necessary on reaching the quicksand to put an additional crib-bed 6 inches less in diameter than the diameter of the crib last put in; it must be concentric with it, and allow of the piles being driven between the two. Care must be taken to drive them down vertically, and after getting them down a few feet, as much of the quicksand as is practicable is taken out and a crib put in. Again the piles are driven down a few feet, the quicksand removed and a crib put in. These operations are continued until the piles are fully down, and the quicksand removed to within 3 feet of the bottom of them, a crib laid to support them and another laid inside and concentric to it, 18 inches less in diameter, to allow the next course of piles to be driven in the annular space between the two. The same operation of driving, excavating and laying cribs is again gone through. When within 3 feet of the foot of these piles another curb 18 inches less in diameter is put in to allow of another course of piles, and so on till the stone head is reached, when the wedging curb is laid and the walling or tubbing run up as expeditiously as possible through the treacherous ground.

Another method of getting through quicksands, consists in sinking by means of hollow cylinders of cast iron, pressed down by heavy weights piled on the top, but sometimes there is considerable difficulty in keeping the cylinders in a vertical position, especially if large boulders are met with.

An ingenious and efficient method of sinking through quicksand is Poetsch's Freezing System, whereby the quicksand is transformed into a solid mass. The quicksand in its changed and solid form is then sunk through in the ordinary manner.

A refrigerating liquid, consisting of a solution of chloride of calcium, is produced on the surface by means of proper machinery.* Through tubes this liquid is conveyed into the shaft required to be sunk through the quicksand. A zone of quicksand must be solidified round the shaft and downwards sufficiently far to form a wall or barrier all round the part to be excavated, and sufficiently thick to resist the surrounding pressure.

The thickness of this wall is determined beforehand by taking into account the depth at which the quicksand occurs, and the thickness of the quicksand itself.

Fig. 28 shows the principle of applying Poetsch's sinking process to a quicksand 20 feet thick. The tubes A A are of wrought iron 8 inches in diameter, provided at their lower extremity with a circular blade D 8 inches high and slightly tapered in form. These tubes are sunk vertically at distances apart varying from 1 foot to 4 feet. Upon their reaching the solid rock below the quicksand, the lower end of each tube is closed with a leaden plug C, fitting into the tapered end-piece and covered with several alternate layers of cement and pitch E to ensure the closure being water-tight. When each of the large tubes A A has been treated in this way, an inner tube B, $2\frac{1}{8}$ inches in diameter, is inserted within the larger tube, and is provided at its lower extremity with an opening F. The large tubes are flanged at G, and by this means attached to a cast-iron branch with three outlets and flanges H, H and J. The outside tubes

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* See Transactions, South Wales Institute of Mining Engineers, vol. xv. pp. 143—151.

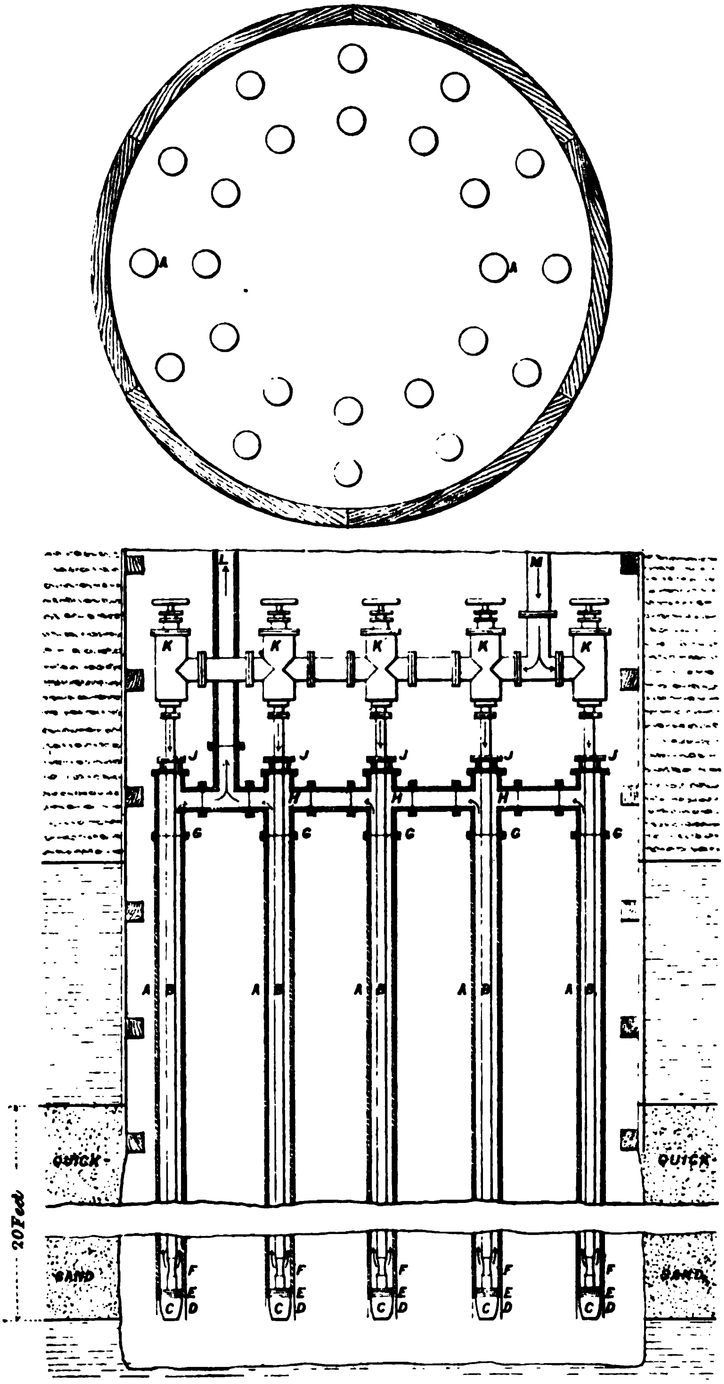

Fig. 28.—Pœtsch's Sinking Process

are all connected by means of the side flanges H, H, whilst the flange at J receives the inner descending tube B.

The course of the liquid, as shown by the arrows, is down the pipe M and through the inner central tubes B, returning in the annular space between the tubes A and B and ascending the pipe L. The whole system of pipes down which the liquid is conveyed is controlled by the valves K K. After ascending the pipe L the liquid again reaches the refrigerating apparatus. The quicksand in contact with the tubes A A thus becomes frozen into a solid mass. The solution is injected by means of a pump into the descending column M.

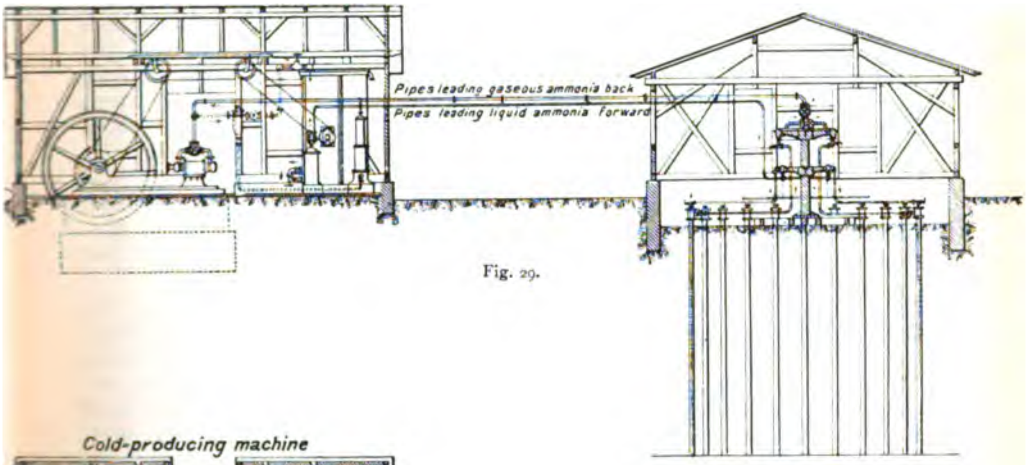
Pipes leading gaseous ammonia back
Pipes leading liquid ammonia forward

Fig. 29.

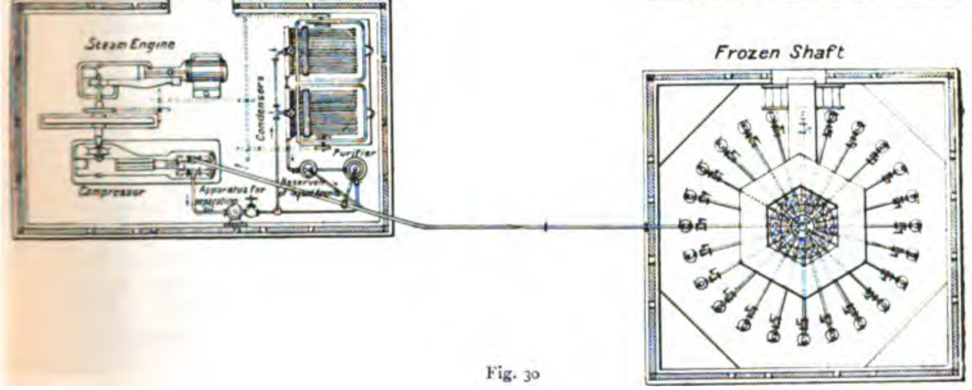

Fig. 30

THE GOBERT SYSTEM OF SHAFT SINKING.

The frozen mass is removed by picks, pointed hammers and crowbars, without the use of explosive. The wall formed by this process round the shafts is about 5 feet thick, and will resist all external pressure until the shaft is permanently secured.

Gobert System.—This is a modification of the Poetsch congelation method, and is specially applicable to the sinking of shafts through shifting sands and water-bearing strata.

Fig. 29 is a vertical section and Fig. 30 a plan showing the refrigerating plant and the shaft to be sunk, the two being as near each other as possible, and the shaft being lightly roofed over as a protection from the weather. The power of the steam-engine required varies with the diameter and depth of the shaft to be sunk, but need not exceed 40-horse-power unless the shaft is deep. The steam engine and compressor are placed horizontally side by side, and connected to the same shaft with a fly-wheel and pulley for belting between them.

Liquid ammonia is forced by the compressor through the series of wrought-iron tubes of the condenser, first to the reservoir, which acts as a kind of governor, and then by the lower of the two pipes seen in the vertical section, to the system of congelation tubes round the shaft. Great heat results from compressing the gaseous into liquid ammonia, and in order to abstract it, the condensers have a relatively large surface and cold water is caused to circulate freely round them. This water is kept in a state of agitation by means of the small water wheels with floats, shown in the drawings, driven by belts off the pulley on the main shaft.

The machinery, by means of the upper of the two pipes seen in the vertical section, exhausts gaseous ammonia from the congelation tubes, sunk vertically beneath the surface of the earth round the site of the shaft, and forces it, condensed into a liquid form, first through the apparatus for separating the oil (see Fig. 30), and then into the condenser.

The compressor piston is freely lubricated with mineral oil, and some of the ammonia comes into contact with and is absorbed by it. The mixture might choke the tubes of the condensers and possibly even reach the congealing tubes, if the two substances were not separated. This is effected chiefly by the oil-separator, but, as an additional precaution, a space is provided at the bottom of each congelation tube for the reception of any oil that may be carried there. The oil retained in the separator is not effectually separated from the ammonia, but is slightly mixed with it; this ammonia, however, is recovered in the purifier, where it is driven off by distillation. The distillation is effected by means of a worm through which steam from the boiler circulates; the ammonia vapour is led by a small curved pipe, seen in Fig. 29, into the main pipe, leading the gaseous ammonia from the shaft to the compressor.

Over the centre of the area forming the intended shaft are two pipe rings, the lower of which is in connection with the ingoing pipe and receives the liquid ammonia, and afterwards distributes it, by the radial pipes to the vertical congelation pipes sunk in a circle below the surface of the earth. The upper ring is in connection with the return pipe, and forms a receiver for the collection of the gaseous ammonia from separate orifices in the same congelation tubes after it has by evaporation in these tubes produced the desired refrigerating effect. The gaseous ammonia is drawn from the upper ring pipe to the condenser through the return pipe.

The liquid ammonia is not allowed to fall to the bottom of the tube and collect in a mass, but in order to cause the evaporation of the greatest possible amount of liquid in a given space of time, the small pipe for leading the freezing liquid through the tube is made to assume either a wavy or a spiral form, as shown in Figs. 31 and 32, in which A represents the congealing tube, B the pipe for leading the freezing liquid, and C small holes for allowing the liquid to escape into the tube, at points more or less frequent, as may be desired. The injecting pipe is led down nearly but not quite to the bottom of the congealing tube, and both pipe and tube are closed at the bottom. The entrance of the congealing liquid into the injecting pipe is carefully regulated and descends slowly in a thin stream, the flow being retarded by the waves or spirals, and giving up a part of itself at intervals.

The source of heat necessary for evaporating the liquid is the higher temperature of the surrounding strata, and this heat passes not only through the thickness of the congealing tube, but also across the frozen wall, which soon surrounds it. By this arrangement the liquid to be evaporated escapes into the congealing tube at all depths simultaneously, and the whole source of heat available is thus utilised at the same time for evaporating the freezing liquid. In other words, the refrigeration is effected simultaneously at all points.

The diameter, number, and arrangement of the holes in the injecting pipe, and also the pitch of the spirals or undulations, are varied in accordance with the depth in order to produce a greater freezing effect at special points, or a uniform freezing in accordance with the requirements. A congelation may therefore be arranged to have the frozen column of larger diameter at the bottom than at the top, on the supposition that the measures are of uniform consistency

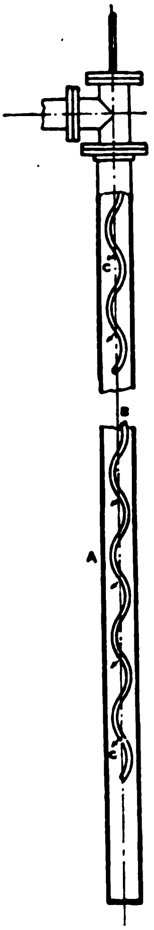

Fig. 31.

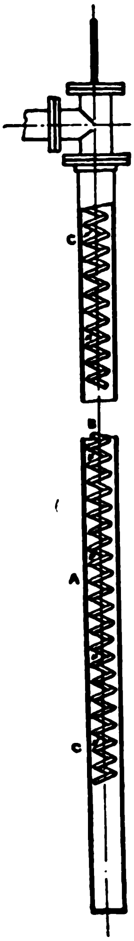

Fig. 32.

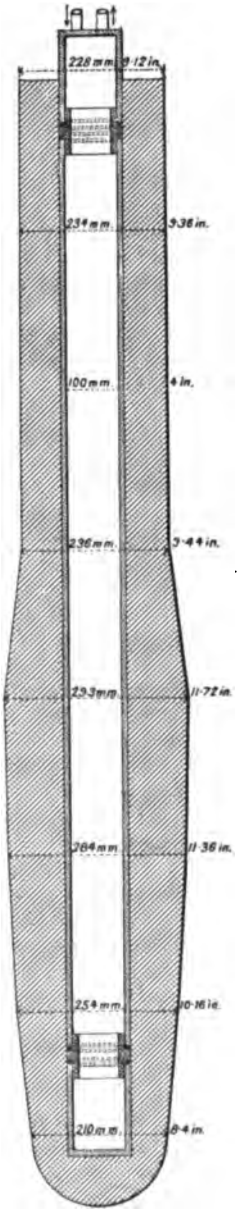

Fig. 33.

The Gobert System of Shaft Sinking.

in order that its stability may be maintained while the shaft is being sunk through it.

The arrows in Figs. 29 and 30 show the course of the ammonia in its passage, as a liquid, from the compressor to the condensers, and then on to the congelation tubes, and also its return, in a gaseous state, from the congelation tubes to the compressor, to be again liquefied, again evaporated, and so on. The same ammonia serves indefinitely, with the addition of a small quantity to compensate for waste.

The process requires a large quantity of cold water for use at the condensers, but this must not be drawn from any point so near the site of the shaft as to create a current, which might oppose and retard the congelation, by licking or washing the congelation tubes, thus depriving them of their refrigerating effect, which would be carried away instead of going into the surrounding sand.

When the wet sand or loose material has been frozen round the tubes, sinking may be commenced with a small windlass placed between the collecting and distributing rings and the circumference of the circle of congelation tubes. The men enter, and the excavated material is removed, laterally, near the surface, between two congelation tubes, where also access is obtained for the segments of tubbing.

More important winding apparatus must of course replace the windlass when the sinking has reached a depth of two or three fathoms; then if the arrangements have been made judiciously, and due precautions taken, the frozen mass will be so large as to require slighter refrigerating power to maintain it than that required for its production. This allows of the removal of one or two radial pipes for distributing the ammonia in order to allow of more space for the working of a winding engine.

In the Poetsch system an uncongealable liquid, cooled down to a temperature below the freezing point of water, is sent through the tubes. Thus, if there be any cracks or faulty joints in the tubes, the liquid will escape into the sand or soil, which is a serious drawback, as the sinking cannot be effected unless all quicksand or water-bearing strata are solidified. The uncongealability of the liquid is, of course, a necessary and indispensable condition in order that it may circulate and freeze the watery mass surrounding the tubes. Leaky joints are extremely difficult to find, and to repair one the whole column of tubes must be raised to the surface; besides, the unfrozen space left by a leaky joint is extremely difficult to freeze afterwards, as the soil is then saturated with uncongealable liquid.

A great advantage claimed for the Gobert modification is that if the congelation tube be surrounded with water and there be a defective joint, the water will simply enter the tube, on account of the pressure therein being less than that outside. If such an accident occurs at all it is usually after congelation has proceeded for some time and the tube is already surrounded with ice; there will then be no interruption in the work. The liquid ammonia always enters the tubes at a temperature above freezing point, and in practice varies from between 20 and 35 degs. Cent. (68 to 95 degs. Fahr.). The cold produced is due to the liquid ammonia becoming volatilised in the tubes.

It is of course impossible to entirely guard against leaky joints. The thrust of the superincumbent measures severely tries them, but special attention has been given to the design of the joints in order to increase their power to resist the strains to which they may be subjected. The method of connecting the ends of congelation tubes has been by screwing one end into another without internal sockets. The thinning of these tubes at the joints frequently caused them to break in being withdrawn from the ground.

The form of joint used by M. Gobert is shown in Fig. 33. Its chief feature is an internal collar, or ring, shown by crossed hatchings in the section. This

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collar has an outside flange of the same outside diameter as that of the tubes which it serves to connect. The flange is undercut on both sides so as to be of dovetailed section. Each end of a tube is also bevelled or curved off so as to afford with the collar flange a groove wider inside than out, holding and compressing the lead ring or washer instead of forcing it outwards, thus affording an absolutely tight joint when the ends of the tubes are screwed on to the collar. In some cases, especially for joining the smaller sizes of tubes, the internal collar is made without a flange, and then only one lead washer is used placed between the ends of the tubes, which must be bevelled and curved just the same as when the collar is flanged. On the tubes being screwed up, they squeeze the washer between them, just as the gland of a stuffing-box compresses the packing. The outer lines of the section, Fig. 33, show the form and extent to which a tube was covered with ice after having been immersed for 32 hours in a tank filled with water; the ice weighed 62 kilogs. (137 lbs.) at the end of the operation. If the tube had been immersed in wet sand instead of clear water, the congelation would have been more rapid. One of the two smaller tubes shown at the top of the congelation tube serves to introduce the liquid ammonia, while the other carries off the gas to the upper ring-pipe.

Instead of ammonia, any other liquid susceptible of easily assuming the gaseous state, such as liquid carbonic acid or liquid anhydrous sulphurous acid, may be employed with a suitable modification of the engine.

M. Gobert proposes to apply his system to the enlargement of a shaft to 3 m. (nearly 10 ft.) from its present diameter of 1 m. (about 3 ft. 3 in.), the shaft being 54½ fathoms deep and tubbed with cast-iron rings, the measures being watery. To avoid the necessity of putting down boreholes for the congelation tubes it is proposed to freeze a sufficient thickness of the mass next the tubbing by leading four or six of the tubes down the shaft and so freeze an annular wall, 2 or 3 yards thick around the existing shaft, and then remove the material necessary in the enlargement in lifts or stopes about 2 yards high, beginning at the bottom. The intention is to lay new tubbing rings of the enlarged diameter of the shaft in sections the height of each stope; and probably replace the old tubbing concurrently with the laying of the new, the space between the tubbings being filled with sacks of wet sand in order to conduct the cold, but leaving the inner tubbing unbolted so as to prevent interference with the swelling of the earth or sand towards the inside due to the expansion produced by freezing. On arriving at the surface it will be necessary to descend again in order to remove the sacks of frozen sand from the inside of the permanent tubbing. It may be possible to avoid the replacing of the inner tubbing, because the four or six congelation tubes might permit of packing the sacks in vertically, and the sand in them would soon be transformed into hard frozen masses. The subsequent removal of the sacks might be effected by leading steam down the tubes to thaw the whole interior of the shaft.

The possibility of producing congelation in water-bearing strata at a great depth without at the same time freezing the surface renders the Gobert system peculiarly applicable to the completion of shafts partly sunk and already lined with water-tight tubbing all the way except at the bottom. Such shafts soon become flooded through breaches in the bottom of the tubbing if the pumps are overpowered. If the Poetsch system be applied to them the effect of freezing throughout from the surface downwards would burst all the tubbing. By the Gobert system pipes may be led down the shaft so as to concentrate the congelation at the bottom. When a plug of ice has been formed under the tubbing, the water may easily be pumped from above, and the sinking of the shaft resumed at a slightly reduced diameter. Such instances of flooding in tubbed shafts are very rare where the ordinary process of shaft-sinking has been followed, because here the pumping power must be adequate to remove all water from the shaft until the sinking reaches a hard impervious rock, and if after the completion of the tubbing a breach or leaky joint should occur the water would not be more than was dealt with before, and therefore would not be in excess of the pumping power still available. But an instance of the kind has actually occurred at the Cannock and Huntington sinking, near Stafford, where two shafts have been sunk by the Kind-Chaudron system. These shafts are 15 feet in diameter inside, being sunk 150 yards, and tubbed with cast-iron rings from the moss-box upwards. Although an enormous sum has been expended in the sinking, these shafts are at present full of water, owing to the failure of the moss-box, which it is thought was cracked by the accidental falling of the column of tubbing upon it.

M. Gobert has prepared plans, shown in Figs. 34 to 38, to illustrate the applicability of his system to the completion of these or similar sinkings, where the deepening is through shifting sands or yielding strata copiously watered.

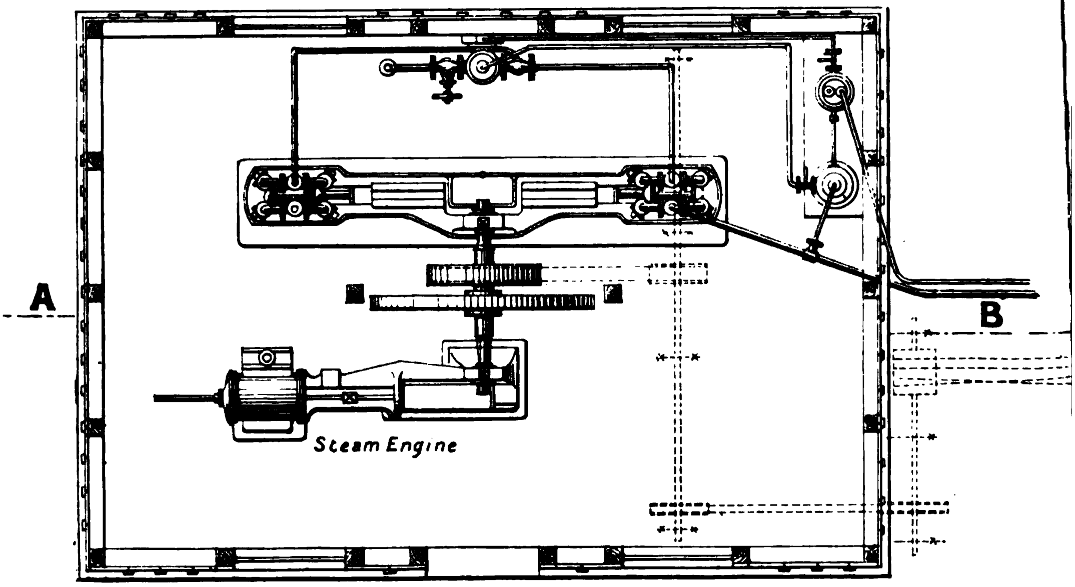

Fig. 36.—Ground View at E F.

The Gobert System of Shaft Sinking.

Fig. 34 is a sectional view of the plant used on the surface, the shed erected over the shaft, and the shaft itself with the congelation pipes led down it. The moss-box and cast-iron tubbing previously inserted are shown. To be successful the material to be passed through from the present bottom to strata impervious to water must not form any obstacle to the congelation pipes being pressed or driven through it. After this is accomplished the continuous working of the engine would transform the material surrounding the tubes into a solid frozen mass, L, L, which would not extend upward far enough to injure the cast-iron tubbing already in, except perhaps one or two of the bottom rings. Fig. 34 shows the sinkers at work removing with pick and shovel the frozen material which is wound to the surface in an ordinary bowk or kibble. It will be noticed that the congelation pipes being led down the existing shaft necessitates its being continued below the moss-box at a reduced diameter—say 12 feet, the upper portion being 15, because an annular wall of protection must always be left to resist the side pressure. The sinking could only be resumed at its full size of 15 feet by placing the congelation pipes in holes specially bored outside the original shaft, the enormous expense of which would for a great depth be prohibitive. When the sinking has progressed to the bottom of the frozen mass a crib is laid and a lining of cast-iron tubbing built up on it. Fig. 35 shows this work in progress, with three rings of tubbing laid. As it is built up the space behind the tubbing is carefully packed with concrete, consisting of a mixture of cement, sand, pebbles, water, and caustic soda, the last being used to ensure the solidifying of the concrete notwithstanding its frozen condition. This inner tubbing is built up to outset any injured rings in the shaft above, and special care is needed to make a water-tight joint at the finish between the inner and outer tubbing. M. Gobert's method of packing within sight of the workmen admits of this being thoroughly done. On the resumption of sinking in the hard ground below where the freezing process was necessary, as soon as a thoroughly sound rock bed is pierced, a crib may be laid in it and tubbing carried up to join that built above, until which time the freezing process may have to be maintained.

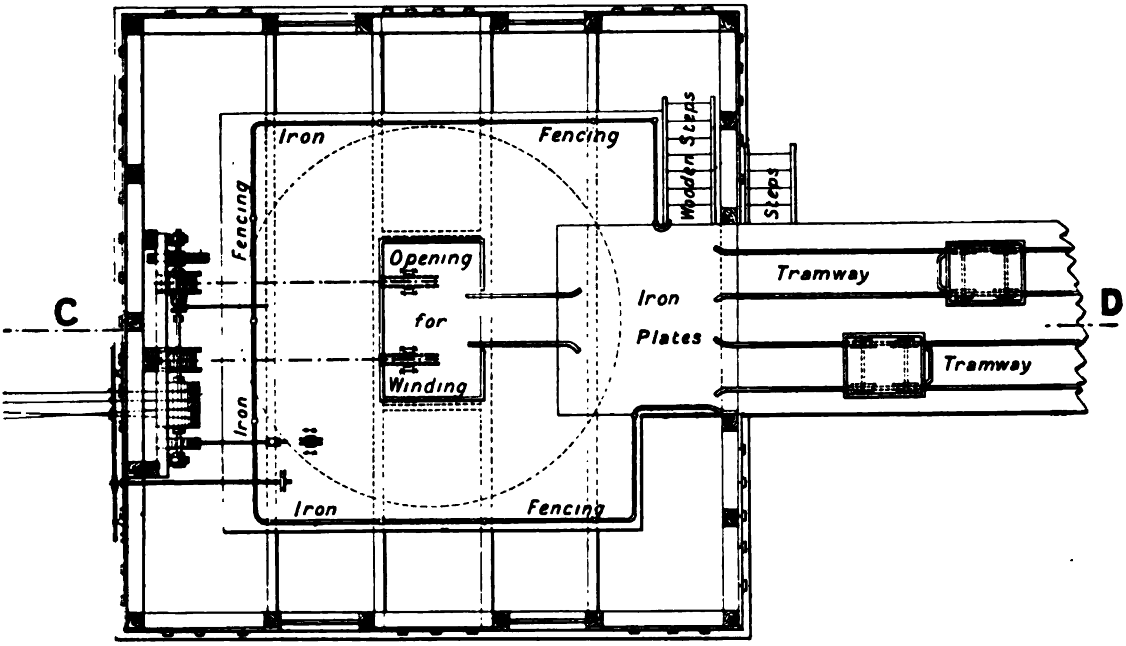

Fig. 37.—Ground View at G H.

The Gobert System of Shaft Sinking.

Fig. 36 shows a ground view of the refrigerating plant which has been described, but which it will be noticed is a little differently arranged here, as also are the pipes for collecting the gaseous ammonia and those for distributing the liquid ammonia shown in Fig. 38; this is merely a detail which does not interfere with the principle of the invention, but the method here shown of placing the separating pipes outside the shaft instead of in central pipe rings, as shown in Fig. 30, is a great improvement, as it leaves the entrance to the shaft freer from obstruction.

Fig. 37 is a ground view of the shaft, showing the staging and hand-rail fencing erected, the tramways leading to the tipping ground, and the opening in the staging through which the kibbles are wound. In Belgium the sinkers frequently make use of ladders in the shaft as a means of ingress and egress, and the drawing shows the winding of the kibbles by the steam engine, which works the compressor and which is kept continually at work. By means of belting and fast and loose pulleys it is possible to make use of the engine thus for limited depths, but in this country climbing shafts by ladders is regarded as a waste of energy, and it is much better to have a separate engine to wind the kibbles and the workmen. Two kibbles may be used at the same time if desired by making suitable arrangements.

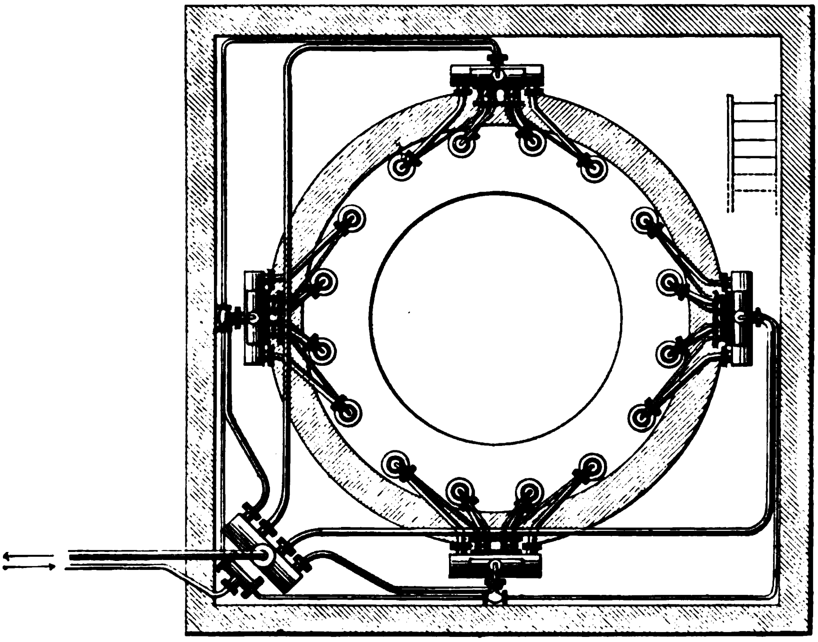

Fig. 38.—Ground View at I K.
The Gobert System of Shaft Sinking.

The cold-producing plant has been specially designed by the Maschinenfabrik Germania, of Chemnitz, that company having made arrangements for undertaking, with M. Gobert, the sinking of shafts by congelation.

The **Kind-Chaudron System** has been successfully applied in some instances for sinking where there are very heavy feeders of water to contend with.

Fig. 39 shows the method adopted.*

The sinkers employed at the shaft are four or five in number, and stand on a working floor about 6 yards below the surface. The shaft is about 4 feet more in diameter over the first 6 yards than below, where it becomes the finished size. The first operation consists of boring out, by machinery, a cylindrical hole of about 4½ or 5 feet in diameter. This hole is afterwards enlarged by a second, or if the pit is large, by a third operation. The first operation is kept at least 33 feet in advance of the second.

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* See Transactions, North of England Institute of Mining Engineers, vol. XX. pp. 187—202.

The cutting of the material is by the same kind of means in both or all stages (the central and enlarging), but the removal of the broken ground is different. In each case the cutting tool (Figs. 40–46) consists of a horizontal wrought-iron bar, to the under side of which are attached steeled teeth. The arrangement of these teeth is such that as the bar revolves round the central axis of the pit, each tooth in falling with the bar through the length of stroke used

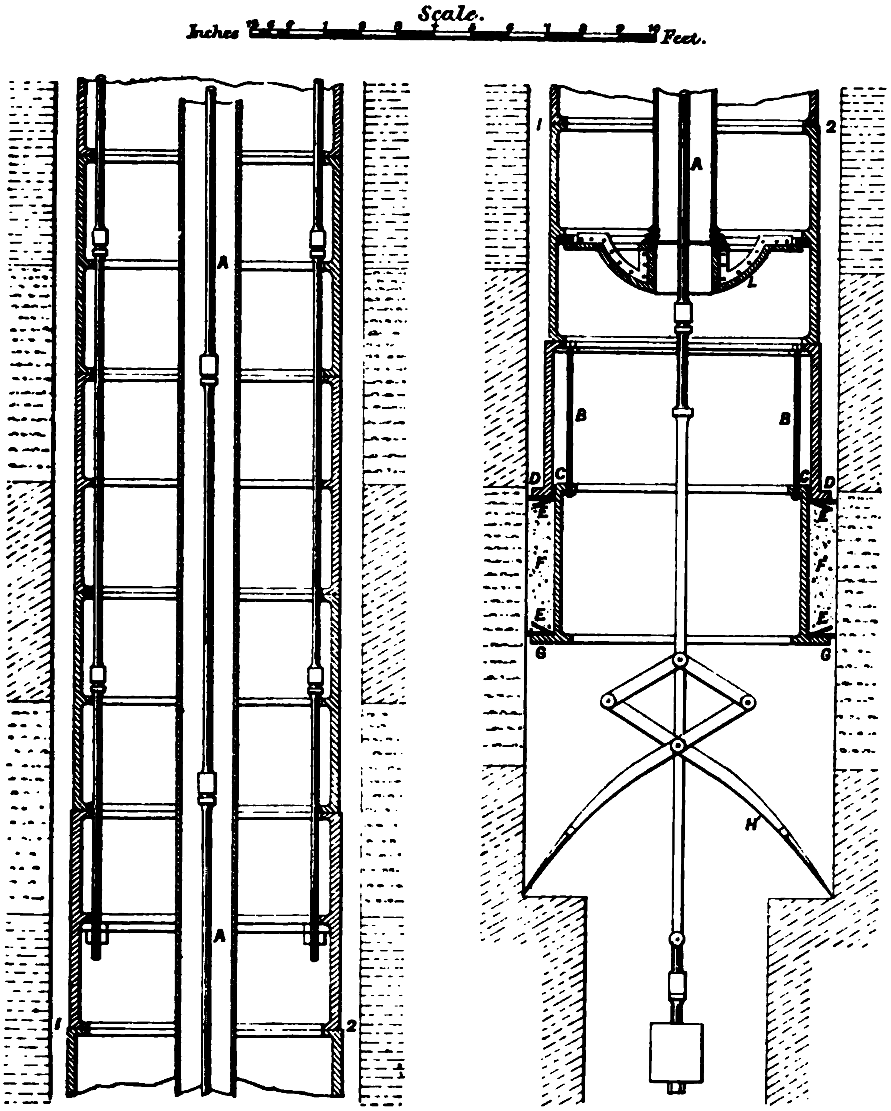


Fig. 39.—Kind-Chaudron System of Shaft Sinking.

(from 10 to 20 inches) cuts for itself an annular portion of the bottom of the shaft. The large and small cutters, or *trépans*, are lifted and turned by the same rods, which are made of pine about 8 inches square and 20 yards long, and

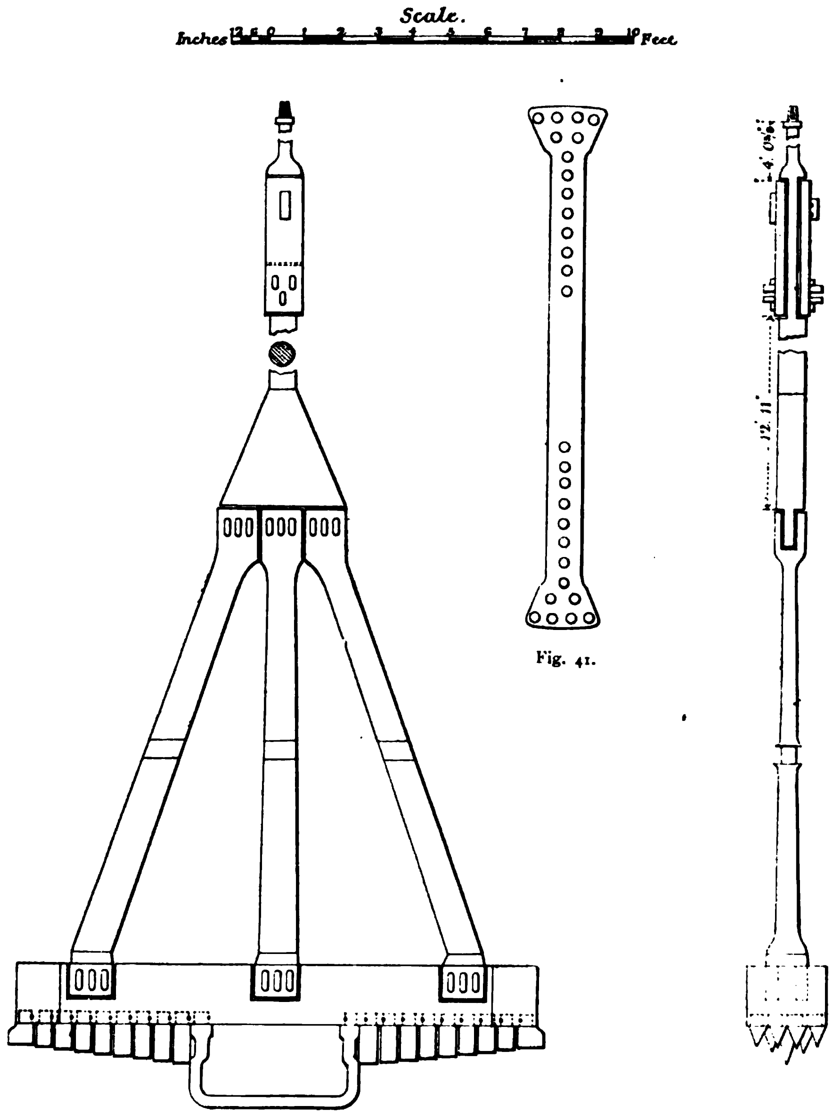


Fig. 40. Fig. 41. Fig. 42.

Kind-Chaudron System—Large Trépan.

connected by male and female screws. Only one *trépan* can be used at a time. In the sinking of a shaft at Maurage, near Mons, a simple lever was placed at the surface level, one end of which was attached to the rods by a strong flat chain, whilst near the other end it had a direct connection with the piston rod of a steam engine. The single cylinder was 39⅝ inches in diameter, with a

40-inch stroke, and was placed vertically below the beam. The lift of the rods was effected by admitting steam to the cylinder above the piston, depressing that end of the lever and raising the rods and cutter, which then fell by their own weight. The lever or striking beam was 23 feet long, 11 feet on the side of

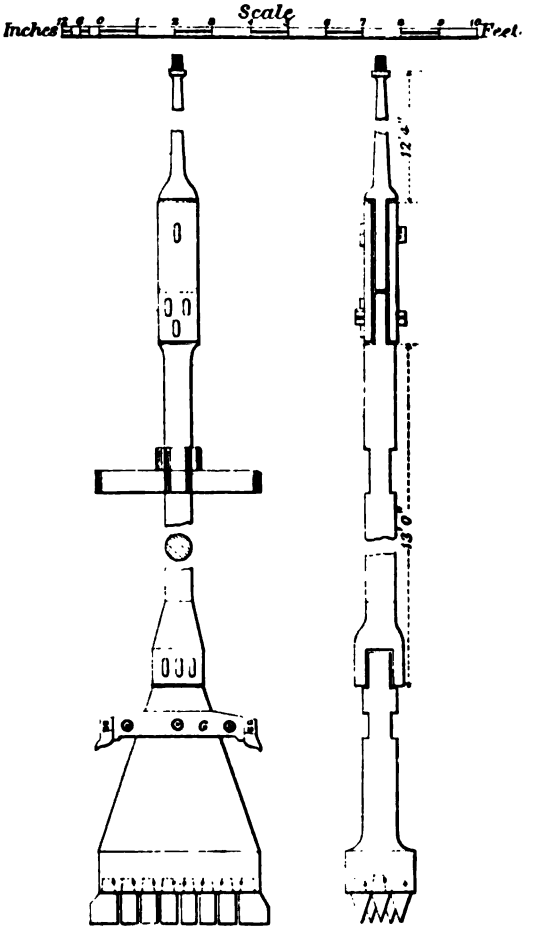


Fig. 43. Fig. 44

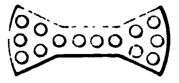


Fig. 45.

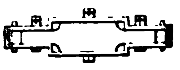


Fig. 46.

Kind-Chaudron System—Small Trépan.

the rods and 12 feet on the side of the engine, and was composed of two logs of wood, one placed over the other, strengthened on each side by a stout plate of iron. Beneath the suspension chain is a lengthening screw, below which is a very strong swivel, by means of which the rotating movement is given to the rods and cutter.

The top length of rod has eyes for the insertion of cross bars. The workmen, standing on the wooden flap doors which close the shaft (except a central hole for the rods to work through) at the level of the working floor, turn the rods by means of the cross bars at each stroke. The smaller *trépan* (Figs. 43–46) requires modification in its construction, according to the nature of the material to be cut. If soft, the bar to which the teeth are attached is suspended by a fork of wrought iron, but for hard rock it is forged in a single piece and weighs about 8 tons. In ordinary ground this cutter advances about 8 feet per day. The teeth are well steeled, fit into sockets in the main bar, and are further secured by a pin easily removed when the teeth require sharpening or renewing.

The cutter is driven about 9 or 10 strokes a minute usually, but sometimes more, and after being some hours at work, it is raised by a small capstan-engine, with a flat hemp rope of 14¼ inches wide by 2⅜ inches thick. The rods require unscrewing (as in the ordinary manner of boring rods) as the cutter is being withdrawn. To clear the whole of the cut material a sheet-iron cylinder 6 feet long, with two valves in the bottom, is lowered and raised by the rods.

The larger *trépan* (Figs. 40–42) weighs about 16 tons and has a bar of wrought iron, to a portion of which teeth are attached as in the smaller *trépan*. The teeth are fixed on that portion of the bar which exceeds the diameter of the hole cut at the first operation. The large *trépan* is guided below by a cradle of iron bars fitting loosely within the smaller diameter.

The arrangement of teeth is such as to cause them to cut a sloping surface at the bottom of the shaft so as to ensure the cut material to roll into the smaller pit, where they drop into a sheet-iron bucket previously lowered into it. The rate of progress varies from 3 feet per day in ordinary ground to 1 foot per day in hard rock.

To obviate the excessive vibration which would otherwise be imparted to the rods by tools of such a weight, a special joint, called a slide piece, of great strength, is applied.

To maintain the boring rods in a vertical position throughout their stroke, guides are attached to the upper part of the implement. In the smaller cutter these consist of two strong iron bars set at right angles (G, Fig. 43), having teeth fixed at their extremity, which slightly enlarge and, at the same time, smooth down the sides of the hole. For the large *trépan*, one of the cross pieces is rigid and the other, at right angles, is hinged on both sides of the main rod in such manner that it can be lowered or raised by ropes during the shifting of tools, through a small opening in the working floor. The guide, when in position, forms a fixed cross, through the central opening of which the cutter rod slides freely up and down.

The pit having been sunk through the water-bearing strata in this way requires tubbing. In this process the lowermost ring, like all the upper portion, is cast in one piece. The lower flange G, Fig. 39, is turned outwards, its upper flange C inwards, and it rests on a bed in water-tight ground below the water-bearing rocks. Upon the lower flange and all round the ring a wall of well-selected moss, F, is packed tightly against it and secured in its position by a net placed at the back of it. To assist in forcing the moss against the side of the shaft, small sheet-iron springs, E E, are placed above and below, the effect of which is to give the pressure a definite direction. On the moss cushion rests the next ring, D. Its bottom flange is turned outwards, the top one inwards, and it is of such a size as to slide down outside the bottom ring, when the moss is sufficiently pressed down by the weight, and upon this sliding ring the ordinary tubbing rings are built. Each flange is truly planed, and between the flanges a ring of sheet-lead, ⅛th of an inch thick, is laid. After screwing up the bolts the lead is beaten in on both sides with hammer and chisel. Each ring of tubbing is from 4½ to 5 feet high, is of extra thickness, and tested on the surface by hydraulic pressure. The bottom simple ring is 2⅝ inches thick, and weighs 11¾ tons for a 14-foot pit. The upper rings are gradually lighter.

To facilitate the gradual lowering of the enormous weight of the tubbing, by means of six rods and screws used for this purpose, a diaphragm or false bottom, L, is attached by screw-bolts at a point near the bottom of the tubbing, and this causes it to float on the water. A central equilibrium tube, A, passes up the shaft from the false bottom, and through cocks, placed at intervals, allows of water being poured into the middle of the tubbing in sufficient quantity as may be required to help its descent. By this means no greater weight than 40 tons rests on the suspension rods.

The bottom ring or moss-box is suspended by light rods, B, to the flange of an upper ring, and is lowered into its position on its seat or bed. The weight of tubbing then bears on the moss and squeezes it down and against the sides of the shaft so as to form a thoroughly water-tight joint.

The annular space between the rings and the shaft is filled in with concrete and allowed to consolidate before the water is drawn out of the pit. The success of the undertaking to a large extent depends on the perfection with which the seat of the moss box is cut and smoothed, and to ensure its suitable condition, a gigantic pair of pincers, H, with arms on the principle of a lazy-tongs, is lowered with and underneath the whole of the tubbing by means of a rod passing up through the central tube. By working the rod up and down, the ends of this tool may be made either to expand to the full size of the shaft, or brought closely together and thus pick up small pieces of stone or other material which may be lying on the bed of the shaft; when in its contracted form it may be passed into the central shaft to be out of the way. After the concrete has set, the water is pumped out, the false bottom taken off by unscrewing the bolts which attached it to a flange, and the moss box is examined. For safety, a lower seating is cut and prepared in the rock a few feet deeper, a wedging curb put in by hand in segments, and the tubbing built up on it to the moss box, against which it is securely wedged. The shaft being now free from water may be sunk deeper by ordinary methods.

The explosives used for blasting in sinking are:—(1) gunpowder, which is useful for rending operations: when the rock is not hard and is also free from water it may be used to much advantage; (2) dynamite, which as a "shattering" agent is a very useful explosive in a wet pit and in a hard rock, as the water does not injure it. Its use, however, has proved dangerous in sinking shafts, owing to missed shots remaining undiscovered, and being accidentally exploded in clearing away the *débris*, or in boring fresh holes. (See also Chapter XVI.)

The following list of shafts having a depth of 1,500 feet and upwards is here given, but such list is by no means exhaustive:—

Country.	Colliery or Mine.	Depth in Feet.
Africa, South .	Nourse Deep	1,578
	Robinson Deep, S.A.R.	1,991
America, United States.	Potsville shaft (disused), Philadelphia and Reading Coal and Iron Company	2,000
	Kennedy Mine, Jackson, California	2,150
	Grass Valley, Idaho	2,182
	California Mine, Colorado	2,260
	Belcher Mine and Crown Point Mine (silver), Comstock, Nevada	3,033
	Yellow Jacket (silver), Comstock, Nevada	3,123
	Tamarack (copper), Lake Superior	(1) 4,450
	Red Jacket, Calumet and Hecla, Lake Superior	(2) 4,900
Australia, Victoria.	Victory and Pandora (gold)	(3) 1,872
	Newington (gold)	1,941

Country.	Colliery or Mine.	Depth in Feet.
Australia, Victoria (*continued*).	Magdala, Stawell	2,409
	Lazarus, Bendigo	3,024
	Lansell's (gold), Bendigo	3,302
Austria-Hungary.	Amalia, Schemnitz, Hungary	1,750
	Einigkeit, Joachimsthal, Bohemia	1,750
	Procopi, Przibram, Bohemia	2,900
	Franz Joseph, Przibram	2,900
	Anna, Przibram	3,100
	Maria, Przibram	3,281
	Adalbert (lead, etc.), Przibram	(4) 3,672
Belgium.	Cécile Pit, Seraing	(5) 1,710
	Marihaye Colliery, Liége	2,100
	Houssu Colliery, Centre	2,300
	Sacre Madame Colliery, Charleroi	2,499
	St. Joseph Réunion Colliery, Charleroi	2,763
	Cipley Colliery, Mons	2,950
	St. André shaft, Poirier Colliery, Charleroi	3,100
	Marchienne Colliery	(6) 3,117
	Viernoy shaft, Anderlues	3,300
	Simon Lambert (disused)	3,489
	Viviers shaft, Gilly, Charleroi	3,750
	Produits Colliery, Mons	(7) 3,937
France.	Ronchamp Colliery, Haut-Saône	1,870
	Hottinguer shaft, Epinac	(8) 2,000
	Treuil Colliery, St. Etienne	2,034
	Montchanin Colliery, Le Creuzot	2,300
Germany.	Freiberg, Saxony (maximum depth)	2,060
	Camphausen Colliery, Saarbrücken	2,296
	Maria Colliery, Hongen, Westphalia	2,300
	Hansa Colliery, Huckarde, Westphalia	2,330
	Concordia Colliery, Olsnitz, Saxony	2,420
	Frieden Colliery, Olsnitz, Saxony	2,515
	St. André (silver), Prussia	2,532
	Samson, St. Andreasberg, Harz	2,560
	Einigkeit, Lugau, Saxony	2,620
	Zwickau (lead, etc.), Saxony	2,637
	Kaiser Wilhelm II., Clausthal, Harz	2,960
Great Britain.	Wollaton Colliery, near Nottingham	(9) 1,500
	Mardy Colliery, No. 3 shaft, Glamorganshire	(10) 1,500
	Tankerville Lead Mine, Shropshire	1,500
	Wheal Sisters, Cornwall	1,500
	Avon Colliery, Glamorganshire	(11) 1,504
	Downside Colliery, Strap Pit, near Bath (disused)	(12) 1,520
	Kingswood Colliery, near Bristol	1,524
	Norley Collieries, Wigan, Lancashire	(13) 1,525
	Hoyland Silkstone Colliery, Barnsley, Yorkshire	(14) 1,530
	Barrow Colliery, Barnsley, Yorkshire	1,530
	New Sharlstone Colliery, Barnsley, Yorkshire	(15) 1,533
	Allen's Green Pit, near Radcliffe, Lancashire	(16) 1,539
	Parr Collieries, St. Helen's, Lancashire	(17) 1,548
	Malago Vale Colliery, Bedminster, Bristol	(18) 1,554
	Pleasley Colliery, Mansfield, Nottinghamshire	(19) 1,560
	Penrhiwceiber Colliery, Mountain Ash, Glamorganshire	1,581
	Boldon Colliery, near Sunderland, Durham	(20) 1,590
	Hickleton Main Colliery, between Barnsley and Doncaster	(21) 1,612
	Clifton Hall Colliery, near Manchester	(22) 1,620
	Langwith Colliery, Nottinghamshire	1,620
	White Moss Colliery, near Ormskirk, Lancashire	1,629
	Ryhope Colliery, West Pit, near Sunderland, Durham	(23) 1,630
	Albion Colliery, Cilfynydd, near Pontypridd, Glamorganshire	(24) 1,635

Country.	Colliery or Mine.	Depth in Feet.
Great Britain (*continued*).	Haydock Collieries, Princess shaft, Lancashire	(25) 1,638
	Great Laxey Lead Mine, Isle of Man	1,644
	New Tredegar Collieries, The Elliot Pits, Monmouthshire	(26) 1,644
	Houghton Main Colliery, Barnsley, Yorkshire	1,650
	Walsall Wood Colliery, South Staffordshire	(27) 1,650
	Newbattle Colliery, Lady Victoria Pit, near Edinburgh	(28) 1,656
	Bradford Colliery, Manchester	1,671
	Lindsay Pit, Wigan, Lancashire	1,680
	Capeldrae Colliery, Fifeshire	1,680
	Loanhead Colliery, Shott's Iron Company, near Edinburgh	1,700
	Braysdown Colliery, near Bath	1,704
	Bedlinog Colliery, near Dowlais, South Wales	1,716
	Collin's Green Colliery, St. Helen's, Lancashire	(29) 1,725
	Clifton and Kirsley Colliery, near Manchester	1,725
	Agecroft Colliery, Manchester	(30) 1,740
	Monkwearmouth Colliery, Sunderland, Durham	(31) 1,745
	Wombwell Main Colliery, No. 3 shaft, near Barnsley	(32) 1,761
	Silksworth Colliery, near Sunderland, Durham	(33) 1,770
	Great Fenton Colliery, Stoke-upon-Trent, N. Staffordshire	(34) 1,785
	Radstock Collieries, Middle Pit, near Bath	(35) 1,794
	Dinas Main or Britannic Colliery, South Wales	(36) 1,794
	Seaham Colliery, near Sunderland, Durham	(37) 1,797
	Mosely Common Colliery, No. 4 Pit, Bolton, Lancashire	1,800
	Shireoaks Colliery, Nottinghamshire	(38) 1,545, 1,590 and 1,815
	Hafod y Bwch Colliery, Ruabon, Denbighshire	(39) 1,815
	Giant's Hall Pit, Wigan, Lancashire	(40) 1,818
	Pendlebury Colliery, Manchester	(41) 1,818
	Bold Colliery, St. Helen's, Lancashire	(42) 1,851
	Lady Windsor Colliery, Ynysybwl, near Pontypridd	(43) 1,860
	Levant Mine (tin and copper), St. Just, Cornwall	1,896
	South Kirkby Colliery, Barnsley, Yorkshire	(44) 1,920
	Easton Colliery, Bristol	1,920
	Pemberton Colliery, Wigan, Lancashire	(45) 1,921
	Oldfield Colliery, Longton, North Staffordshire	(46) 1,923
	Carn Brea Mine (tin and copper), Redruth, Cornwall	1,932
	Hampstead Colliery, South Staffordshire	(47) 1,950
	Abram Colliery, Leigh, Lancashire	(48) 1,950
	Universal Colliery, Aber Valley, South Wales	(49) 1,950
	Shelton Collieries, Hanley, North Staffordshire	(50) 1,950
	Wigan Junction Colliery, near Wigan, Lancashire	(51) 1,950
	Sneyd Colliery, Burslem, North Staffordshire	(52) 1,959
	Glebe Pits, Stoke-upon-Trent, North Staffordshire	1,998
	Niddrie Colliery, Portobello, Edinburgh	2,010
	Longton Hall Colliery, North Staffordshire	(53) 2,013
	Park Lane Colliery, Wigan, Lancashire	(54) 2,061
	Bardsley Colliery, near Ashton-under-Lyne, Lancashire	(55) 2,100
	Ackton Hall Colliery, near Wakefield	(56) 2,160
	Kiveton Park Collieries, near Sheffield, Yorkshire	(57) 2,190
	Moss Collieries, No. 4 Pit, near Wigan, Lancashire	(58) 2,205
	Bickershaw Colliery, Leigh, Lancashire	(59) 2,210
	Dowlais Cardiff Colliery, Abercynon, South Wales	(60) 2,220
	Cadeby Main Colliery, near Doncaster, Yorkshire	(61) 2,253
	Harris' Navigation Colliery, near Pontypridd	(62) 2,279
	Alexandra Colliery, Wigan, Lancashire	(63) 2,337
	Cook's Kitchen, Cornwall	2,436
	Rosebridge Colliery, Wigan, Lancashire	(64) 2,446
	Florence Colliery, Longton, North Staffordshire	(65) 2,571
	Dolcoath Mine (tin and copper), Camborne, Cornwall	(66) 2,582
	Astley Pit, Dukinfield, near Manchester (workings)	(67) 3,150
	Ashton Moss Colliery, near Manchester (workings)	(68) 3,360
	Pendleton Colliery, Manchester (workings)	(69) 3,474
Norway.	Kongsberg (silver)	1,900

(1) The average winding speed in the shaft is 3,200 feet per minute.

(2) A pair of quadruple engines is used for winding, each with its four cylinders, 18 inches, 27¾ inches, 48 inches, and 90 inches, in diameter, and stroke of 60 inches, driving, through gearing, conical winding drums, 14 to 24 feet in diameter and 12 feet wide, which lift cages carrying 6 tons of ore at the rate of a mile in one and a half minutes.

(3) 25 gold mines in Victoria exceed 2,000 feet in depth.

(4) At the deep shafts of Przibram tapering wire ropes made of special crucible cast steel, having a tensile strength of 114 to 120 tons per square inch, are used for winding.

(5) Cécile Pit 14¾ feet diameter. There are nine shafts in Belgium exceeding 2,610 feet in depth. The average depth of all Belgian collieries is 1,420 feet.

(6) At the Providence shaft, Marchienne, flat tapered ropes made of crucible cast steel are used for winding. The breadth of the rope varies from 7·87 inches at the thick to 6·69 inches at the thin end, and the average weight is 8·2 lbs. per foot, and last about 12 months. The winding engines, of 2,000 horse-power, have cylinders of 43 inches in diameter and 78 inches stroke, and raise a load of 12½ tons from a depth of 3,117 feet. The load consists of 6½ tons, the weight of the cage and tubs and 6 tons of coal.

(7) At the Sainte Henriette shaft, flat aloe ropes are used to lift a load of 6¼ tons from a depth of 3,937 feet. The ropes taper in breadth from 16·5 inches to 8·6, and in thickness from 1·93 inch to 1·14. The average weight per foot is 7·4 lbs. The ropes last about 24 months.

(8) Pneumatic hoisting apparatus successfully applied for ten years, thus altogether dispensing with the use of winding ropes. A tube, 5 feet 3 inches in diameter, fitted with a piston, was fixed in the shaft. The piston was attached to a nine-deck cage, carrying 9 tubs, each holding 10 cwt. of coal. To raise the loaded cage the air above the piston was exhausted, the atmospheric pressure beneath then gradually forcing up the load. For the descent, exhaustion was stopped and air allowed to pass upon the top of the piston. The tubs were removed from the cage three at a time by means of three double doors provided in the tube both at the top and the bottom.

(9) The west shaft is an upcast 12 feet in diameter, sunk 1,500 feet and bored about 300 feet further to prove the Kilburn and other seams of the Lower Coal Measures.

(10) No. 3 is an upcast shaft 16 feet in diameter and 1,500 feet in depth to the Five-feet Seam. It was sunk in 1894 and 1895. The shaft lining consists of 9-inch brickwork set in Aberthaw lias lime. In places the brickwork is from 18 to 24 inches in thickness.

(11) 1,449 feet to Six-feet Seam loading stage, there being 55 feet of sump. There are two shafts sunk, each being 15 feet in diameter inside the walling.

(12) The sinking of this shaft was continued from the depth of 660 feet to that of 1,520 feet, the deepening having been completed December 9, 1874. The strata dip north at an angle of 40°.

(13) To the Arley Mine or Orrell four-feet Seam. No. 4, an upcast and winding shaft, is 12 feet in diameter; No. 5, a downcast and winding shaft, is 16 feet in diameter, and 292 yards south-east of No. 4. The shafts are lined with ordinary bricks and mortar without the insertion of any cast-iron tubbing. Considerable feeders of water were encountered in sinking the No. 5 shaft, which are still being raised to the surface.

(14) 20 feet diameter.

(15) To Haigh Moor Seam.

(16) To the Cannel Mine, which is 216 feet below the Trencherbone Seam. The shaft is a downcast 11½ feet in diameter, lined throughout, there being 1,013 feet of 9-inch brick walling from the surface, then 67 feet of cast-iron tubbing and 459 feet of brick walling to the bottom.

(17) Two shafts, Nos. 4 and 5, each 18 feet in diameter, are sunk 1,548 feet to the Arley Mine.

(18) To the Bedminster Little Seam. There are two shafts, situated 66 yards apart. The downcast, called Malago, or Old Pit, is 14 feet in diameter, and 1,500 feet in depth to the shaft inset from which a cross-measure drift or branch is driven to cut the Bedminster Great Vein. The upcast, called Argus, or New Pit, is a furnace shaft 15 feet in diameter, and 1,740 feet in depth. From the bottom of this shaft a level cross-measure drift is driven. Both shafts are walled throughout, and fitted up for the daily raising and lowering of persons and the raising of minerals.

(19) Two shafts, 80 yards apart from their centres, each 14½ feet in diameter, are sunk 1,542 feet to the Top Hard Seam. The sinkings pierced 250 feet of Magnesian Limestone or Permian strata, through which and water-bearing strata below, considerable feeders of water issued, the maximum amounting to nearly 1,000 gallons a minute. The lining of the shafts consists of 54 feet of 18-inch brickwork from the surface, followed by 351 feet of cast-iron tubbing, and then 1,137 feet of 9-inch red brick walling to the floor of the Top Hard Seam.

(20) To the Hutton Seam; 1,458 feet to the Bensham Seam.

(21) Two shafts, 65 yards apart, are sunk 1,612 feet to the Barnsley Seam. Both shafts are 18 feet in clear diameter and lined as follows:—9 feet of 9-inch red brickwork set in cement from the surface, then 150 feet of cast-iron tubbing, followed by 1,453 feet of ordinary red brickwork, the thickness of which varies from 9 to 14 inches. The sinking was rapidly performed. Begun in December, 1892, it was completed in June,

1894, this period of nineteen months including the placing of pumps and the insertion of one wedging crib, tubbing and walling in each shaft. The shaft at which the pumping engine was placed was continuously kept deeper than the other, which was consequently more or less drained by the same engine.

(22) 1,605 feet to the Trencherbone Seam. The upcast shaft is 9 feet in diameter and the downcast 10 feet. Both shafts are lined throughout, there being about 300 feet of cast-iron tubbing in each, the remainder consisting of ordinary 9-inch brickwork.

(23) To the Hutton Seam. A boring was made from the Hutton Seam in the shaft to a depth of 114 feet, the shaft and boring together thus proving 1,744 feet of strata from the surface.

(24) There are two shafts, each being 19 feet in clear diameter, walled throughout. The sinking commenced in January, 1885, and was finished in February, 1887. On Dec. 21, 1887, 1,000 tons coal were raised.

(25) To the Arley Mine. Shaft 17 feet in clear diameter, belled out to 20 feet near the bottom.

(26) The east shaft is a downcast 18 feet in diameter and 1,590 in depth to the Lower four-feet Seam, there being 54 feet of sump. The sinking was rapidly performed. Commenced in 1888, it was completed to the Lower four-feet Seam in 15 months, including the lining of 9-inch brickwork set in Aberthaw lias lime. There was no trouble with the water, because it drained to the west shaft either naturally or through drifts. The west shaft is an upcast, also 18 feet in diameter, 43 yards distant from the east shaft. Begun in 1883, the sinking reached the Red Seam, 1,320 feet from the surface, 23 months afterwards. Later a shaft was sunk 270 feet further to give a connection between the Lower four-feet Seam and the upcast. Various beds of sandstone were penetrated in sinking, and these yielded water, but no attempt was made to keep such water back by cast-iron tubbing or water-tight walling. The west shaft was lined nearly throughout with firebrick, 9 inches long, tapered from 5¼ to 6 inches in breadth at the back and 4 inches deep, set in Aberthaw lias lime. The space between the back of the 9-inch brickwork and the sides of the shaft was filled up with rubble stones without mortar. Open spaces were left in the rubble 3 inches square at intervals around the shaft, which allowed the water to drain into garlands of cast iron placed about every 15 yards in depth, the shaft being gradually enlarged for their reception.

(27) To the Yard Seam. Two shafts, each 15 feet in clear diameter, are sunk at a distance of 123 yards apart. The No. 1, or north shaft, is a downcast and has three sections of cast-iron tubbing inserted, the entire length of which is 360 feet. The No. 2 or south shaft is an upcast and winding shaft; it has two sections of cast-iron tubbing, making together 240 feet in length. The remaining portions of the shafts are lined with ordinary brickwork.

(28) To a horizontal drift to cut the Jewel Seam, which was intersected in the shaft 1,548 feet from the surface. Inclination of the measures 1 in 2·3. The sinking of the shaft, 20 feet in clear diameter, was begun in March, 1890, and completed in December, 1893. Some feeders of water were kept back from the shaft by means of concrete walling faced by 9 inches of red brickwork. By means of a movable steel scaffold suspended on wire ropes, the masons' work of lining the shaft with bricks proceeded simultaneously with the excavations in the pit bottom.

(29) 1,692 feet to the Lower Florida Seam; 33 feet of sump. No. 1 is a downcast shaft 16 feet in diameter, and No. 2 is an upcast shaft 14 feet in diameter, the two shafts being of the same depth.

(30) Two shafts are sunk, 22 yards apart from centres, each 9½ feet in diameter, and 1,740 feet in depth to the Doe Seam. The sinking of the shafts occupied a period from 1852 to 1856. The lining consists of 50 feet of brick walling set in cement from the surface, then 450 feet of cast-iron tubbing, following which are 1,240 feet of ordinary 9-inch brick walling. In 1896, two new shafts were being sunk; No. 3, a downcast 15 feet in clear diameter, and No. 4, an upcast 14 feet in clear diameter, 30 yards north of No. 3 shaft. The estimated depth to the Trencherbone Seam at these new shafts was 2,250 feet. In the No. 3 shaft a stratum of quicksand 4 feet in thickness, 8 feet below the surface, was piled through, a double row of piles having been used.

(31) 1,701 feet to the Hutton Seam, below which are 44 feet of sump. A boring was made from the bottom of the sump and proved the Harvey Seam 117 feet below or 1,862 feet from the surface. The Maudlin Seam was cut in the shaft 1,589 feet from the surface. The Monkwearmouth pits are notable as being amongst the foremost which pierced the Magnesian Limestone before reaching the Coal Measures. The sinking, which was commenced in May, 1826, did not reach the coal measures until August, 1831. The thickness of strata over the coal measures is 330 feet, of which 117 feet are alluvial, and 213 feet Magnesian Limestone. At the bottom of the limestone, the feeders of water amounted to 3,000 gallons per minute. These were successfully dealt with during the sinking, and effectually kept back by means of cast-iron tubbing. For many years, this colliery was regarded with great interest, both from a scientific and a commercial point of view, because of the speculative nature of the enterprise, and the indomitable perseverance shown in sinking a

shaft which was, for many years, the deepest in the world. It has been stated that, when completed, the undertaking, although a single shaft, had cost between £80,000 and £100,000. The face-workings are now more than 2 miles from the shaft, and afford a good example of work carried on in a high temperature due to great depth of shaft and long communicating passages. The colliery was one selected for hygrometric observations by the Royal Commission appointed to inquire into the several matters relating to coal in the United Kingdom, whose report was published in 1871.

(32) No. 3 shaft, 12½ feet in diameter, was sunk in 1862 to the Barnsley Seam and afterwards deepened to the Parkgate Seam. In 1884 the shaft was sunk still further and reached the Silkstone Seam at the depth of 1,761 feet from the surface. The shaft is lined chiefly with 9-inch brickwork.

(33) 1,740 feet to the Hutton Seam, there being 30 feet of sump; 1,620 feet to the Maudlin Seam. No. 2 is a downcast shaft 16½ feet in diameter, divided by 3-inch wooden brattice into two equal parts, the two parts forming separate winding compartments from the Maudlin and Hutton Seams. Sinking operations were commenced at this shaft in December, 1869, and completed May 1, 1874. No. 1 is an upcast shaft, its centre being 64 yards distant from the centre of No. 2 shaft. The sinking of No. 1 shaft, which is 14 feet in diameter, was begun in August, 1869, and completed April 4, 1873.

(34) To Ash Seam, Homer and Sutherland Pits, which have been working since about 1880; 924 feet to the Great Row Seam at the Pender and Bourne Pits. The Pender is 13 feet in diameter and the Bourne 8 feet, these two having been in operation since 1876.

(35) An oval shaft originally sunk 872 feet, first through New Red Sandstone to the Coal Measures lying unconformably beneath, and then on through the Radstock series of seams to the lowest or Nine Inch Vein, which is unworkable to profit. The sinking is notable as having proved the Great Overlap Fault by twice passing through the Slyving, Under Little, and Bull Veins. The difference of level between the same seam in the shaft is over 200 feet. A single cage was used in the shaft, it being loaded with "puts" or sledges of coal or rubbish, which were brought from the workings on iron frames with wheels, but taken therefrom on reaching the shaft. In 1884 and 1885 the shaft was deepened to the sixth seam of the Farrington series, 1,794 feet from the surface. The deepened portion is circular in form, being 10 feet in clear diameter.

(36) An instance of rapid sinking, a single shaft having been sunk and walled throughout 18½ feet in clear diameter to the depth of 1,566 feet to the Five-feet Seam in sixteen months, while another of the same diameter was sunk to the depth of 1,794 feet in a little over nineteen months.

(37) The downcast shaft, 14 feet in diameter, is divided into two equal parts by 3-inch wooden brattice, thus forming two winding compartments, named No. 1 and No. 2. This shaft, commenced in April, 1849, was in order for the raising of coal in 1852. It was sunk to the depth of 1,797 feet, having cut the Harvey Seam at the depth of 1,688 feet and the Busty Seam at the depth of 1,744 feet. The upcast or No. 3 shaft is 220 yards distant from the downcast. It was begun in August, 1845, and sunk to the depth of 1,613 feet, having reached the Hutton Seam at the depth of 1,513 feet. The size of this shaft was reduced by an inner lining of tubbing from 14 to 13 feet diameter in 1869, because of the decay of the first tubbing. The lining of the upcast shaft consists of 212 feet of freestone blocks, 12 inches in the bed, set in cement, followed by 306 feet of cast-iron tubbing, then 295 feet of freestone walling similar to that at top, 48 feet of cast-iron tubbing, and then below this to the bottom the lining is of freestone blocks.

(38) The cage, with a load of 34 cwt. in five tubs, is raised from a depth of 1,545 feet in 45 seconds, an average speed of 2,100 feet per minute.

(39) Shafts sunk 1,815 feet in the year 1869 to the Main Coal Seam.

(40) To Cannel Mine.

(41) 1,781 feet to the Doe Coal Seam. A downcast shaft 10½ feet in diameter, and an upcast 9½ feet, were sunk 24 yards apart in 1846 to the Ram's Mine, a depth of 1,194 feet. About 1892 the downcast was deepened to the Doe Seam. Both shafts are lined with ordinary 9-inch brickwork from the surface to the depth of 162 feet; beneath that there are 312 feet of cast-iron tubbing, and from thence to the bottom the lining consists of 9-inch brickwork.

(42) To the Lower Florida Seam in No. 3 shaft; 1,821 feet in Nos. 1 and 2 shafts. No. 1 is an upcast shaft 12 feet in diameter in the clear, No. 2 a downcast 16½ feet in diameter.

(43) Two shafts were commenced in 1884, and completed after 20 months' sinking operations. The downcast shaft is 19 feet in diameter and 1,680 feet in depth to the Four-feet Seam upper loading stage, and 1,800 feet in depth to the lower loading stage in another seam. The upcast shaft, 82 yards west of the downcast, is 17 feet in diameter and 1,860 feet in depth, including 60 feet of sump. Only a little water had to be raised during the sinking, and pumps were not required. The shafts are lined throughout with ordinary firebricks set in Aberthaw lime, 9 inches in thickness and upwards.

(44) There are two shafts, 50 yards apart, each 15 feet in diameter, 1,905 feet in depth to the Barnsley Seam, 9 feet 7 inches thick. The lining of the shafts consists of 360 feet

of cast-iron tubbing from the surface downwards, and 1,545 feet of brickwork to the floor of the Barnsley Seam. The bricks are tapered and 9 inches and upwards in the bed.

(45) 1,888 feet to the Arley Mine; 1,702 feet to the Orrell five-feet Seam. The King shaft, 89 yards distant from the Queen shaft, is the upcast and is 18 feet in diameter. The Queen is a downcast shaft 16 feet in diameter. The sinking of these shafts was commenced on December 2, 1867, and completed to the Arley Mine in the Queen shaft on September 2, 1870. The time occupied in sinking and tubbing with 207 feet of cast-iron segments in the King shaft and 158 feet in the Queen shaft was 2 years and 3 months. In the King shaft the tubbing commences about 150 feet down and extends below the Pemberton five-feet Seam. Except where tubbed, the King shaft is lined throughout with firebrick lumps 11 inches long and 6 inches deep. The Queen shaft has similar firebrick lining of 10 inches length.

(46) 1,869 feet to the Yard Seam. Two shafts, 13 yards apart, each 12 feet in diameter in the clear, were commenced in 1840 and sunk originally to work the Great Row Seam at the depth of 407 feet from the surface. These shafts were deepened to the Yard Seam in 1885, and are lined throughout with 9-inch brickwork. They pass wholly through Coal Measures.

(47) 1,860 feet to the Thick Coal Seam. Two shafts have been sunk to the same depth, one as a winding and downcast shaft, 15 feet in diameter, and the other as an upcast and Pump shaft, 45 yards distant, and 12 feet in diameter. The estate of 600 acres, in the centre of which the shafts are placed, is freehold, and was acquired by the Hampstead Colliery Co. for £101,000. The sinking commenced June 29, 1875, and was completed April 17, 1880, having lasted nearly 5 years. About 1,300 tons of coal are raised per day of 8 hours by the one pair of winding engines. The shafts were begun very near the top of the "red rocks" and not far below the base of the middle Bunter, and passed through 1,353 feet of red, purple and brown sandstones and marls before the ordinary Coal Measures were reached. These "red rocks" of the West Bromwich district are considered by some to be of Upper Coal Measure age, while others think they should be classified as Permian.

(48) To the Arley Mine. Shaft 18 feet in diameter.

(49) An instance of rapid sinking; one shaft, 18½ feet in clear diameter, was sunk to the depth of 1,950 feet and lined throughout with brickwork of a minimum thickness of 9 inches in a little under twenty-seven months.

(50) The Deep Pit, 11½ feet in diameter, is sunk 1,950 feet to the Hard Mine coal. No. 1 and No. 2 Rowhurst shafts, 9 and 12 feet in diameter respectively, are sunk 1,590 feet, having passed through the Rowhurst Seam at the depth of 1,380 feet. Stone drifts are driven at the 1,590 feet level to intersect the Rowhurst Seam.

(51) To the Arley Mine. The downcast shaft is 14 feet in diameter, and the upcast, placed 40 yards north of the downcast, is 12¼ feet in clear diameter. Both shafts are lined throughout, chiefly with 9-inch red brickwork and ordinary mortar. In weak strata 18 inches of brickwork was built.

(52) No. 2 shaft is 13½ feet in diameter and 1,863 feet in depth, to the insets from which stone drifts are driven to intersect seven seams of coal. No. 1 shaft is the upcast and 12 feet in diameter, being 1,836 feet in depth to the cross-measure drifts for the air-returns. These shafts were sunk in 1849 to work the Winghay or Knowles Coal Seam at the depth of 570 feet, and were enlarged and deepened in 1887.

(53) No. 2 shaft, 15 feet in diameter, is sunk to the depth of 2,013 feet to the Moss Seam.

(54) A downcast and an upcast shaft, each 16 feet in diameter were sunk, 2,061 feet to the Arley Mine or Orrell four-feet in 1877. Both shafts are lined throughout with 9-inch brickwork and ordinary mortar.

(55) To the Royley Mine, which proved only 2 feet in thickness and so was not worked. The lowest seam worked is at the 1,635 feet level.

(56) To Beeston Seam; 1,806 feet to the Silkstone Seam. No. 1 shaft, 18 feet in clear diameter, was commenced in October, 1892, and completed about the beginning of 1895. No. 2 shaft is 16 feet in diameter.

(57) Two shafts, 90 yards apart, were commenced in May, 1866, and completed to the Barnsley or Top Hard Coal Seam in December, 1867. No. 1 shaft is 13 feet in diameter and 1,203 feet in depth, while No. 2 is also 13 feet in diameter and 1,200 feet in depth to the Barnsley Seam. Both shafts are lined with cast-iron tubbing commencing near the surface and extending for a depth of 300 feet and below with red brickwork 9 inches in thickness or more. In 1888 the shafts were deepened to the Silkstone Seam and walled throughout, the seam being reached at 2,190 feet from the surface. Both shafts are now used for winding coal.

(58) 2,130 feet to the Arley Mine; 1,650 feet to the Cannel Mine. No. 4 shaft is 15 feet in diameter, lined with 9-inch brickwork set in ordinary mortar; in faulty ground and weak strata the brickwork is increased as required.

(59) Shaft 18 feet in diameter and sunk to the Wigan six-feet Mine.

(60) Two shafts, 65 yards apart, centre to centre, and each 20 feet diameter in the clear, are sunk 2,220 feet to the 9-feet seam, and lined, one shaft throughout and the other nearly so, with brickwork of a minimum thickness of 12 inches. The sinking occupied a period from 1890 to 1896. The Pennant rock was reached after passing through 102

feet of alluvial deposit. Water was kept back from this deposit by walling. A carefully levelled bed was prepared in the Pennant rock, 18 feet below the bottom of the alluvial deposit, to form a foundation for the watertight walling. A solid wall, 4 feet 6 inches in height, was first built, after which cast-iron blocks, with 1½-inch perforations, were placed at certain distances apart round the shaft. As the building of the wall was continued, a wooden pipe was led from the perforation in each cast-iron block, which served to convey the water to the shaft bottom, from whence it was afterwards pumped to the surface. Above the walling in which the blocks were inserted the building consisted of bricks and cement 3 feet 2 inches in thickness. This was made up of an inner ring 2 feet thick and a concentric ring 1 foot thick. The annular space of 2 inches between these two rings was filled up with cement and sand, the 1-foot wall being backed up solid against the sides of the shaft. This walling is carried up nearly to the surface. After the completion of the building some months were allowed to elapse in order that the cement might set, and then the cast-iron perforations were plugged and the water effectually shut out from the shafts. The south shaft is lined with cast-iron tubbing 2½ inches thick between the depths of 236 and 390 feet from the surface. At the bottom of the tubbing the pressure was over 100 lb. to the square inch. In the north shaft the same strata that were tubbed in the south shaft yielded less water, and were walled through by watertight brick and cement walling. A pair of 42-inch diameter cylinders with 7-foot stroke and Cornish valves is used to wind coal at the north shaft. Two trams are placed, one in front of the other, in a single-decked cage. The trams are made of steel, the weight of each being 12½ cwt., and the carrying capacity 40 cwt. of coal. The gauge of tram roads is 3 feet 2 inches. The time occupied in winding is 52 seconds, and in changing, 8 seconds. About 1,000 tons of coal per day are raised. It is intended to wind coal also at the south shaft.

(61) Two shafts, 87 yards apart, each 16 feet diameter in the clear, are sunk 2,253 feet to the Barnsley Bed, 9 feet 8 inches in thickness. The sinking of the shaft was commenced in April, 1889, and completed in November, 1892. The lining of No. 1 shaft, which is the downcast, consists of 411 feet of cast-iron tubbing from the surface, and below this 1,842 feet of ordinary red brickwork and mortar 9 inches to 24 in thickness. There was difficulty in sinking through loose boulder clay from the surface downwards, which was piled through.

(62) Two shafts, each 17 feet in clear diameter, were sunk in the period between 1875 and 1882. No. 1, or south shaft, is the downcast and is 2,082 feet in depth to the Upper four-feet Seam. No. 2, the upcast shaft, is placed 58 yards north of No. 1, and is 2,157 feet in depth to the Six-feet Seam. Both shafts are lined with firebricks, set in Aberthaw lime, the thickness varying from 9 inches to 18. During the sinking no efforts were made to keep back the water either by cast-iron tubbing or water-tight walling.

(63) To the Arley Mine. The Alexandra shaft is 19 feet in diameter in the clear, and was sunk to the depth of 771 feet about the year 1874 to the Pemberton four-feet Seam. From 1885 to 1887 the shaft was deepened on the same diameter from the Pemberton four-feet Seam to the Arley Mine. The time occupied in the deepening was nearly 2 years. The lining of the shaft consists of red bricks set in mortar made entirely from blast furnace slag, chiefly 9 inches in thickness, but in faulty parts 6 feet of solid brickwork was built against the side walls of the shaft. Mr. Bertram's "Visor" apparatus to prevent over-winding, is in use in connection with the winding engine. The time occupied by one winding from a depth of 1,930 feet is 45 seconds, and of changing 6 tubs 18 seconds, in all 63 seconds, so that there are about 56 windings per hour.

(64) 2,420 feet to the Arley Mine, there being 26 feet of sump. The greatest speed of the cage in the shaft recorded at Rose Bridge Colliery is 5,100 feet per minute or about 57 miles an hour.

(65) 2,490 feet to the Yard Seam; 2,214 feet to the Moss Coal and 1,056 feet to the Great Row Seam. Two shafts, 52 yards apart, were sunk to the Moss Seam, and commenced working coal in 1877, the sinking operations having taken about 5 years. No. 2 is a downcast shaft 14 feet in diameter. No. 1 is an upcast 12½ feet in diameter to the Moss Coal. About 1892 No. 1 shaft was deepened to the Yard Seam, the shaft being belled out to 17 feet in diameter at the Yard Seam. Later No. 2 shaft was deepened also.

(66) 2,400 feet on underlie of lode.

(67) Depth of shaft 2,095 feet; to shaft inset, from which a stone drift is driven to cut the Black Mine 2,059 feet. There are two shafts 50 yards apart, the downcast being 12 feet in diameter in the clear and the upcast 11 feet. The shafts are similarly lined, the lining consisting of 12 feet of ashlar walling, 18 inches in the bed, set in Portland cement; 528 feet of 9-inch ordinary brickwork; about 150 feet of cast-iron tubbing; 1,369 feet of 9-inch brick walling to the shaft insets; 36 feet of 9-inch brickwork to the bottom of the sump. The sinking of the shaft to this depth occupied 10 years; the outlay on sinking and plant amounting to about £100,000.

(68) There are two shafts, 60 yards apart from centres, each being 16 feet in clear diameter. Sinking commenced in March, 1875, and was completed in May, 1882. The depth to the Saltpetre Seam is 2,684 feet,

to the Black Mine 2,852 feet, while the total depth of the shafts is 2,880 feet. The seams dip 1 in 4. Barometer readings, March 1, 1887, were:—

At the surface 30·10 inches.
At bottom of the shaft 33·30 „

Time occupied in drawing loaded cage up the shaft 1 min. 25 sec.; time occupied in changing 35 seconds. Boiler fires at the bottom of the upcast shaft and a furnace placed midway between top and bottom produce the ventilation, amounting to about 100,000 cubic feet per minute. Very little firedamp is given off in the workings.

(69) Depth of shaft, 1,575 feet to the Ram's Mine. From the bottom of the shaft an incline has been driven a distance of 4,700 feet, dipping about 1 in 3. The downcast shaft is 8 feet in diameter, the upcast 7½ feet. A furnace is placed at the bottom of the latter.

At the commencement of sinking the Ashton Moss shafts, which are the deepest in the United Kingdom, difficulty in the form of an alluvial deposit presented itself. This consists of different layers of clay, clay-marl, gravel, and quicksand, and extends from the soil immediately under the surface to coal-measure rock, the distance from the surface being 49 yards, 2 feet, 7 inches. The layers of gravel and quicksand were copiously watered, and the whole covering over the coal-measures was of the loose character peculiar to such deposits. Shafts in other parts of the country have encountered similar alluvial deposits caused by the gradual settlement of sediment from flowing waters. These deposits vary in character, but are mostly of clay, silt, gravel, or sand. If the sand contains much water it becomes a quicksand, and on being tapped, runs into the pit from all sides and gives much trouble. Sometimes large boulders or pebbles are embedded in the clay, and sometimes sand and gravel are intermixed. At the Norwood Colliery, near Gateshead, in the county of Durham, beneath a few inches of soil, about 100 feet of alluvial deposit was pierced, the top layer being of dark mud with vegetable matter, closely resembling peat.

Quicksands are not only troublesome in the sinking of a pit, but also a source of danger to machinery and buildings on the surface, because of the insecurity of foundation. Any extensive rush of quicksand into the sinking shaft at no great depth below the surface, or any gradual up-rising of the pit bottom, is almost certain to be followed by a subsidence of the surface and possibly serious damage to costly machinery. Poetsch's freezing system already described has been most successfully applied under these circumstances.

The New Red Sandstone contains soft beds of sandstone, which have presented great difficulties to those sinking shafts through them in order to win the coal measures. The soft sandstones are heavily watered, and the force of flow into the sinking shaft carries down disintegrated portions which settle at the bottom of the shaft. Here they impede the working of the pumps, by wearing the bucket leathers, and otherwise add to the labour of sinking, inasmuch as the running sand has to be removed.

The method of conducting sinking operations through the drift or alluvial accumulation at the Ashton Moss Colliery is shown in Figs. 47, 48, and 49.

These, however, do not show the alternations of clay, gravel, and quicksand in the drift. The shaft was carried down through the overlying material 24 feet in diameter inside the 9-inch brick lining shown in the Figs. until it reached the top bed of the coal-measures. During the sinking this 9-inch ring of brickwork was continued from the surface downward in sections, as it was required. In order that it might be upheld securely, the walling was entirely suspended from the surface, four massive wood beams being placed across the top of the shaft to carry the weight, which was distributed by means of 12 wrought-iron hanging rods 2½ inches in diameter. From this point the sinking was resumed at a reduced diameter of 17 feet 6 inches, and continued until a sound bed was reached which would carry the permanent walling crib. This was then carefully laid and a 9-inch ring of brickwork built upon it. When this reached the bottom of the alluvial deposits where the shaft widened, a 4½-inch brick ring was started off the coal-vapour, and is frequently saturated. In many cases, owing to the dryness of workings at great depths, means are taken to saturate the air as it travels from the shaft in order to lay the dust. If the only consideration were that of the power of the workmen to bear a high temperature, the very opposite course should be adopted, and some method be devised to deprive the entering air of moisture. If this were done, it would ensure a moderately dry atmosphere reaching the face, and by assisting free evaporation from the body would enable the workmen to endure a higher temperature than would be possible in a moist atmosphere. But the drying of the air would also cause the roadways to be filled with dangerous clouds of fine coal-dust, which can at present be prevented only by frequent watering, or by the destruction of the coal-dust.

As the temperature of a moist atmosphere rises above 80° F., the less labour will it be possible for miners to perform in it, and the higher will be the cost of working. The normal heat of the blood in the human body is 98° F. In atmosphere a little higher than this, work can only be carried on at all in short periods, the shifts of workmen being relieved by removal to a cooler region.

As questions are sometimes given at the examinations, having reference to the cubical contents obtained from sinking shafts, the following example is given and worked out in order to show candidates how to do similar calculations.

> *Question* 1.—A pit is sunk 17 feet 6 inches in diameter, and walled with good bricks 13 inches in the bed, 6 inches deep by 12 inches long inside, but more at the back—being moulded to suit the circle of the pit—the diameter of the shaft when the walling is finished being 15 feet in the clear. The pit is 100 fathoms deep.—How many cubic feet of excavation would be taken out, and assuming 14 cubic feet of it to weigh a ton, state the total weight? Also, if walled from top to bottom, how many bricks of the above dimensions would be required, and how much would they cost at £5 per thousand?

Here we have to get the area of a circle whose diameter is 17 feet 6 inches, and to multiply it by the depth of the pit in feet to get the cubical contents.

$$17{\cdot}5 \times 17{\cdot}5 \times {\cdot}7854 \times 600 = 144{,}317 \text{ cubic feet,}$$

$$\text{and } \frac{144{,}317}{14} = 10{,}308{\cdot}38 \text{ tons.}$$

To find the number of bricks we must find the circumference of a circle whose diameter is 15 feet; $15 \times 3{\cdot}14159 = 47{\cdot}124$, and if we allow $\frac{1}{4}$ of an inch for mortar at the joints we should require $\frac{47{\cdot}124 \times 12}{12{\cdot}25} = 46{\cdot}16$, the number of bricks which we should require for one ring; allowing $\frac{1}{4}$ of an inch for the horizontal joints the pit would have $\frac{100 \times 6 \times 12}{6\frac{1}{4}} = 1{,}152$ rings. Therefore $1{,}152 \times 46{\cdot}16 = 53{,}176{\cdot}32$, the number of bricks required, and $\frac{53{,}176{\cdot}32 \times 100}{1{,}000} = 5{,}317{\cdot}632$ shillings = £265 17*s.* 7*d.* as the cost of the bricks.

CHAPTER V.

FITTING UP THE SHAFT AND SURFACE ARRANGEMENTS.

Arrangement of Pit Bottom for Small and Large Trams—Shaft Gates—Conductors—Buntons—Keeps—Pit Cages—Safety Cages—Detaching Hooks—Pit Head-gear—Pulleys—Ropes—Capping Round and Flat Ropes—Observations for Users of Ropes—Tables of different qualities of Round and Flat Ropes and of Chains—Method of Splicing Ropes—Shaft Signals—Pit Stage—Tipplers—Screens and under Railways—Winding Engines—Conical and Spiral Drums—Steam-brake to prevent over-winding—Counterbalancing the Load in Shaft—Rules for Winding Engines—Calculations of Sizes required under given Conditions—Questions and Answers on Steam and Steam-engines—Systems of Winding Coal up Shafts without using Drums.

The shaft bottom and roadways, for some distance, leading from the pit bottom are generally arched. Where small trams are to be used the space round about the shaft bottom is usually laid with flat sheet-iron for facilitating the operations.

Where large trams are used rails are laid leading to each cage from opposite directions. This allows of the empty trams being propelled from the cage on one side as the loaded ones enter it at the other.

Where flat sheets are used, they allow of light full tubs, or the lighter empties, being quickly turned in any direction without having to follow a particular course.

The pit is sunk a few feet below the level of the flat sheets to form a sump, and into this the water (if any) drains; from thence it is raised direct by the pumps placed in the shaft or conveyed elsewhere to be dealt with. The pit bottom is arranged so that the loaded tubs are pushed towards the cage down a slightly falling road, and the empty tubs pass out of the cage on the opposite side of the shaft.

The top of the pit and any intermediate loading places between the top and bottom, are provided with gates for the protection of those moving about.

Shafts are fitted with Conductors or Guides, which, if of wood or iron, are attached to buntons or crosspieces fixed across the pit and which have either been built into the walling or are afterwards let into it. The strength of the buntons must be proportioned to the size of shaft and the weight of the load; for a shaft 10 feet in diameter with single cages carrying one tram of 12 or 15 cwts., Memel or red pine, 9 inches by 3 inches, placed at intervals of six feet in the shaft, would be sufficient. The guides (if of wood) should also be of Memel pine, not less than 4 inches by 3 inches in section, and properly bolted to the buntons. Bolts and nuts are preferable to wood screws which are often used for this purpose. There is usually only one guide on each side of the cage, but the arrangements respecting them are various, according to the requirements of the case. Frequently, instead of wood, bridge or single headed rails are used for guides, and in some cases angle iron, they being kept in line by suitable fish-plates and bolts, and securely fastened by bolts to the buntons. In Lancashire and Yorkshire some pits have guides consisting of round bars of iron fixed at the pit bottom and screwed up to the head frame. There are two rods for each cage, the cross bar of which, having a ring at each end, runs upon the rods.

In most of the large collieries in South Wales wire-ropes are used as guides, fixed to wooden balks at the shaft bottom and to the head frame, where they are tightened by screws; another means of keeping them tight is to suspend heavy weights from their lower extremities beneath the balks, or by weights hanging over pulleys on the surface.

Where the depth and consequently the cage-speed is great, three and sometimes four of these guides are required to prevent excessive vibration. In some instances two additional ropes are suspended between the cages to prevent one cage from catching the other in passing. Rigid guides are so fixed that the cages shall have not less than 9 inches of clearance as they pass each other, and if iron wire guides are used and the pit a deep one there should be from 12 inches to 18 inches of clearance, according to the depth of the shaft, the number of guides used, and the speed of the cages in the pit.

"KEEPS," "FANS," or "SHUTS" are supports for the cage on its arriving at the surface or shaft bottom, and at intermediate loading places, if there be any. They are arrangements of counterbalanced levers, and those placed on the pit top offer no obstacle to the ascent of the cage, which after passing by the "keeps" is lowered by the engine-man on to the supports. With double-decked cages, when the tub on the bottom deck has been changed and a signal received that the tub in the top deck (which it must be remembered stands on the shaft bottom "keeps," when the bottom deck of the other cage is on the "keeps" at the surface) is also changed, the engine-man lifts the cage from the supports, and the attendant, by means of a lever, pulls them back clear of the cage until the bottom deck is lowered below them, when the attendant lets go his hold of the handle and they form a support to the top deck. During the change here, the tub in the bottom deck is changed at the pit bottom, and this being effected, the cage is lifted by the engine-man, the attendant pulls back the "keeps," the cage is lowered, and when it has descended clear of the "keeps" they are allowed to spring back ready for use again. The "keeps" at the shaft bottom are necessarily handled differently. As the loaded cage leaves the shaft bottom the attendant there pulls the handle of the "keeps" back and secures it there, by this means preventing the "keeps" from protruding in the pit. As the cage in its downward course approaches him, he takes the handle of the lever which works the "keeps" in his hand, and having allowed the bottom deck of the cage to pass below the level of the "keeps" they are allowed to spring out and support the top deck of the cage. The tub is changed here whilst the bottom deck of the other cage is changed at the surface. The "keep" handle will not require further attention from the attendant below until the cage has left for the surface when he secures the handle back in its place.

Stauss's *patent keeps*, shown in Figs. 50 to 55, have been designed to dispense with the lifting of cages before they are lowered into the pit.

Figs. 50 and 51 show the arrangement of these keeps, fixed on wooden spring cantilevers, to take the shock of the cage when lowered on to the keeps, and Figs. 52 to 55 show the details.

The wear and tear of the winding ropes and engines are reduced, because owing to there being no lifting of the cage before descending, the accompanying jerks are avoided; and jerks are the main cause of deterioration of ropes and engines. When these keeps are used, the winding rope must of course be adjusted in length, so that the cage does not fall after drawing back the keeps. The sinking of the cage should not amount to more than the slack of the rope when unloaded.

The length of the rope can easily be adjusted by means of Freudenberg's *cage-adjusting hangers*, as shown in Figs. 56 and 57, or similar appliances. This is only necessary for the first few days with new ropes; later on very seldom.

The Stauss keeps are made in various sizes by the Humboldt Engineering Co., Kalk, near Cologne.

The casting *a* carries the rocking-shaft *l*, the hand-lever *h* and lever *k*, both keyed on, the shaft *d* with the two hanging links *ee* fastened on and the connecting pin *b*. On the latter the one-armed keep *c* turns freely, being also supported on the inclined surface *z* of the bed-plate *a*; further, *b* carries the link *f*, which forms a toggle-joint with the lever *k* by means of the pin *i*.

Fig. 50.

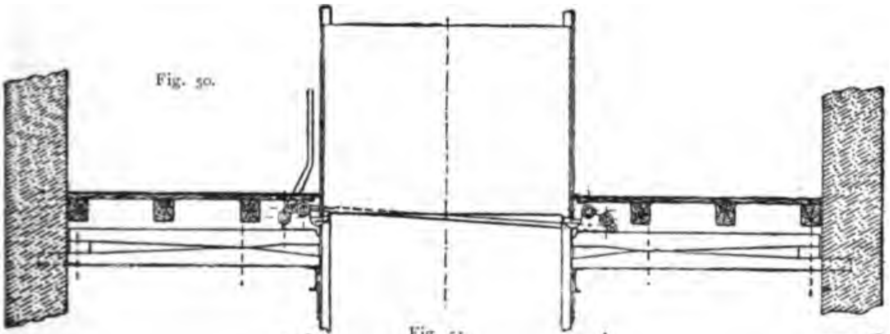


Fig. 51.

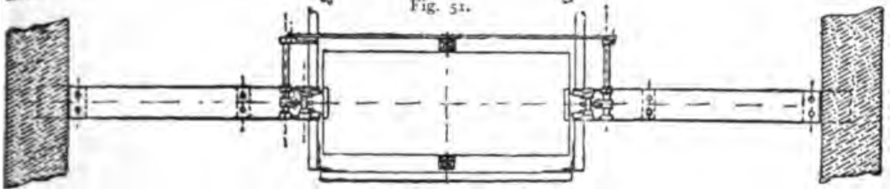


Fig. 52.

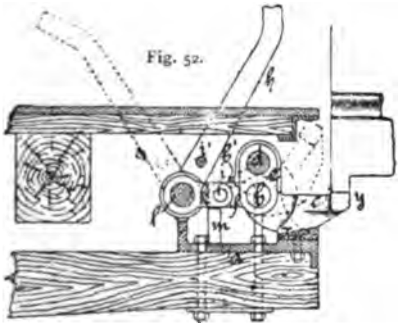


Fig. 55.

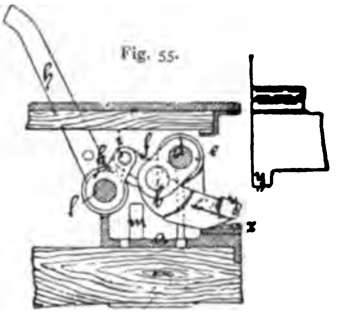


Fig. 53.

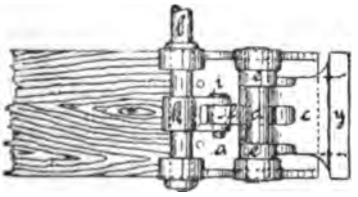
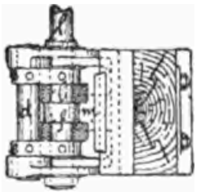


Fig. 54.



The Stauss Keeps.

The arrangement works as follows: If the cage is to be held and prevented from going down the pit, it is lowered on to the keeps *c*; these are supported by the surfaces *x* and the pin *b*, the latter being prevented from moving vertically or horizontally. It is prevented from moving vertically by the hanging links *e*, which press upwards against the shaft *d*, and horizontally by the toggle *f*, which thrusts against the pin *i* of the lever *k*, and through the latter against the rocking-shaft *l*. The whole arrangement is thus locked and any movement prevented, as the weight of the hand-lever *h* presses the lever *k* downwards upon the block *m*, which is fixed to the bed-plate *a*; and prevents any further downward movement.

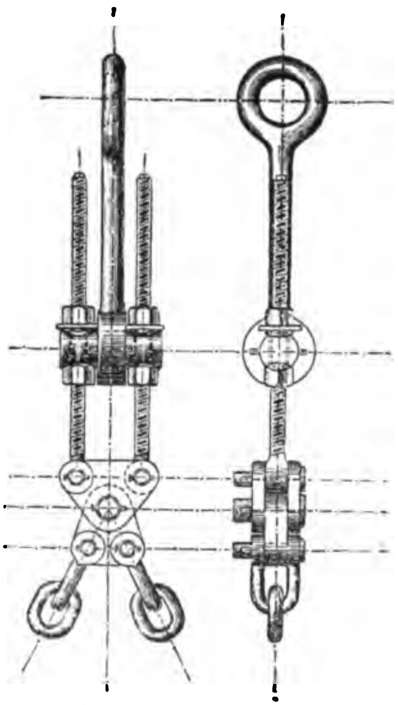


Fig. 56. Fig. 57.

FREUDENBERG'S CAGE-ADJUSTING HANGERS.

When *h* is thrown over into the dotted position shown in Fig. 52, through an arc of 60°, *i* comes into position *i'* and *b* to *b'*, thus drawing the keeps *c* away from under the cage and downwards, so that the cage is free to descend. Fig. 55 shows the relative positions of the different parts when the keeps are drawn back.

When the cage has again been lifted to the pithead, *h* is pushed over into its first position, this action carrying the keeps forward and the cage can again be lowered on to them.

Owing to the keeps *c* being free to turn round the pin *b*, the cage cannot catch fast in them, if they should be pushed out too soon. No injury can result from such action, because the cage would then only lift the keeps up into the dotted position in Fig. 52, and after it had passed above them they would fall back again into their proper position by gravity.

The friction which takes place between the surfaces *x* and *y* and in the joints when the keeps are withdrawn is easily overcome by the hand-lever, as the weight of the cage itself helps and endeavours to push the keeps backwards down the incline of 9° marked *x*. Further, as soon as the hanging links *e* have left the vertical position and *b* is turning round *d* as centre, the weight of the cage acting on *b* through *c* helps the movement of rotation and the force acting on *i*. The further the keeps recede, the greater this force becomes, until it may finally cause a sudden release of the keeps unless the angle of the incline *x* is suitable for gradual release, which is obtained when *x* is inclined at an angle of 9°.

The use of these keeps has many advantages as compared with other makes, such as a considerable saving of time and of steam, and less wear and tear on the ropes and machinery, owing to the lifting of the cage to release the keeps being unnecessary. This saving of time amounts to from 3 to 6 seconds, according to the skill of the engine-man, for each lift, with single-deck cages; and for cages with several decks the saving is more in proportion.

The CAGE is a receptacle for the tubs traversing the pit either empty or full. It is also the usual means of transport for the workmen and all others between the surface and the different loading stages in the shaft. The pit timber, workmen's tools, horse food, and water, and frequently the horses themselves, are lowered by means of the cage.

When men are riding in one cage no loaded or empty tubs are placed in the other, or in an under or over deck of that holding them. If men are not in both cages one is allowed to run empty.

The cage is usually made of wrought iron but sometimes of steel. As to its form, it is of course governed by the shape of the division of the shaft it has to run in, and it may be single or double decked, or have more decks than two if desirable. Again each deck may have one or more tubs placed in it as may be desired and arranged. Each deck floor is laid with rails to allow of the tubs being pushed in whilst the loaded ones are pulled out on opposite sides. There are various modes of keeping the tub secure in the cage during its ascent or descent. One of these is by having "false bottoms" in the cage, which is an arrangement whereby the floor of that part of the deck on which the tubs are placed falls or sinks an inch or two below the other and outer portions of the deck, but when the cage rests on the keeps all the deck floor is on one level, allowing the tubs to be changed.

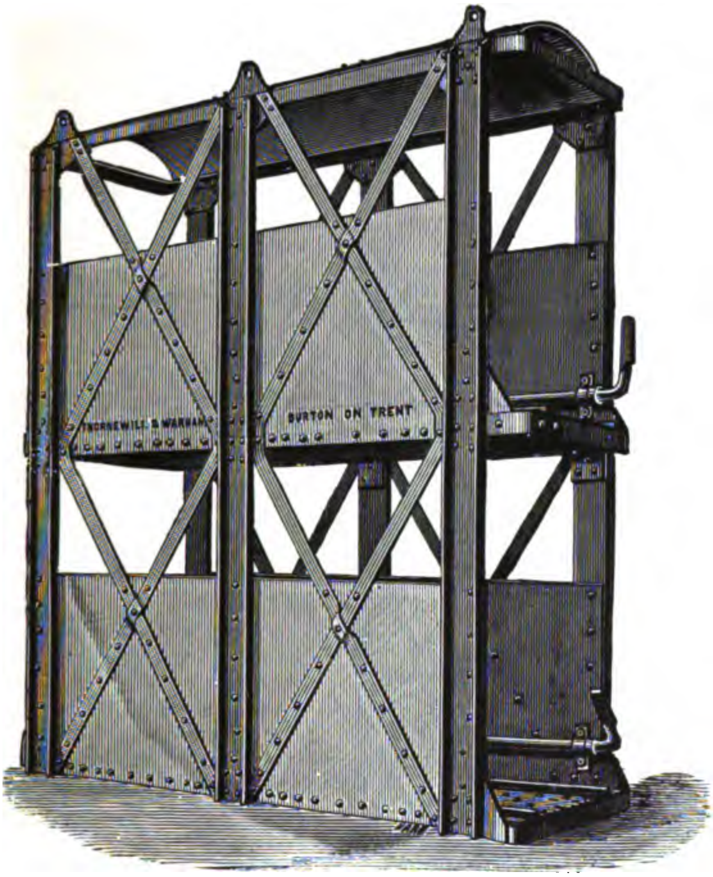


Fig. 58.—Double-decked Cage.

Another mode of securing tubs in the cage is a bar running through the cage, and at either end is placed a short lever which turns down or up on being pushed;

when down it covers the ends of the tubs and prevents their moving, when up it allows them to be changed.

The best form of catch is that which grips the axles on the tub being pushed in, without the necessity for the attendant to have to put his foot on the cage to work the catch. Sometimes a catch in the floor secures the tubs in place.

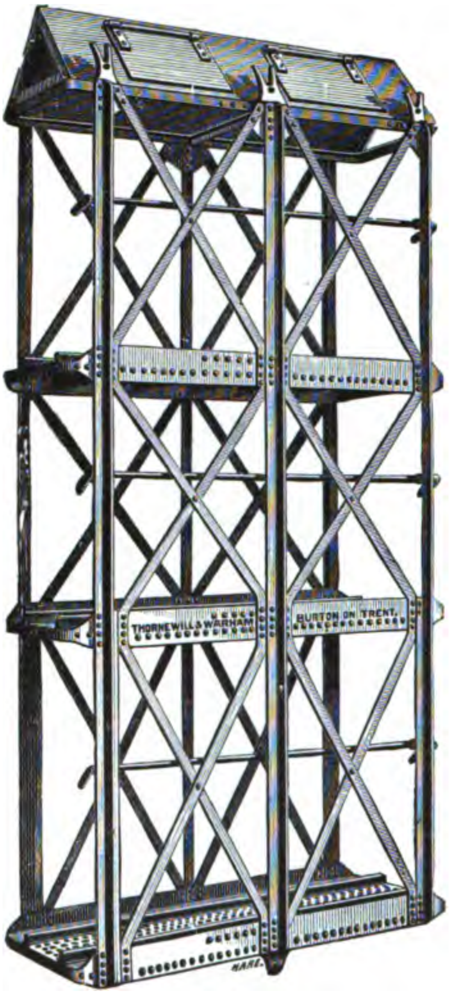


Fig. 59.—Treble-decked Cage.

The slides of the cage fit loosely to three sides of the wooden conductors, and are slightly bell-mouthed. They are applied at the upper and lower bars of the framing.

A two-decked cage would have 3 such slides on either side of it, in the usual arrangement adopted. The top of the cage is provided with an iron bonnet or cover for the protection of persons whilst descending or ascending. The cage is suspended from the rope by four short chains called "bridle" or "bull" chains, one being at each of the upper corners, and in the case of heavy cages from the middle of the longer sides as well, so that in the latter case there would be six bridle chains.

Fig. 58 shows a double-decked cage, and Fig. 59 a treble-decked cage, as made by Messrs. Thornewill & Warham, Engineers, Burton-on-Trent. The former is steel throughout, the deck frames being angle steel, the uprights of channel steel, and the bracings of flat steel. The deck frames have cross-bearers of angle steel, with "knee" ends riveted to the frames. The floor of each deck consists of perforated steel sheets.

The cage-hangers are of forged steel and sufficiently large to form a gusset to receive the uprights, cross-bracing, and top frame; they have horns forged on them to prevent the D link on bull chains from falling over when the chains are slack.

The treble-decked cage is of similar construction, the uprights being of angle instead of channel steel.

The tub-catches are plain bars with bent ends, working in suitable chocks fixed to the uprights. The roof of each cage is provided with doors, so that long pit wood and other articles may be carried on the upper deck. All rivet holes are drilled and rivets where possible put in by machine. The cages are provided with guide-cheeks for square, or loops for wire rope conductors, and each deck is fitted with rails. The finished weight of the double-decked cage is 2 tons, 3 cwts., 2 qrs., without the bull-chains. They are in daily use at the Walsall Wood

Colliery, Cannock Chase, where each cage lifts 8 tubs, holding 48 cwts. of coal per lift from a depth of 550 yards. The treble-decked cage weighs 2 tons, 14 cwts., 2 qrs., without the bull-chains, and is in use at the Hamstead Colliery, near Birmingham, where each cage lifts 6 tubs, holding 84 cwt. of coal per lift from a depth of 620 yards.

All cages should be provided with guards to be fixed to, or removed from the top of cages at will, so as to protect workmen who have to stand on the cage roofs or covers during the time they are examining or repairing the shaft, and where the cover is of a sloping or curved form, a horizontal floor should be attachable over it to enable the workmen to move about and handle their tools.

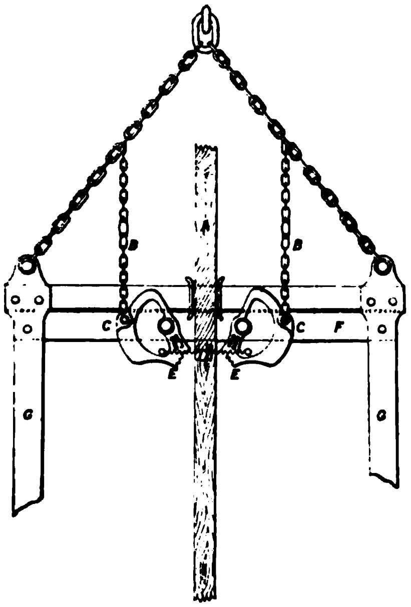


Fig. 60.

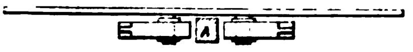


Fig. 61.—Broadbent Safety Cage.

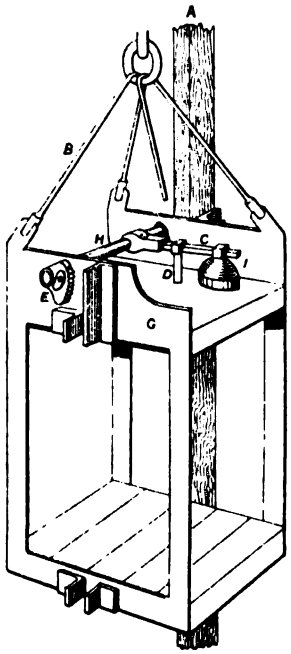


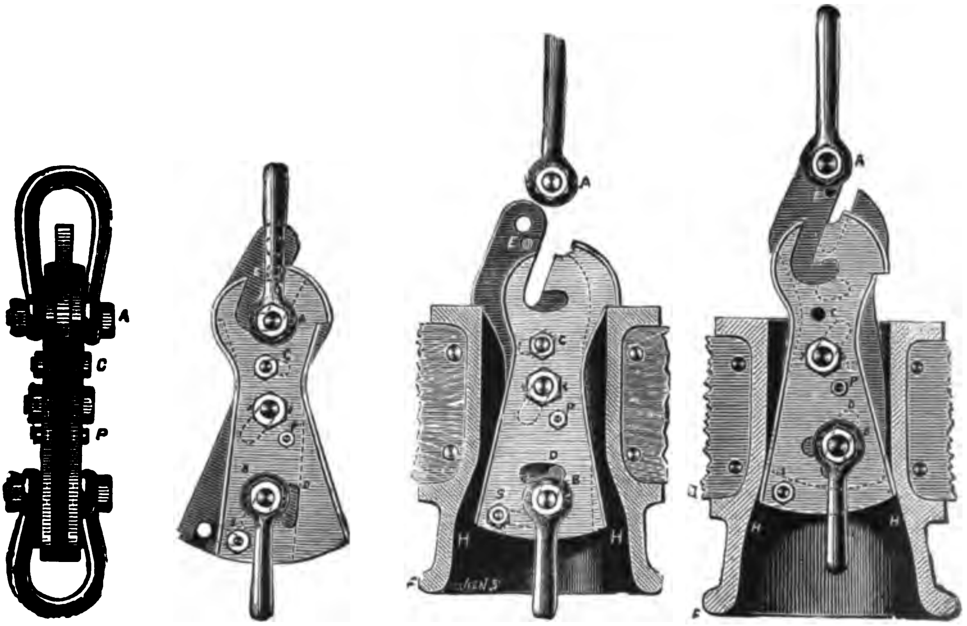
Fig. 62.—Calow's Safety Cage.

Sometimes cages have an attachment for arresting their descent in the shaft in the case of a broken rope, and are then called "safety cages." A great number of these safety appliances are before the public, but they do not seem to come into general favour. They all consist of a contrivance for gripping the guides when the strain of suspension is removed.

The uncertainty, however, of their acting when the necessity arises—it may be years after being put in—seems to preclude their adoption as a means of safety. Probably the feeling is that, as something must be relied upon, and as the complication and multiplication of appliances, the greater the liability for something to go wrong, it is preferable to rely on good ropes, properly looked after, and worked with a due regard to their margin of safety.

Broadbent Patent Safety Cage.—Fig. 60 shows a side elevation of the apparatus, and Fig. 61 a plan of the eccentrics and guides of Broadbent's patent safety cage. AA are the guides; B the suspending chains; CC levers to which the suspending chains are attached, and which, when the weight is carried, keep the eccentrics EE from coming in contact with the guides; D is a spring for giving motion to the levers CC, and eccentrics EE, if the rope breaks; F a wrought-iron plate at side of cage to carry the apparatus; GG frame-work of cage. From this description it will be readily understood that if the suspending rope breaks, the springs will force the eccentrics EE against the guides, and the cage will remain suspended.

It will also be observed that every time the cage is on the keeps, both at the top and bottom of the pit, the rope will slacken and the apparatus will come into action and press upon the guides. By altering the edges of the eccentrics they can be made to suit iron wire guides.



A. B. C. D.

Fig. 63.—Ormerod's Safety Link.

Calow's Safety Cage.—This differs from Broadbent's inasmuch as it is not dependent on its action on any direct attachment to the rope. Fig. 62 shows a perspective view of the apparatus.

A is a guide, the other being removed for the sake of exhibiting the mechanism of the cage; B the suspending rods or chains; E an eccentric carried by a shaft H, on which is also keyed a lever, C; D is a spring which is made to suspend the weight I, and keep the eccentrics E clear of the guides so long as the speed of the descending cage does not approach that of a falling body. But by the cage falling, or through any sudden jerk, the pressure of the weight I on the spring D ceases to be so intense, and therefore, the spring lifts it and sets in motion the eccentric E, which grips the falling cage. The apparatus, therefore, does not come into action each time the cage is stopped or rested on the keeps.

There are many other safety cages, but the principle of their operation is somewhat similar to those described.

Detaching Hooks.—Closely associated with safety cages are contrivances of a similar character for preventing the cage from falling when severed from the rope through overwinding. The object sought to be accomplished in these overwinding safety appliances is to cause the link by which the cage is suspended from the rope to release its hold of the rope and take hold of a portion of the framework of the headgear.

The Mines Act, 1887, does not make the use of any overwinding appliance compulsory, but a limit of speed is fixed, when men are being raised, if the mine is not provided with an automatic contrivance to prevent overwinding.

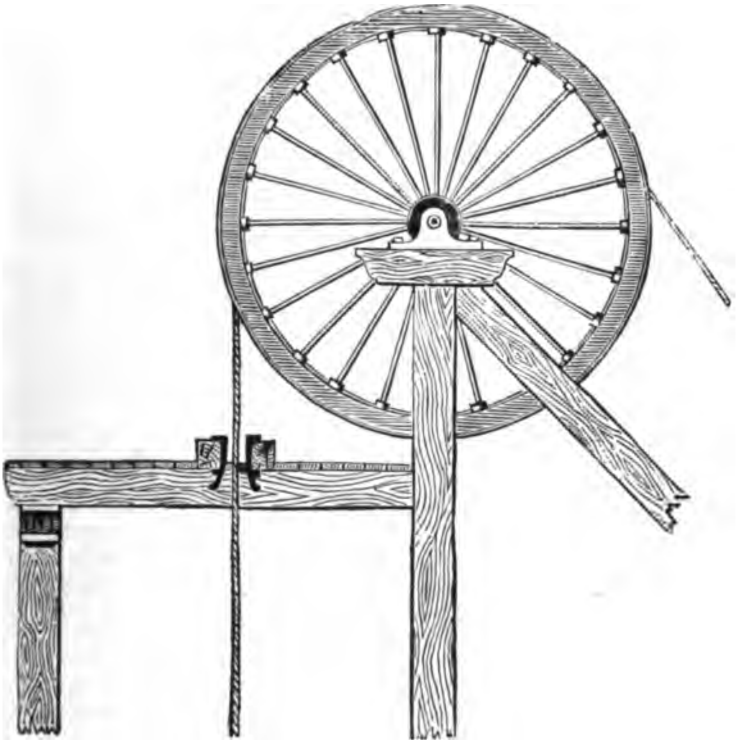


Fig. 64.—Arrangement of Catch-plate in Headgear for Ormerod's Safety Link.

Ormerod's is one of the best of these safety links, and is the invention of Mr. Edward Ormerod, of Atherton, near Manchester.

Fig. 63 (A) is a cross view of the link.

Fig. 63 (B) is a side view of the same.

Fig. 63 (C) represents the position it assumes when wound up into the cylinder, the rope shackle being disconnected, and the link firmly locked in its position.

Fig. 63 (D) shows the rope shackle re-connected for lowering the link through the cylinder.

Fig. 64 shows a section of the cylinder as fixed in the headgear or pit frame, also the platform for convenience in re-connecting the rope shackle.

It will be seen on reference to the engravings that the apparatus when in ordinary use, as in Fig. 63 (B), is wider at the bottom than the top; but in the event of overwinding, the link is drawn into the bell-mouthed cylinder F F in Fig. 63 (C), the wide part of the link at H H coming in contact with the cylinder at F F, thereby closing the bottom part of the link, also causing the top part to expand and the projections to catch over the top of the cylinder, while at the same time the rope shackle A is forced out of its seat, thus being allowed to go free; the bottom shackle B drops into the slot D and locks the link firmly in its position. The cage being suspended from the chain cannot fall back. To prevent the possibility of the link becoming disarranged in ordinary work, a small pin, P, is inserted through the plates, which pin is sheared off as the apparatus passes into the cylinder.

For lowering the cage the shackle is attached to the ear on the middle plate as shown in Fig. 63 (D). On removing the pin C, and slightly winding the rope, the middle plate (having a slotted hole in it) is elevated into the position shown, and allows the apparatus to pass down through the cylinder, and safely lower the cage.

The following advantages are claimed for it:—

1. It is self-contained. The load in ordinary work being carried from the outside plates only, thereby avoiding appreciable wear to its working parts.
2. The clasp stud E, which lips over the top part of the outside plates, considerably strengthens the hook in case of any excessive strain or jerk, and also assists the hook in taking the first shock in case of overwinding at a very high speed.
3. When detached the middle plate constitutes additional metal thrown out, and the hook is therefore actually considerably stronger than when in its working position.
4. The bell-mouthed cylinder for detaching the hook is a substantial and exceedingly strong fixing for the headgear, and is never liable to be torn away, neither does it collapse or injure the ropes through vibration.
5. The cylinder also affords a much more effective entrance for the hook than any other appliance whatever.

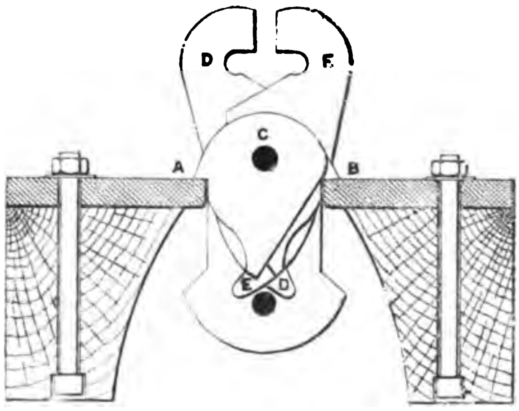


Fig. 65.—Forster and Brindle's Detaching Hook.

Forster & Brindle's is another of these detaching hooks, and is shown in Fig. 65. Two similar plates, A and B, are suspended upon a pin, C, and are made so that in passing through the detaching plate they are pressed in. On emerging from the upper end of the detaching plate they open out again so as to prevent the hook from returning. In case of a partial overwind the hook enters the detaching plate, but perhaps not sufficiently to cause shearing of the rivet which must precede detachment; the plates then clutch the point of sustainment and prevent the cage from falling back. Where the overwind is complete, the plates A and B are pressed in as they pass through the sustaining plate, and immediately after being through, are pressed out by plates D and E, which also cause detachment. The rope is expeditiously re-attached, to effect which the connection with the rope is made, and the weight tightened on it, the two plates, A and B, are pressed in by hand, and kept in whilst being lowered through the detaching plate, after which work may be resumed.

There are many other patent detaching hooks, but the difference between them and those described is not sufficient to justify a description of each.

To some extent the same remarks apply to the use of disconnecting or detaching hooks as those on safety cages. They have been known to fail at the moment of their need—probably owing to the high speed at which the cage was overwound. A lamentable example of this occurred in Yorkshire in 1886, where the cage with ten men in it was overwound, broke the bolts and fastening of the catch-plate and carried it away. The hook detached at the same time and the cage and catch-plate fell down the shaft and the ten men were killed.

It is necessary sometimes to have pipes placed in the shaft to convey steam or compressed air to an underground engine, and to place pumps or a rope communicating with an underground engine plane from the engine at the surface, but the method of securing the pumps will be described later on.

The **Headgear**, Fig. 66, consists of a pulley-frame which may be of pitch pine timber, or wrought iron, that in the Fig. being of timber. The uprights, two or four in number, forming the frame, and the two backstays or inclined pieces are the main features of a pulley-frame, and are so arranged as to best resist the strain on the pulleys, which is of a twofold character, viz., the vertical strain from the load in the shaft and a side strain towards the engine. The uprights vary in height from 30 to 70 feet, and are generally made of pitch pine from 12 to 16 inches square. The backstays are of the same sectional size, and should be placed at an angle to the uprights, and as nearly parallel as possible to the ropes from the drum to the pulleys. A good rule is to let the rope make not less than an angle of 45 degrees in going over the pulley. Diagonals and strengthening pieces are also placed across the uprights near the pulleys for strengthening their support. Pulley-frames made of iron admit of a greater length than wood. It is unwise to have the pulleys placed too near the level of the pit top, for with engines on the first motion, and having large drums, half a stroke of the engine may take the cage to the pulleys.

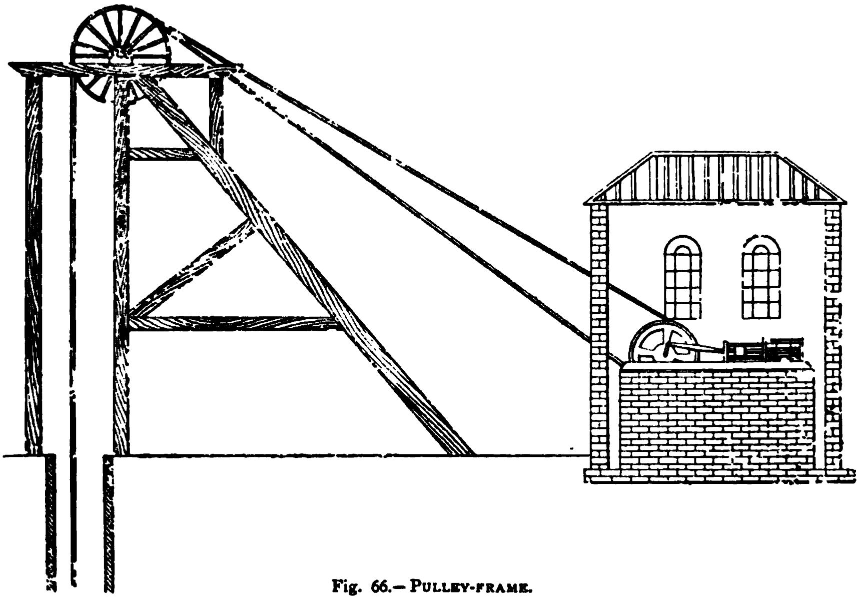


Fig. 66.—Pulley-frame.

One of the inclined pieces should have steps and a hand-rail formed on it, or a separate ladder provided, so as to admit of an attendant going up to examine the head-gear and oil the pulleys.

Each upright should rest in an iron footing placed upon a specially prepared pillar to take the vertical pressure. The backstays should each rest on ashlar or concrete foundation, and not against the engine house wall. Where space is limited and the engine house is necessarily erected rather near the shaft, there is no objection to the backstays being taken through the engine house wall so as to rest against the engine pillars. They should always be taken up to the centre of the pulleys, and not as sometimes seen to a point below this which gives the backstays less resisting power to the strain on them. The whole framework should be frequently painted for the preservation of the material composing it.

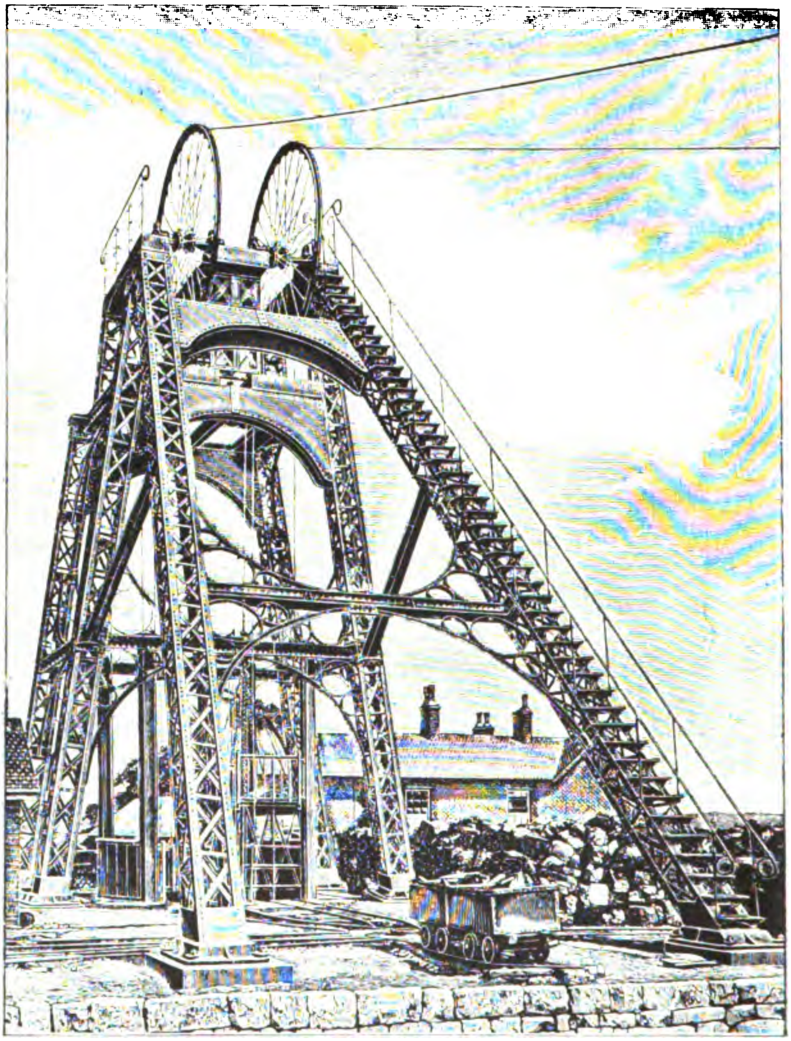


Fig. 67.—Pit-head Frame erected by Messrs. Thornewill & Warham at the New Hall Park Collieries.

The pulleys, usually placed side by side, have wrought-iron arms, the rim and central boss being generally of cast iron. They should be placed with proper regard to the lead of the ropes off the drum to the pulleys, so that the angle of the rope to each is equal. The rim is made to suit the kind of rope to be used, and is grooved accordingly. Pulleys are usually from 10 to 20 feet in diameter. In deciding on a suitable size, it must be borne in mind that ropes receive more injury from working over small pulleys than over large ones, and consequently wear out quicker. A good plan is to have the pulleys the same size as the drum of the winding engine, a rule for ascertaining which is given later in this Chapter. Provision must always be made for their adjustment and oiling.

Fig. 67 shows a Pit-head frame which is elegant in appearance and of great strength, as made by Messrs. Thornewill & Warham, Engineers, of Burton-on-Trent, and erected at the Earl of Carnarvon's New Hall Park Collieries, near Burton-on-Trent. The general arrangement of the pit bank is also shown in the Figure.

The height from pulley centres to pit bank is 40 feet, the pulleys being 15 feet in diameter. The legs are of open lattice work with 4″ × 4″ × ½″ angle iron bars connected by flat bars 2½″ × ½″, and are suitably cross-braced and stiffened by plate girders and spandrils of various sections.

The platform around the pulleys is fenced and access thereto is obtained by a stairway on one of the back-legs.

The pulleys are of the usual type, having cast-iron rims and bosses with wrought-iron arms, and are fitted with steel spindles having journals running in pedestals with adjusting screws and ample lubricating boxes attached.

The legs are provided with cast-iron plates at their feet, bedding on stone blocks mounted on brick pillars, and secured by large foundation bolts.

For lighter loads and small plants the frames are constructed of I and ⊔ iron sections and are very neat, and strong, and preferable to wood.

When required for shipment the frames are erected, and marked before being taken to pieces, and the necessary bolts provided for locking together in re-erection.

Ropes are now usually made of steel wire of different qualities, all being stronger than iron for the same size; and they may be round or flat. The round are preferable and certainly the most popular. Ropes of hemp are also used, but only to a limited extent, and are gradually falling into disuse for winding.

Wire ropes should be carefully protected with best water-proof grease. The safe working load of ropes may be taken at from $\frac{1}{5}$th to $\frac{1}{8}$th of the breaking strain. A close approximation to the safe working load of ordinarily-made wire ropes moving at high speeds is found by multiplying the weight of the rope per fathom in pounds by 5 for iron wire, and by 8 for steel wire, and consider the product as hundredweights. Thus, an iron wire rope, weighing 16 lbs. a fathom, has a safe working load of $16 \times 5 = 80$ cwt.; a steel wire rope of the same weight, $16 \times 8 = 128$ cwt. These are only rough approximations—rules that can be easily carried in the memory. André, in his *Treatise on Coal Mining*, gives the following rules to find the safe working load:—$C = \sqrt{4\ L}$ for iron wire and $C = \sqrt{2{\cdot}4\ L}$ for steel wire, where $C =$ the circumference of the rope in inches and $L =$ the safe working load, and therefore $L = \frac{C^2}{4}$ for iron wire and $\frac{C^2}{2{\cdot}4}$ for steel wire.

The following rules were at one time used by some rope manufacturers, but the difference in quality of the materials used renders the rule of little practical value. Owing to this difference probably, there is a want of uniformity in the strength of similarly-sized ropes quoted by manufacturers. Let $B =$ breaking weight in tons, and $W =$ weight per fathom in pounds. Then for hemp ropes $W = B$; for iron wire ropes $W = {\cdot}55\ B$; and for steel wire ropes $W = {\cdot}33\ B$. Or, where $C =$ circumference of rope in inches (where the ropes are flat C will answer for the perimeter) and $B =$ breaking weight in tons.

Then for hemp ropes $B = \cdot 25\ C^2$; for iron wire ropes $B = 1\cdot 5\ C^2$; and for steel wire ropes $B = 2\cdot 5\ C^2$.

The working or safe load is for—	Round	Flat	
Hemp ropes . . .	$\frac{1}{7}$th	$\frac{1}{8}$th	of breaking strain.
Iron wire	$\frac{1}{7}$th	$\frac{1}{8}$th	do.
Steel wire . . .	$\frac{1}{6}$th	$\frac{1}{7}$th	do.

Suppose, then, it becomes necessary to know the diameter of wire rope which would be equal in strength to a hemp one of 4 inches diameter, it would be found thus: the ratio that the square of the diameter of an iron wire rope bears to a hemp one in point of strength is as $1\cdot 5 : \cdot 25$. Therefore as $1\cdot 5 : \cdot 25 :: 4^2 : 2\cdot\dot{6}$, and $\sqrt{2\cdot\dot{6}} = 1\cdot 633$ inch as the diameter of the iron wire rope. Similarly, if it be required to know the diameter of steel wire rope which would be equal in strength to the hemp one of 4 inches in diameter and the iron wire rope of 1·633 inch in diameter, the reasoning would be thus: the ratio that the square of the diameter of the steel wire rope bears to the hemp one in point of strength is as $2\cdot 5 : \cdot 25 :: 4^2 : 1\cdot 6$ and $\sqrt{1\cdot 6} = 1\cdot 265$ inch as the diameter of the steel wire rope.

The duration of a rope depends upon circumstances. If placed in a wet shaft where the water is impregnated with mineral acids, or in an upcast shaft where gases injure the ropes, they will not last so long as those working in dry shafts. Ropes subjected to rapid winding are injured more than where slow speeds of winding prevail. The average duration of flat wire ropes is usually taken at one year, and that of round ropes at a year and a half; but it is impossible to fix a hard and fast line, on account of the great difference in circumstances under which ropes work. The speed of winding, freedom from chemical action, careful attention and greasing, the number of hours used per day and the number of days per year, are all elements affecting the duration. In putting on a rope care should be taken to pass it through the exterior of the drum on an easy curve, and then to coil it round the drum-shaft once or twice before securing it thereto. A rope should be long enough to admit of a couple of coils on the drum when the cage is lowered to the pit-bottom. The other end of the rope is secured to the cage by "capping."

The most frequently used method of "capping" a round wire rope is as follows:—A strip of iron is worked into suitable shape to form the cap. The central portion is well rounded for a few inches, sufficient when the two ends are afterwards bent over towards each other to form a strong loop or bow to take the **D** link which holds the cage bridle-chains. The two sides of the cap are rather smaller at the ends than next the loop, and are curved to receive the rope between them. Rivet-holes are drilled opposite each other in the two sides. Besides this socket, two rings of iron are prepared of such sizes as to afterwards fit tightly over the joint. These rings are closed before going on, and fit somewhat loosely over the rope; they are passed over the end whilst hot. About 2 feet back from its end a piece of copper wire is now bound round the circumference of the rope. The wires at the extremity are then opened and folded back upon the rope, the fold taking place at the copper-wire binding. Another length of copper wire is carefully and tightly wound over the folded wires so as to bring them into close contact with the rope and prevent any wires from projecting. As the binding proceeds upwards the folded wires are thinned gradually by cutting off a few at a time, until, when the binding has reached the top, a cone has been thereby formed at the extremity of the rope. In this state the rope is inserted between the two sides of the socket, which are easily spread open for the purpose. The sides are then, by means of clamps, pressed tightly against the rope and held firmly so, whilst rivets are driven through the socket and the heads formed on the rivets. The operation of riveting is attended with some practical difficulty. The holes are not drilled recklessly through the rope, as a drill in advancing straight across would sever many of the wires obliquely crossing its course, and so weaken the rope. To avoid cutting the wires and yet obtain a passage through them and past the cores without injury, a short needle is used, having a point at one end and a head and eye at the other. The needle is struck by a hammer or turned by passing a short rod through the eye, whilst attempts are made to direct it to form an opening between the wires and past the cores, without cutting them, in line to the corresponding rivet-hole on the opposite side. Frequently the needle, after piercing the rope, reaches the other side considerably off the desired point, and many fruitless attempts are made before the object sought is accomplished. When the rivets, usually three in number, are closed, the larger of the two rings, which was put over the rope last, is brought down over the joint and firmly pressed into its position on the socket near its bottom end. The other ring is then served similarly. It takes up a position near the top of the joint. As the rings cool they contract and tightly grip the socket.

Fig. 68 (A) shows the capping on a rope performed in this way. The object of the rings is to prevent the socket from opening or parting from the rope. They give additional security to the rivets by resisting the force within the joint (resulting from lifting and raising the cages); and after the rope has been running some time, if the rivets give way, the rings still have considerable grip on the rope, because the bottom end of the socket is larger in diameter than the top (due to the folded and cut wires), and the strain caused by lifting the cages tends to pull the rope through, and so wedge into closer contact the rope, socket and rings.

At some collieries the practice is to put the rings on in a cold condition. Where put on hot, it often happens that the time taken to bind the copper wire and drive the needles has allowed the rings to cool. They may, however, be heated again whilst on the rope before being dropped into place on the socket.

Another method of capping a round wire rope is shown in Fig. 68 (B). A piece of round iron sufficiently long to form the socket and of slightly tapered form, has a conical opening bored through it. Near its lower extremity, which has also the larger diameter, a bolt-hole is cut, the two circular openings being opposite each other. The rope is threaded through the aperture from its upper end, the opening being just large enough to allow of this. The lower extremity is slightly larger. The wires of the rope are folded back when the rope is well through the socket, and the end, as now shaped, pulled back into the aperture. The two bindings of copper wire should be wound as in the previously described method.

The folded wires prevent the rope from being pulled farther than a point sufficiently clear of the bolt-hole. Molten lead is now poured from a ladle in through the larger diameter of the aperture, the socket being inverted for the purpose. The lead fills up all the interstices between the rope and the inner sides of the socket. It is then allowed to cool, when the lead and rope form a hard solid mass within the aperture strong enough without rivets to resist the strains put upon it afterwards. A D shaped link for the reception of the cage chains is secured by a square-headed bolt passed through the bolt-hole in the socket and kept firmly in place by a cottar, driven down vertically at the tail end of the bolt. Besides the molten lead, a plug with a thread formed on it is screwed into the socket on the under side.

A third method of capping a round pit-wire rope consists in preparing a wrought-iron socket with the loop formed at its lower extremity and welding the socket down one side, leaving the central conical opening as in the other methods. Rivet-holes are then drilled in the socket, and the manner of further procedure is much the same as in the first described method. The first given method of capping is preferable, as it allows of the rope within the socket to be partially examined, whereas in the other methods the rope at the cap is quite hidden from view.

At some collieries it is customary to fix the cappings without the use of rivets. Where this practice is followed, three or four rings or hoops are used, according to the size of the rope, the weight to be lifted, and the length of the cap. After slipping the hoops up the rope in their proper order according to size, the rope may be prepared in one or two ways. Small wire is tightly bound round the rope two feet from the end, the strands are untwisted and the core of the rope removed. The wires are then bent back the opposite way to the strand, care being taken to tuck them under every other strand, and also to cut off some wires between each tuck so as to make the rope taper for receiving the cap; or it may be done after cutting away the core by bending back the whole of the wires around the rope, thinning them out to the required taper. In each case after the tucking or turning back is completed, small wire is wound round outside so as to make the rope as solid and uniformly taper as possible. The cap, having been made hot at the bow, is then laid to the rope and by means of clamps pressed down tightly to it; the hoops are then slipped down over the cap and driven home.

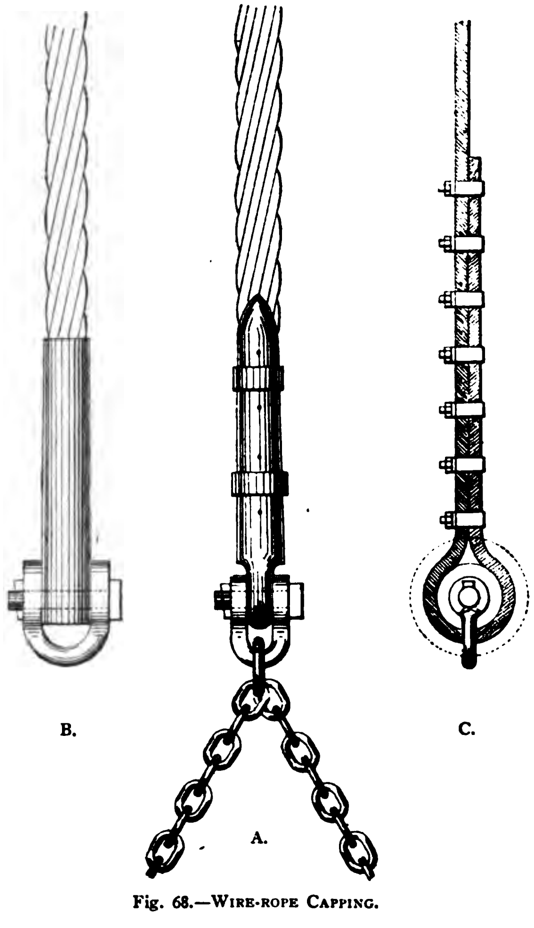


Fig. 68.—Wire-rope Capping.

Caps should be made of the best selected iron, and those for round ropes be made upon a tapered mandrel, so that they may be well shaped.

In frosty weather, the first cage journey should not be started with a full load —it is much better to give the ropes one or two runs with light loads before starting full work.

The systems of capping round pit wire-ropes are adopted also on incline and engine plane ropes, except that in their cappings the rings, or hoops, which would be inconvenient, are omitted.

With flat wire ropes, the method of capping differs from any just described. Fig. 68 (c) illustrates the plan adopted. A portion of the end of the rope is doubled back over a wooden roller (protected by a thin covering of iron) through the centre of which is a bolt-hole. Above the roller, the surface of the doubled back portion of the rope is pressed firmly against the surface of one side of the rope. It is secured in this position by a series of glands placed about a foot apart. The gland consists of a three-sided strip of flat iron except at the extremities where it is rounded, and a thread cut to take a nut. Another flat plate, which is a straight plate rather longer than the width of the rope, is passed over the ends of the three-sided plate whilst the latter is in position, accurately fitting round the joined portion of rope. A nut is then run on the thread prepared at either side of the gland, and the doubled rope is tightly gripped within the two plates, as it is enclosed by their four sides.

Two discs or cheeks of thin iron, having bolt-holes cut to correspond with that in the roller, are bolted one at either side of the roller, each bolt going through the two discs and the roller. The diameter of the discs is sufficient to form flanges to the roller and keep the rope securely on it. A **D** shaped link is then fastened by means of a bolt to the underside of the roller and this link receives the cage chains.

In some instances, a short iron cylindrical tube is used without the interior packing of wood, round which the rope is passed, and the **D** link secured below by a bolt passed through the interior of the tube. No flanges are used, the **D** link at the bottom of the rope preventing its slipping from the iron roller, which is slightly longer than the width of the rope. This is a very simple method of capping, but not quite so safe as some others.

Another method of capping flat wire ropes is by using a wooden block of a pear-shaped instead of a circular section. The thin end is placed upwards, and it is bored with a central hole. Where the rope passes round this wooden block long or short clamps are used. The clamps are shaped to suit the blocks, and are provided with bolt-holes to receive bolts and nuts above and below the wooden block for securing the rope. With a long pair of clamps, there may be one bolt below and three or four pairs above. With a short pair of clamps, one bolt is placed below and a pair above the block. Above the short clamps two or three separate pairs of short plates, rather longer than the width of the rope, are placed, each pair being provided with bolt-holes through which, by means of two bolts and nuts, they are brought to bear upon the rope between them. Before being tightened, oval-shaped pieces of wood are inserted between the two surfaces of the doubled rope at points intermediate to the pairs of plates. These bulge the rope out and form it into wedge-shaped portions which tend to keep the rope tight against the plates and so prevent it from slipping.

At some collieries, a practice, which is to be recommended, is to re-cap the ropes periodically, say every 3 months or so, whether there is any apparent necessity or not.

When fixing a new rope it should be carefully uncoiled off a reel used for the purpose.

The following observations for users of wire-ropes, which have been issued by

Messrs. Wilkins & Co., the well-known wire rope manufacturers, are much to the point, and deserve the attention of all interested:—

"*Qualities of Steel Wire.*—There are four qualities of steel wire used for wire-rope making, viz.:

	Breaking Strain.		
Extra plough steel	110 to 120	Tons	Per square inch Sectional area.
Mild plough steel	95 to 100	"	do.
Best patent steel	80 to 85	"	do.
Bessemer steel	40 to 45	"	do.

"*Plough Quality Steel Wire.*—Ropes made of good patent steel wire of a tensile resistance equivalent to 80 to 85 tons per square inch, usually prove more satisfactory than those made of plough quality steel wire, which is always very speculative, especially when run at high speed and round pulleys except those of very ample diameter.

"*Specifications.*—Users of wire ropes would often buy more advantageously by first submitting specification of working to the wire ropemaker for advice as to the kind of rope best adapted to any particular work. Specifications should state: 1. Length of rope. 2. Size of gear. 3. Speed. 4. Load, exclusive of rope. 5. If for wet workings. 6. Rate of incline. 7. Particulars of curves.

"*Working Loads.*—Many ropes are seriously damaged by being overloaded. The maximum working load at average speed, including weight of rope, should not exceed an eighth of the breaking strain, or at high speed a tenth of the breaking strain.

"*Gear.*—Great care should be taken that wire ropes are not worked round drums or over pulleys (at brows or angles of whatever degree) of insufficient circumference, and that they do not strike against any hard substance while in motion.

"*Storing.*—Wire ropes must be very carefully stored. They should on no account be placed on the ground, but on sound planks raised several inches from the earth, so that they may be free from damp. They should also be covered with a tarpaulin and regularly inspected from time to time.

"*Uncoiling.*—Much care should be taken in uncoiling wire ropes, to prevent kinking. The coil should not be laid stationary, but should be placed on a turn-table or reel. Unwind from the outer end.

"*Grease.*—To prevent corrosion, all working ropes should receive a regular dressing thoroughly laid on by passing the rope between roller brushes well fed with wire rope grease.

"*Starting.*—The greatest strain on a rope being at the moment of starting, every care should be taken to insure perfect steadiness of movement, as jerking is ruinous to ropes.

"*Changing Ropes.*—A rope may be changed from a smaller to a larger drum, but not from a larger to a smaller one.

"*Lightning Conductors.*—Undoubted evidence exists of the explosion of fire-damp in collieries through sparks from atmospheric electricity being led into the mine by the wire ropes of the shaft and the iron rails of the galleries. Hence the headgear of all shafts should be protected by proper lightning conductors, as suggested in the Report of the Lightning Rod Conference, 1881.

"*Safety.*—Nothing costs too much on which life depends."

The following tables of breaking strains and working loads of ropes, and also the weights and strengths of chains, are issued by Mr. Frederick W. Scott, another well-known rope manufacturer:—

Comparative Table of the Different Qualities of Steel Wire Round Ropes, showing Breaking Strains and Working Loads.

Approx. Weights per Fathom.		Bessemer Steel or Substitute for Charcoal Iron.	Crucible Cast Steel.	Improved Patent Steel.	Improved Plough Steel.	Breaking Strain. Tons.	Working Load. Cwts.
Size. Cirfe.	Lbs. p. Fm.						
¾	¾	1⅛	1	⅞	¾	2	4¼
⅞	1	1⅜	1¼	1 1/16	⅞	3	6¾
1	1¼	1⅝	1½	1¼	1	4	8¾
1⅛	1½	1¾	1⅝	1⅜	1⅛	5	11
1¼	1¾	2	1¾	1½	1¼	6	13¼
1⅜	2	2¼	2	1⅝	1⅜	7½	16½
1½	2½	2⅜	2⅛	1¾	1½	9	20
1⅝	3	2½	2¼	2	1⅝	10½	23¼
1¾	3¼	2¾	2½	2⅛	1¾	13	29
1⅞	3¾	3	2⅝	2¼	1⅞	14½	32¼
2	4¼	3⅛	2¾	2⅜	2	17	37½
2⅛	4¾	3⅜	3	2½	2⅛	18	40
2¼	5¼	3½	3⅛	2⅝	2¼	21	46½
2⅜	6	3¾	3¼	2¾	2⅜	22	49
2½	6½	4	3½	3	2½	25½	56¾
2⅝	7	4¼	3¾	3¼	2⅝	28	62¾
2¾	8	4⅜	3⅞	3⅜	2¾	31	69
2⅞	8½	4¾	4	3⅝	3	36	80
3	9	4⅞	4⅛	3¾	3⅛	39	86
3⅛	9½	5	4⅜	3⅞	3¼	42	93
3¼	10¼	5½	4⅝	4	3½	49	109
3⅜	10¾	5¾	5	4½	3¾	56	125
3½	12	6	5½	4¾	4	67	149
3⅝	12¾	6½	6	5	4¼	76	169
3¾	13½	7	6¼	5½	4½	88	196
4	15¾	7¼	6⅝	5¾	4¾	94	209
4⅛	16¼						
4¼	17½						
4⅜	18¾						
4½	20						
4⅝	20¾						
4¾	21½						
4⅞	22						
5	22¾						
5¼	24						
5½	26						
5¾	30						
6	34						
6¼	37						
6½	40						
6⅝	43						
7	48						
7¼	52						

The weights of ropes given are for wire cores in strands and main core of hemp.

From 3½″ to 7¼″ the weights are for ropes made compound.

The weights of compound ropes vary according to construction.

For hemp cores in the strands deduct ⅛th for sizes down to 3⅜″.

For wire main core add ⅕th to the weights given above.

Comparative Table of the different qualities of Steel Wire Flat Ropes, showing Breaking Strain and Working Loads.

Approximate Weight per Fathom.		Bessemer Steel, or Substitute for Charcoal Iron.	Crucible Cast Steel.	Improved Patent Steel.	Improved Plough Steel.	Breaking Strain.	Working Load.
Size.	Lbs. p. Fm.					Tons.	Cwts.
$2\frac{1}{2} \times \frac{1}{2}$	13	$2\frac{1}{2} \times \frac{1}{2}$	...	...	...	$19\frac{1}{2}$	43
$2\frac{5}{8} \times \frac{1}{2}$	14	$2\frac{3}{4} \times \frac{1}{2}$	...	...	...	23	51
$2\frac{3}{4} \times \frac{1}{2}$	15	$3 \times \frac{5}{8}$	...	...	...	25	55
$3 \times \frac{5}{8}$	19	$3\frac{1}{4} \times \frac{5}{8}$	$2\frac{1}{2} \times \frac{1}{2}$	...	...	28	62
$3\frac{1}{8} \times \frac{5}{8}$	21	$3\frac{1}{2} \times \frac{11}{16}$	$2\frac{3}{4} \times \frac{1}{2}$	$2\frac{1}{4} \times \frac{3}{8}$	...	32	71
$3\frac{1}{4} \times \frac{5}{8}$	22	$3\frac{3}{4} \times \frac{3}{4}$	$3\frac{1}{4} \times \frac{5}{8}$	$2\frac{3}{4} \times \frac{1}{2}$	$2\frac{1}{4} \times \frac{3}{8}$	36	80
$3\frac{1}{2} \times \frac{11}{16}$	23	$4 \times \frac{3}{4}$	$3\frac{1}{2} \times \frac{11}{16}$	$3 \times \frac{5}{8}$	$2\frac{1}{2} \times \frac{1}{2}$	44	$97\frac{3}{4}$
$3\frac{3}{4} \times \frac{3}{4}$	27	$4\frac{1}{4} \times \frac{13}{16}$	$3\frac{3}{4} \times \frac{3}{4}$	$3\frac{1}{8} \times \frac{5}{8}$	$2\frac{5}{8} \times \frac{1}{2}$	49	109
$4 \times \frac{3}{4}$	30	$4\frac{1}{2} \times \frac{7}{8}$	$4 \times \frac{3}{4}$	$3\frac{1}{4} \times \frac{5}{8}$	$2\frac{3}{4} \times \frac{1}{2}$	55	122
$4\frac{1}{4} \times \frac{13}{16}$	32	$4\frac{3}{4} \times \frac{7}{8}$	$4\frac{3}{8} \times \frac{13}{16}$	$3\frac{1}{2} \times \frac{11}{16}$	$3 \times \frac{3}{8}$	68	151
$4\frac{3}{8} \times \frac{13}{16}$	33	$5\frac{3}{4} \times 1$	$4\frac{7}{8} \times \frac{7}{8}$	$3\frac{3}{4} \times \frac{3}{4}$	$3\frac{1}{4} \times \frac{5}{8}$	79	$175\frac{1}{2}$
$4\frac{1}{2} \times \frac{7}{8}$	31	$6 \times \frac{3}{4}$	$5 \times \frac{15}{16}$	$4 \times \frac{3}{4}$	$3\frac{1}{2} \times \frac{11}{16}$	91	202
$4\frac{3}{4} \times \frac{7}{8}$	32	...	$5\frac{3}{4} \times 1$	$4\frac{1}{2} \times \frac{7}{8}$	$3\frac{3}{4} \times \frac{3}{4}$	105	233
$4\frac{7}{8} \times \frac{7}{8}$	34	...	...	$4\frac{3}{4} \times \frac{7}{8}$	$4 \times \frac{3}{4}$	110	244
$5 \times \frac{15}{16}$	37	...	...	$5\frac{1}{4} \times \frac{15}{16}$	$4\frac{1}{4} \times \frac{13}{16}$	125	277
$5\frac{1}{4} \times \frac{15}{16}$	38	...	...	$5\frac{3}{4} \times 1$	$4\frac{1}{2} \times \frac{7}{8}$	148	329
$5\frac{3}{4} \times 1$	41	The weights of ropes given are for wire in centre of each strand, if made hemp centres, deduct about $\frac{1}{5}$th.					
$6 \times \frac{3}{4}$	44						

Conductors or Guide Rods.

Circumference	$2\frac{1}{4}$	$2\frac{1}{2}$	$2\frac{3}{4}$	3	$3\frac{1}{4}$	$3\frac{1}{2}$	$3\frac{3}{4}$	4	$4\frac{1}{4}$	inches.
Weight per fathom ...	$8\frac{1}{4}$	9	$10\frac{1}{2}$	$11\frac{1}{2}$	$13\frac{1}{2}$	$15\frac{7}{8}$	$18\frac{1}{2}$	$31\frac{1}{2}$	25	lbs.

Table of Weights and Strengths of Chains.

Diameter.	Weight per Fathom.	Proof Strength.	Diameter.	Weight per Fathom.	Proof Strength.
Inches.	Lbs.	Tons.	Inches.	Lbs.	Tons.
$0\frac{5}{16}$	$5\frac{1}{2}$	1·27	$1\frac{3}{16}$	76	29
$0\frac{3}{8}$	8	1·83	$1\frac{1}{4}$	84	32
$0\frac{7}{16}$	$10\frac{1}{2}$	2·5	$1\frac{5}{16}$	93	35
$0\frac{1}{2}$	$13\frac{3}{4}$	4	$1\frac{3}{8}$	102	38
$0\frac{9}{16}$	17	5	$1\frac{7}{16}$	111	41
$0\frac{5}{8}$	22	6	$1\frac{1}{2}$	120	44
$0\frac{11}{16}$	26	7·25	$1\frac{9}{16}$	128	48
$0\frac{3}{4}$	30	10	$1\frac{5}{8}$	136	52
$0\frac{13}{16}$	36	11·5	$1\frac{11}{16}$	142	56
$0\frac{7}{8}$	42	13	$1\frac{3}{4}$	148	60
$0\frac{15}{16}$	49	15	$1\frac{13}{16}$	150	65
1	55	18	$1\frac{7}{8}$	162	70
$1\frac{1}{16}$	60	22	$1\frac{15}{16}$	171	75
$1\frac{1}{8}$	68	26	2	180	80

Self-acting incline ropes, and hauling ropes frequently require splicing as a result of breakage. Pit ropes, are, for obvious reasons not spliced.

A common method of repairing a broken rope is by a shackle joint. The method of capping a rope, shown in Fig. 68 (A), is to a large extent followed, but the rings over the joint would be greatly in the way and are therefore not used. A socket with bow having been riveted to one end of the broken rope, the other end is similarly treated and the two ends are joined by means of a link passed through the two loops and carefully closed, see Fig. 69. A rope joined in this way may last a long time, but there is an increased amount of friction caused by the joint passing over the rollers.

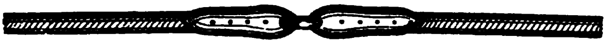


Fig. 69.—Repairing Broken Wire-rope.

The following method of splicing ropes is advised by Mr. Frederick W. Scott, of Reddish, near Stockport:—

"In splicing a wire rope the greatest care should be taken to leave no projecting ends or thick parts in the rope. Heave the two ends taut, with block and fall, until they overlap each other about twenty feet. Then open the strands of both ends of the rope for a distance of ten feet each; cut off closely the main heart or cores (see Fig. 70), and then bring the open bunches of strands face to face, so that the opposite strands interlock regularly with each other.

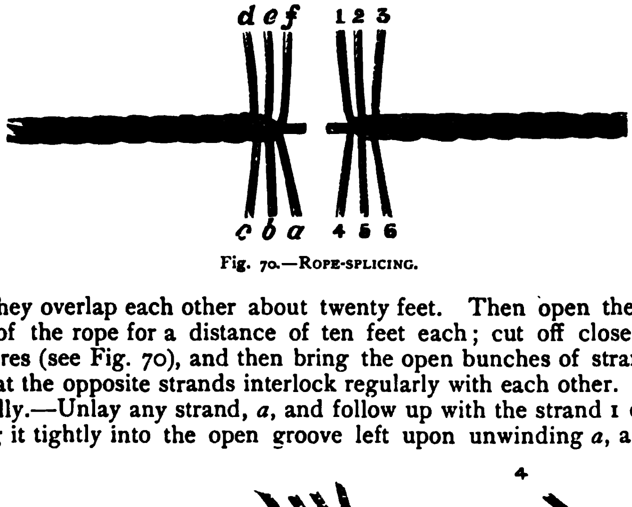


Fig. 70.—Rope-splicing.

"Secondly.—Unlay any strand, *a*, and follow up with the strand 1 of the other end, laying it tightly into the open groove left upon unwinding *a*, and making the twist of the strand agree exactly with the lay of the open groove, until all but about six inches of 1 are laid in, and *a* has become twenty feet long. Next cut off *a* within six inches of the rope (see Fig. 71), leaving two short ends, which should be tied temporarily.

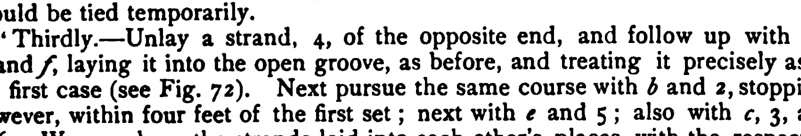


Fig. 71. Rope-splicing. Fig. 72.

"Thirdly.—Unlay a strand, 4, of the opposite end, and follow up with the strand *f*, laying it into the open groove, as before, and treating it precisely as in the first case (see Fig. 72). Next pursue the same course with *b* and 2, stopping, however, within four feet of the first set; next with *e* and 5; also with *c*, 3, and *d*, 6. We now have the strands laid into each other's places, with the respective ends passing each other at points four feet apart, as shown in Fig. 73.

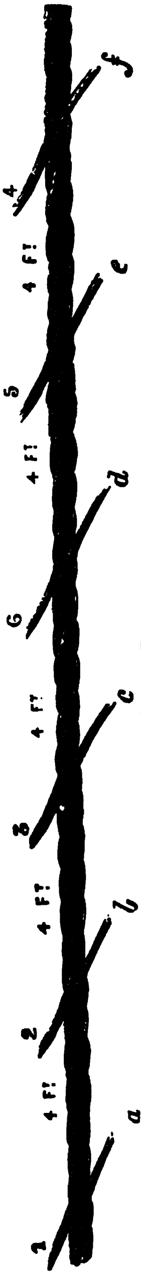


Fig. 73.—Rope-splicing.

"Fourthly.—These ends must now be secured and disposed of, without increasing the diameter of the rope, in the following manner:—Insert marlinespike through the centre of rope and cut out six inches of main core, and place the end of 1 under *a* into the place occupied by the core, and then cut out the core in the same way on the right and place the end of *a* into the place of the core in like manner.

"The ends of the strands should be straightened and lapped with fine hemp seizing before being put in. Then dispose of the remaining ends alternately in the same manner. After having done this, the rope should be well closed, and any unevenness or irregularity can be taken out by pounding it with a wooden mallet.

"In cases where ropes are heavily worked, when tucking in the ends pass No. 1 over *a* and *b* over No. 1. This mode of splicing ensures a very tight grip, and has been found to answer admirably."

This method of splicing is most effective, but as it occupies a considerable time to perform it is not usual to splice the rope during the working hours of the day. To do so then would interfere with the pit traffic, more especially in the case of a hauling rope, and seriously reduce the day's landings and workmen's earnings. At well-conducted collieries, spare shackles are kept ready for use at a moment's notice. If a rope break, the rope repairer, provided with the shackles, joins the broken ends in accordance with Fig. 69, and the pit is soon in full operation again. But this is regarded only as a temporary expedient for getting the day's coal out with the least possible reduction in output. When operations for the day cease, and at a time when the temporarily joined rope is not in use, the rope splicer descends the shaft, takes off the shackles and carefully performs the splice shown in Figs. 70—73.

Shaft Signals.—Separate and distinct signals are required between the pit top and every stopping or landing place in the pit and *vice versâ*. The system may be either the ordinary one of wires working bells or hammers, or by electric signals, speaking tubes, or telephones. Whatever system is employed a code of signals is necessary.

In deciding on a system of signalling the importance of some means of communication between the men in the cage while being drawn up or let down the shaft, should be kept in view. Electricity offers the best means of attaining this object in dry shafts, as each cage may work through a wire, or a couple of wires placed close together, within easy reach of those in the cage. Besides the facility thus offered for the occupants of the cage to signal to the surface in case of accident, electric signals would be convenient to those engaged in the examination or repairing of the shaft.

An electrical shaft signal has been invented and patented by Mr. W. Armstrong, jun. It is an electrical appliance by which signals can be interchanged between those in a cage either in motion or at rest and the engineman. An insulated core is carried in one of the strands of the winding rope, the lower end being attached to a ringing key in the cage, and the upper to a commutator on the engine shaft, through which the current passes to an ordinary stroke bell. Telephones are used and conversation carried on between the occupants of the cage and the engineman at any portion of the winding.

THE STAGE AND ITS FITTINGS.—The stage should be arranged to suit the particular circumstances of the colliery, but a good plan is to raise it on cast-iron columns 20 to 24 feet above the railway level to allow for proper screening arrangements and for the largest trucks to be loaded under the screens.

Besides screens the staging should carry the weighing machine, tables and cabins. There may also be a small smithy for sharpening picks, &c., and a workshop for repairing tubs if space is available; but none of these erections should in any way interfere with the view of the winding engineman, which must be clear and uninterrupted, so that he can watch the cages as they come up the pit.

The "tippler," "tumbler," or "kick-up" is a contrivance for facilitating the discharge of the coal out of the tub on to the screen; it is placed close to the top of the screen and the tub on being pushed into it, in some cases turns right over and in others sufficiently so to allow the coals to pass gently into the screen. The attendant afterwards easily puts it into its original position and returns to the shaft with the empty tub.

A recently improved patent kick-up works automatically, and may be associated with a self-indicating weighing machine. The tub on entering the tumbler causes the weighing machine to register its weight and directly afterwards turns over, shoots out the coals, and returns to its usual position. The weight of empty tub is then registered and deducted from the previously taken gross weight. The self-righting tippler is so made as to allow the full tub to follow in and push the empty forward as it enters. A vessel is attached to the bottom of the kick-up in which is placed a liquid, the weight of which is sufficient to cause the tub to right itself. It thus gives the kick-up its automatic action. The tub turns over side-ways, not end-ways as is usual.

In placing the upright columns which carry the stage, care should be taken that they rest on a solid foundation of stone and that they are well arranged, and with at least 2½ feet clear space between them and the side of the waggons while being loaded under the screens, so as to avoid accidents. The floor about the railway under the screens may be paved to enable the coal to be swept up unless the pipes, &c., laid, render it inadvisable. If more than one screen is erected to load on the same line there should be sufficient space between them to allow for the largest railway wagons. The length and pitch of the screen will depend on circumstances—the size, quality and freedom from impurities in the coal raised. A very usual pitch is 1 in 2. The sides are usually of wood or iron, the bottoms of cast-iron plates. There should be a slight fall from the shaft to the screens. The screen bars may be of wrought-iron or steel and are usually placed to have from ¼ to 1½ inch space between them, according to the circumstances of the colliery. The railways about the screens should be laid at such gradients that empty wagons will, by gravity, quietly move under, and loaded wagons move away from the screens on being started. The greatest difficulty in the matter is, that if the gradients are suitable for good weather, they are certain to be too flat for winter during frost and snow; if they are made to suit winter weather, they are too steep in good weather or during rain, and entail much labour in either pushing, or braking and spragging in these extreme seasons. Some wagons run much better than others, so that an inclination suitable to one may not answer so well for another. It is usual to adopt an inclination of from ⅜ths of an inch to ½ an inch per yard.

Where greater facilities are required for cleaning and preparing different kinds of coal and of various sizes, such as nuts, beans, peas, small, duff, &c., more elaborate arrangements are provided, which are worked by machinery, such as revolving riddles, moving bands, vibrating screens, and cleaning tables. There should be 6 feet of space between each line of railway, and sufficient siding accommodation for empties and also for loaded wagons for one day's work.

Winding-Engine.—One of the most important of the surface arrangements is the winding-engine, which is vertical, horizontal, single, or double; it may be a beam engine or geared. The horizontal, direct-acting coupled engines are unquestionably the best. A single engine causes delay and is an annoyance when it gets on "centre" and every part of the horizontal engine is more open to inspection by the engineman, than the vertical; is easier cleaned, oiled, and repaired. With 60-foot high pulley frames there should be not less than 20 yards between the centre of the drum and the centre of pit, but with higher pulley-frames the distance must be proportionately greater. The under rope of the drum is subject to more strain than the other because it is bent one way in coiling on the drum and another in passing round the pulley.

A few winding-engines are condensing, but this arrangement is not easy of application, owing to the rapidity of winding and the frequent stoppages and startings. Occasionally winding-engines work expansively but the intermittent working of the engines prevents a more general adoption of this plan, though condensing and expansion both help to economise fuel. Compound engines have in a few instances been applied to wind coal. The Great Western Colliery Co. have erected compound engines to wind coal at one of their shafts near Pontypridd, the steam for which is supplied by Lancashire boilers working at 120 lbs. pressure—a higher steam pressure than formerly prevails, 60 lbs. being a very common steam pressure; and modern Lancashire boilers are made to work at or above 100 lbs. The length of the cylinder is about double its diameter, and is then considered to be well proportioned. The best position for the winding-engine is on a level with the stage top where the tubs are pulled out of the cages, or but slightly above that level as it thus affords the engineman on duty a clear and uninterrupted view of the pit top. It should never be placed below the stage level. The cylinders of the winding-engine should in all cases be behind the drum, so that when the engineman is at the handles, all parts of the machinery are before him as he looks towards the pit top. The drum for a round rope may be either plain, conical or spiral. André gives the following rule for plain cylindrical winding drums. Assuming 10 feet to be the minimum diameter for a wire rope 1 inch in circumference, add 6 inches to the diameter of the drum for every increase of $\frac{1}{4}$ of an inch in the circumference of the rope. Thus a $4\frac{1}{4}$ inch circumference rope will require a drum 10 feet + 6 feet 6 inches = 16 feet 6 inches in diameter. At very great depths there is a disadvantage in having the drum excessively large, for owing to the inertia at the lift the power required is much in excess of that during the latter part of the ascent of the load. This fact has led to the introduction of conical and spiral drums, so as to equalise, at least to some extent, the strain on the engine.

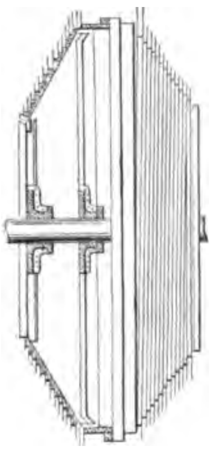


Fig. 75.—Scroll or Spiral Drum.

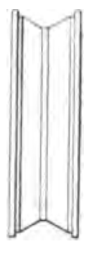


Fig. 74.—Cone Drum.

In conical and spiral or scroll drums shown in Figs. 74 and 75 respectively, the diameter of the drum is least when the engine lifts the load and increases as the cage advances up the shaft. There is an objection to the cone drum, because the rope is liable to slip on it. A spiral drum should have considerable difference in its diameters at the beginning and end of the winding, and to avoid liability of slipping, the last few coils of the rope in the winding should lap on a plain part of the drum as shown on the sketch given. The spiral drum is the most perfect form of counterbalancing known, but is enormously heavy and costly, and unless for very deep shafts, these disadvantages will outweigh the advantage gained by its counterbalancing effect.

In winding-engines having cylinders of 25 inches in diameter and upwards, Cornish double beat valves, being more easily worked, are preferable to the slide valve, with which, unless it is balanced by a special arrangement, there is much friction, consequently the engine becomes unwieldy, and there is a difficulty in reversing it with the steam on. The Cornish valve may be opened and closed and put into any position with ease without special appliance.

Connected with the crank shaft either directly or by means of gearing is the drum for winding, the eccentrics for working the valves in the steam chest placed alongside the cylinder and, usually, the indicator which shows the engineman the position of the cages in the shaft is worked off the crank shaft itself. The throttle valve is usually placed immediately under the engineman and by means of a handle communicating with the valve, he is able to turn the steam into the engine or to stop its entry at pleasure. Another handle near him communicates with the reversing gear, and a foot brake is generally attached to the drum, but sometimes a steam brake is applied in preference. A stuffing box with gland in the end of the cylinder, prevents the escape of steam there, as the piston rod works through the gland.

A simple form of automatic brake has been devised to prevent overwinding. At a point, which the cage, in its ascent, ought not to reach when under proper control, are placed levers overhanging the shaft. The cage, on striking these levers, puts into action a steam brake which acts directly on the drum.

Fig. 76 represents a pair of high-pressure winding-engines as made by Messrs. Thornewill & Warham, Engineers, of Burton-on-Trent. The cylinders are 26 inches in diameter with a 5-foot stroke, the winding-drum being of cast iron.

The cylinders are bolted and keyed to planed facings on the bed-plates, the covers being fitted with glands and stuffing-boxes, brass-bushed and provided with square-threaded gland-bolts and nuts. The covers are well stiffened and have an ample number of joint-bolts and studs.

Branches are cast on the cylinder to receive the nozzle boxes, each having fitted one steam- and one exhaust-valve, both being Cornish double-beat valves made of gun-metal, with stems of steel or phosphor-bronze, and are fitted with bridles to receive the lifting-cams on the rock-shafts which are operated by the link motion. The reversing motion shown is of the shifting-link type, but the Gooch and Allan link motions are fitted when preferred. The rods and shafts are of best iron, and the pins, links, and dies are of steel; the eccentrics and straps are of best cylinder metal.

The pistons are of cast-iron of box section, and packed with Oldham's or other makers' special rings if required; the piston-rods are of mild steel, the cross-heads of hammered iron with steel gudgeons; the connecting-rods of hammered iron fitted with brasses, straps, gibs and cottars; the cranks of hammered iron with steel pins, and are shrunk when hot and afterwards keyed on the crank-shaft which is preferably of best hammered scrap-iron.

The guide-bars and blocks are of cast-iron, planed and having means for adjustment when worn.

The bed-plate is of hollow box section, well ribbed, and having facings to receive the various mountings.

The main bearings are fitted with heavy gun-metal steps secured by iron caps and bolts.

The winding-indicator consists of a cast-iron open-fronted column containing a long screwed spindle on which is a nut with a finger. The lower end of the column is bolted to the engine bed-plate, and is fitted with a pair of bevel wheels, one of which is keyed on the small drag-shaft driven off the main crank-pin, and the other to the upright screwed spindle. The nut is prevented from turning round by having a projection fitting into the open front of column, and when the screw is rotated, the nut travels upwards or downwards along the slot in the column. A gong is fitted at each end of the nut's traverse, so that the nut-finger may give notice by striking the bell when the cage is a pre-determined distance

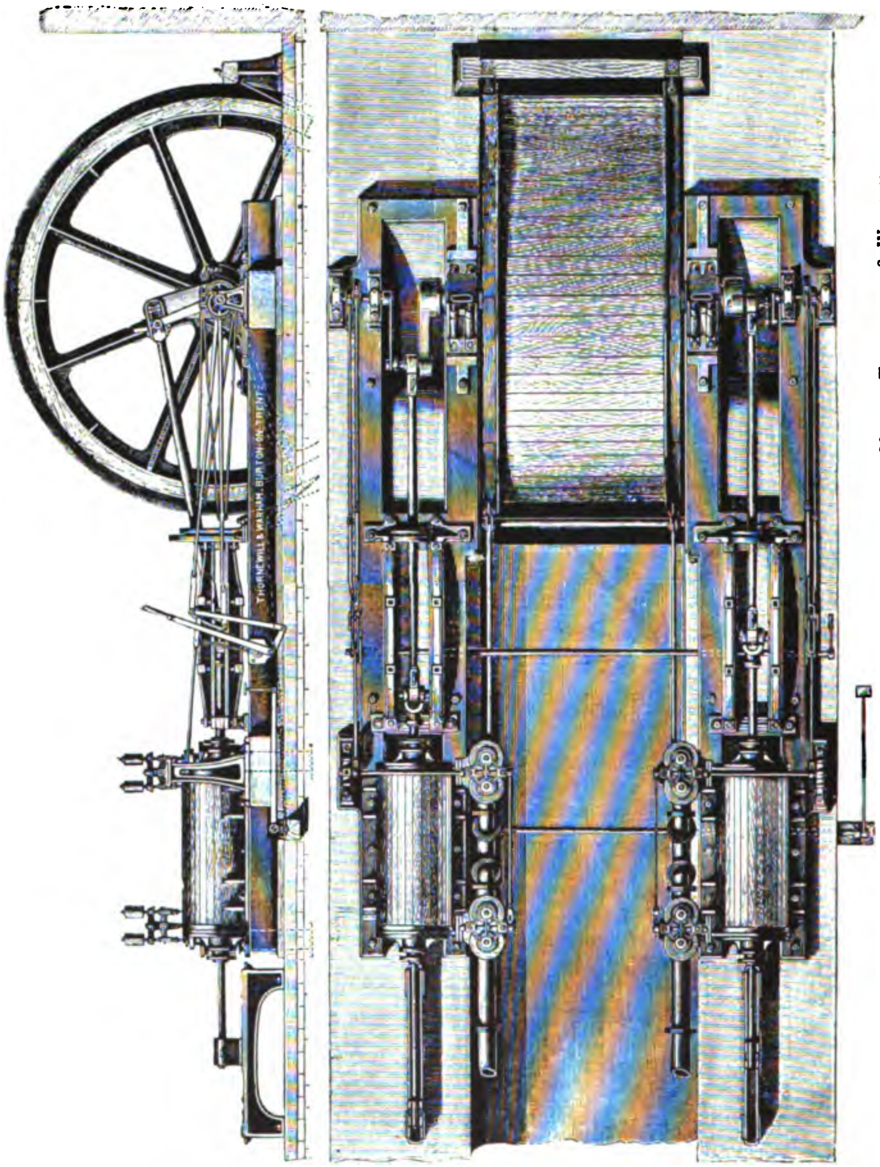


Fig. 76.—Pair of High-pressure Winding-Engines as made by Messrs. Thornewill & Warham.

from the end of its journey. This is a positive indicator, and is superior to any type depending on cords, chains, or clock-faced dials.

The drum has three "centres" to support the wood-lagging, and all are firmly keyed to the crank-shaft.

A brake, having two straps with the necessary connections, is fitted, and the reversing-handle, throttle-valve-handle, and brake-treadle, are arranged conveniently for the engineman.

All the parts are machined, and the necessary portions of the engine polished. The cylinders are clothed with non-conducting composition and mahogany lagging strips secured by polished brass bands.

All holding-down bolts are supplied, and also steam and exhaust-pipe connections to the outside of the engine-house.

The timber and labour for drum and brake-laggings are usually provided by the purchaser, but the contractor turns up the laggings after they have been fitted.

Engines are now being fitted with wrought-iron built drums; being much lighter than cast-iron they enable the engines to start more quickly and work more rapidly.

When automatic expansion is desirable it is usual to fit a tumbler to each steam-valve spindle swinging on a pin kept up to its position by a spring. The tumbler has an arm which comes into contact with a cam, the position of which is regulated by the governor that disengages the tumbler-toe from the steam-lifter on the rock-shaft; the steam-valve and spindle are then free to fall, the rate of descent being controlled by a dash-pot fitted with an adjustable valve.

COUNTERBALANCING.—In all cases of winding coal up vertical shafts with drums of equal diameters, a great disadvantage arises from the fact that the working strain is much greater on the engine at the commencement than at the end of winding.

Frequently, where the shaft is deep, the load at the "lift" is doubled, because the weight of the rope exceeds that of the coals, tubs, and cage, to be drawn up.

Let us suppose a case of a pit 1,530 feet deep in which the weight of the double-decked cage used is 36 cwt., there being one tub carried in each deck, the weight of the bridle chains is 4 cwt., that of each tub 7 cwt., with a carrying capacity of 1 ton.

The total weight for the engine to lift from the bottom would be $36+4+20+20+7+7=94$ cwt. $=10{,}528$ lbs., + the weight of the rope, which, say, is a round steel wire of 4 inches circumference weighing 8 lbs. per yard or $510 \times 8 = 4{,}080$ lbs. With a 51-foot circumference or 16·2338-foot diameter plain cylindrical drum the number of strokes in making a winding will be $\frac{1{,}530}{51}=30$.

We may proceed to consider the load on the engine in this case from a statical point of view, omitting all consideration of momentum and inertia which very much complicate the question of counterbalancing.

It will be clearer with the help of a diagram. In Fig. 77, from the zero line OO′ set off horizontally by means of the scale, the load in foot-pounds at the end of each stroke, to ascertain which multiply the load by the leverage of the winding drum, and also set off vertically the number of strokes in the winding.

The full weight at the lift is $(10{,}528+4{,}080)\times 8{\cdot}12$ the leverage $=118{,}617$ foot-pounds. At the end of the first stroke it will be $\{10{,}528+(493\times 8)\}\times 8{\cdot}12=117{,}512{\cdot}7$ foot-pounds. At the end of the second stroke it will be $\{10{,}528+(476\times 8)\}\times 8{\cdot}12=116{,}408{\cdot}4$ foot-pounds, and so on, the number of foot-pounds diminishing regularly 1,104·3 at each stroke, so that when the full tub reaches the surface the foot-pounds will be $10{,}528\times 8{\cdot}12$ the leverage $=85{,}488$. The line AA′ in Fig. 77 represents this, the full cage-load throughout the winding.

Similarly the weight in foot-pounds of the empty cage rendering assistance in its descent is shown on the diagram Fig. 77 by the line BB′, it being plotted on the opposite side of the zero line to that showing the weight of the full cage. It may be estimated thus:—

When at the surface, the foot-pounds will be (36 + 4 + 7 + 7) × 112 = 6,048 lbs. × 8·12 the leverage = 49,109·8. At the end of the first stroke it will be {6,048 + (17 × 8)} × 8·12 = 50,214·1. At the end of the second stroke it will be {6,048 + (34 × 8)} × 8·12 the leverage = 51,318·4, and so on, the number of foot-pounds increasing regularly 1,104·3 at each stroke, so that when the empty cage reaches the pit-bottom the foot-pounds will be {6,048 + (510 × 8)} × 8·12 the leverage = 82,239.

The actual work done by the engine is shown on the diagram, Fig. 77, by line CC′ which is obtained by deducting the foot-pounds of the empty cage from those of the full one.

The power necessarily exerted at the lift is 118,617 − 49,109·8 = 69,507·2 foot-pounds and at the end of the winding 85,488 − 82,239 = 3,249 foot-pounds.

A case may readily be imagined where the foot-pounds at the end of the winding will be a negative instead of a positive number, or in which the engine may have to exercise a retarding influence in consequence of the preponderance of the weight of the descending cage and rope over that of the full cage. For instance, take a shaft 1,530 feet deep as before in which the cage, chains, and tubs are of the same weight as there given, but at which flat ropes 4½ inches × $\frac{7}{8}$ weighing 15½ lbs. per yard are used for winding by means of a drum whose diameter at the start is 20 feet. Then, by the formula given on page 799 of this work, the number of revolutions will be

Scale 100,000 foot-pounds = 1 inch.

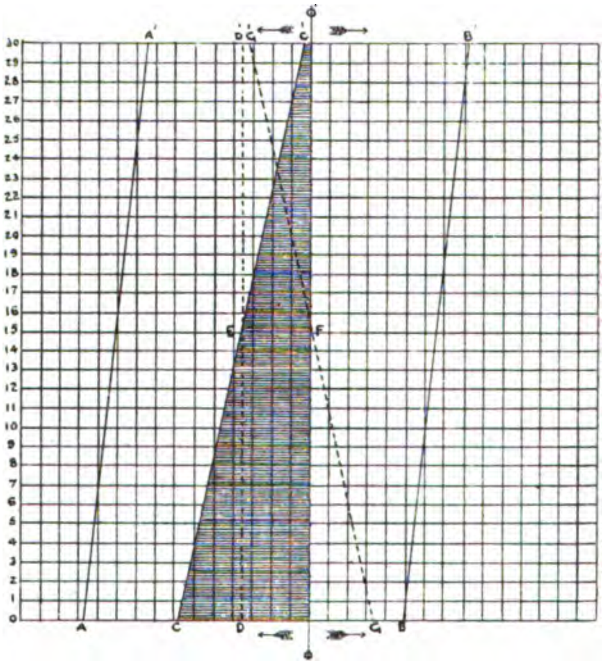


Fig. 77.—Diagram showing the Work done by Winding Engine.

$$n = \frac{\sqrt{11{\cdot}45915 \times 510 \times {\cdot}875 + \left(\frac{240}{2}\right)^2} - \frac{240}{2}}{{\cdot}875}$$

= 22·5 revolutions or strokes in the winding.

Here the weight of the full cage at the lift is $\{10,528 + (510 \times 15\frac{1}{2})\} \times 10$ the leverage = 184,330 foot-pounds. When the full cage reaches the surface the foot-pounds will be 10,528 × 11·64 the leverage = 122,546.

Similarly the weight of the empty cage when at the surface is 6,048lbs. × 11·64 the leverage = 70,399 foot-pounds, and when at the pit-bottom = $\{6,048 + (510 \times 15\frac{1}{2})\} \times 10$ the leverage = 139,530 foot-pounds.

The power necessarily exerted at the lift then is 184,330 − 70,399 = 113,931 foot-pounds, and at the end of the winding 122,546 — 139,530 = − 16,984 foot-pounds: that is, instead of a positive, a negative power of 16,984 foot-pounds must be exercised by the engine.

Referring again to Fig. 77, the dotted line DD′ represents the work actually to be done so that the load may be uniform throughout the winding, because the triangle DEC = the triangle D′EC′. In foot-pounds it is $\frac{3,249 + 69,507}{2} =$ 36,378, and is represented by FE. In order to keep the load on this line without deviation, it would be necessary to use a counterbalance whose action is shown by the line GG′ on the diagram, because the triangle GFO = the triangle CED, and the figure O′C′EF + the triangle G′FO′ = the figure O′D′EF. The counterbalance at the lift then must exercise a reducing effect to the load of 69,507·2 − 36,378 = 33,129·2 foot-pounds, but become regularly less as the winding proceeds, until at meetings it is nothing. As the winding continues beyond the cage meetings, the engine has in addition to the load to raise the counterbalance, which at the completion of winding must be represented by 33,129·2 foot-pounds.

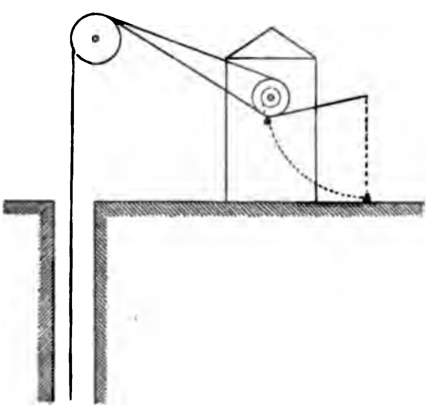


Fig. 78.—PENDULUM COUNTERBALANCE.

Different methods of counterbalancing, with a view of assisting the winding-engine at the commencement of the wind, have been adopted at different times, some of which are as follow:—

Rope balance.—This consists of a tail-rope attached to the cages in the shaft. It is fastened to the bottom part of the cage at the surface and is conveyed to a point below the loading-stage in the pit, where it passes round a pulley and then up to the other cage, under which it is secured. The object sought by this method is to make the load uniform; when the cage is lifted from the shaft bottom, the weight of the ropes in the shaft balance each other, and will continue to do so throughout the wind. An objection to the tail-rope in the shaft is the extra strain on the capping of the ropes.

Another method is by a *pendulum counterbalance*, which, however, is only applicable to shallow shafts, where the necessity for counterbalancing is not urgent. A weighted pendulum is raised by a chain attached to a drum on the shaft of the winding-engine, see Fig. 78. At the commencement of winding this pendulum is in a horizontal position, and therefore the full weight is acting to aid the engine against the load. As the winding proceeds to half-winding an increasing portion of the weight is supported by the rod, which at half-winding is in a perpendicular position, and then as the load is brought to the surface the rod returns to a horizontal position, the weight thus acting against the engine with increasing force during the last half-winding.

A third method consists in a *counterbalance chain*, see Fig. 79, which is frequently applied with very good results to assist the winding-engine. This requires a separate pit or well about 50 yards deep for the chain to work in. A rope is fixed to the drum-shaft of the engine and to the balance-chain in the small pit. The balance-chain would be 50 yards long, and is so arranged that with one cage at the surface and the other at the shaft bottom the whole length of chain is hanging in the small pit. The rope by which it is wound up allows the whole of the balance-chain to rest upon the small pit bottom when the ascending and descending cages meet in the shaft. The rope passes over the drum-shaft in a contrary direction to the drawing-rope.

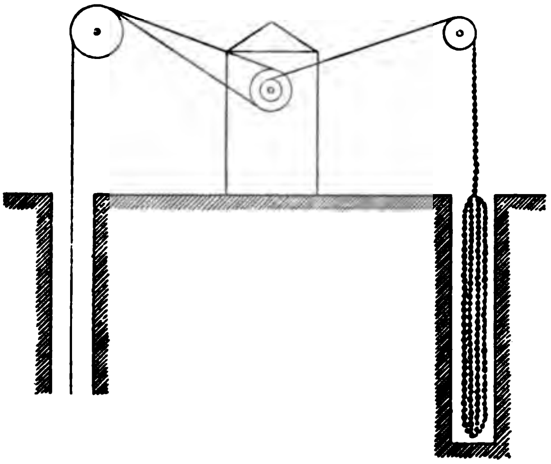


Fig. 79.—Chain Counterbalance.

A fourth method is that of the *inclined plane*, Fig. 80. Usually this is only applicable to shallow shafts, but where the engine-house is situated on high ground, and a gradual slope from it can be obtained, it admits of a long travel for the tub or truck, and so becomes applicable to greater depths of winding. Instead of the bunch of chain used in the last described method, a weighted tub or truck is attached, and this travels over an inclined plane the gradient of which must vary throughout its course, and be steepest at the commencement of the plane next the engine-house. Any attempt to use this kind of counterbalance where the gradient is uniform for the tub or truck to move on is quite useless. The rails laid should form a curve, and not break abruptly from one uniformly inclined short portion to another. At half winding the tub or truck reaches the end of its run, and is wound upwards towards the engine-house as the winding proceeds; on completion of the winding it returns to its starting point.

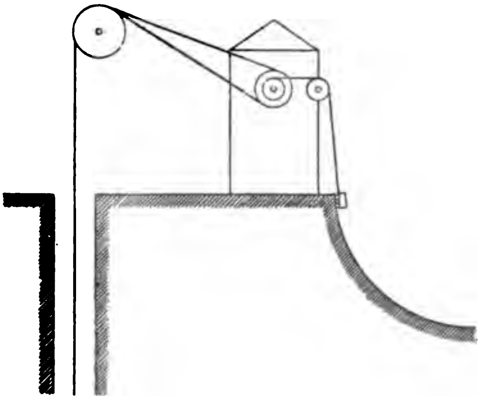


Fig. 80.—Inclined Plane Counterbalance.

Other methods of counterbalancing are by using the cone and scroll drums in preference to plain or cylindrical drums, and these have already been alluded to.

A plain conical drum, to be effective as a counterbalance, requires an angle so great as to be dangerous on account of the liability of the rope to slip. Any advantage to be derived from having a safe angle being trifling, it is really not worth the risk.

Rule 29 of the Mines Act, 1887, renders the use of flanges or horns to the drum compulsory, and if the drum is conical, there must be appliances sufficient to prevent the rope from slipping.

Supposing, in the instance we have been considering, a rope-balance be applied, and that it is of precisely the same weight per yard as the round winding rope, then it is obvious that no calculations are required, for the ropes, cages and tubs must balance each other throughout the wind; the line EF, the actual weight of the coal will be the load on the engine which will be uniform during the ascent of the cage from the shaft bottom to the surface.

The pendulum counterbalance is not applicable to a pit of this depth, we therefore proceed to consider the chain counterbalance in the instance given. Assuming the bunch of chain has to be hung in a 51-yard pit, then as the engine makes 30 revolutions, the rope roll for the counterbalance must have a circumference of $\frac{51 \times 3 \times 2}{30}$ = 10·2 feet, or a diameter of 3·24675 feet. The leverage then is 1·62338 foot, so that the weight of the bunch must be $\frac{33,129{\cdot}2}{1{\cdot}62338}$ = 20,407·5 lbs., or 9 tons, 2 cwt., 23 lbs.

When the cages are at meetings, and the whole of the bunch is lying coiled at the bottom of the counterbalance pit, there would not be such relief as would arise from a perfect counterbalancing effect, because there would be the rope in the counterbalance pit, 51 yards and possibly 19 above the pit = 70 yards at, say, 10 lbs. per yard = 700 lbs. × 1·62338 = 1,136 foot-pounds. The effect of this would cause the line DD′ to deviate from a straight one slightly at E, but a very good attempt would be made to produce a counterbalance.

Next: to design an inclined plane counterbalance in the example we are considering, which is, say, 51 yards long and the diameter of counterbalance rope roll 1·62338 foot as before:—Let a weighted truck of, say, 30 tons be used. Then for its effective weight to be similar to that of the chain bunch or 9 tons, 2 cwt, 23 lbs., the inclination for the first stroke or 10·2 feet = $\frac{30\text{ tons}}{9\text{ tons, 2 cwt., 23 lbs.,}}$ = 1 in 3·293. The foot-pounds at the next stroke must be reduced $\frac{1}{15}$th or $\frac{33,129{\cdot}2}{15}$ = 2,208·6. 33,129·2 − 2,208·6 = 30,920·6. $\frac{30,920{\cdot}6}{1{\cdot}62338}$ = 19,047 lbs. The inclination for the next stroke then must be $\frac{30 \times 2,240}{19,047}$ = 1 in 3·528. The foot-pounds must be reduced at the next or 3rd stroke $\frac{2}{15}$ths of 33,129·2 or 4,417·2. 33,129·2 − 4,417·2 = 28,712. $\frac{28,712}{1{\cdot}62338}$ = 17,686·5. The inclination for the remaining strokes will be:—

At the commencement of the 3rd stroke $\frac{30 \times 2,240}{17,686{\cdot}5}$ = 1 in 3·799.
" " " " 4th " $\frac{30 \times 2,240}{16,326}$ = 1 in 4·116.
" " " " 5th " $\frac{30 \times 2,240}{14,965{\cdot}5}$ = 1 in 4·49.
" " " " 6th " $\frac{30 \times 2,240}{13,605}$ = 1 in 4·939.
" " " " 7th " $\frac{30 \times 2,240}{12,244{\cdot}5}$ = 1 in 5·488.
" " " " 8th " $\frac{30 \times 2,240}{10,884}$ = 1 in 6·174.
" " " " 9th " $\frac{30 \times 2,240}{9,523{\cdot}5}$ = 1 in 7·056.
" " " " 10th " $\frac{30 \times 2,240}{8,163}$ = 1 in 8·232.
" " " " 11th " $\frac{30 \times 2,240}{6,802{\cdot}5}$ = 1 in 9·878.
" " " " 12th " $\frac{30 \times 2,240}{5,442}$ = 1 in 12·349.

At the commencement of the 13th stroke $\frac{30 \times 2,240}{4,081\cdot5} = 1$ in 16·464.

„ „ „ „ 14th „ $\frac{30 \times 2,240}{2,721} = 1$ in 24·697.

„ „ „ „ 15th „ $\frac{30 \times 2,240}{1,360\cdot5} = 1$ in 49·393.

„ „ end „ 15th „ becoming level.

As the winding proceeds toward completion, the loaded tub changes its direction after the 15th stroke and returns over the inclined plane whose gradients have just been estimated.

Now, to consider the last system of counterbalancing, viz., the spiral drum. If the initial diameter be 16 feet, the weight of the full cage is represented by $(10,528 + 4,080) \times 8$ the leverage $= 116,864$ foot-pounds. The weight of the coal must always remain to be overcome by the engine even in a perfect counterbalance. This would be $20 + 20 = 40$ cwt. of coal $\times 112 \times 8 = 35,840$ foot-pounds, and as the empty cage and tubs at the surface weigh 6,048 lbs., the final radius would be $\frac{116,864 - 35,840}{6,048} = 13\cdot39$ or a diameter of 26·78, changing regularly for each revolution. The number of revolutions such drum would make $= \frac{1,530}{\left(\frac{16 + 26\cdot78}{2}\right) \times 3\cdot14159} = 22\cdot76$ and therefore the diameter must increase regularly $\frac{26\cdot78 - 16}{22\cdot76} = \cdot47$ foot at every revolution.

Where a flat rope is used for winding, lapped one coil over the other, it has to a very slight extent, a counterbalancing effect. The difference between the extreme diameters is, however, a negligible quantity.

Miscellaneous.—In laying out the surface arrangements, a considerable amount of thought is necessary to ensure a thoroughly satisfactory and economical working after completion. The particular circumstances and requirements of each colliery must be thoroughly mastered in order to design an effective scheme. The engines and buildings must be adapted to their work, and after the coal is brought to the surface every operation connected with its weighing, screening, cleaning, and after-disposal, should be calculated to give as little manual labour as possible in working, and at the same time yield the various sizes of coal in a good, clean, and marketable condition.

A carelessly laid out bank top renders it necessary to keep employed a large number of workmen to deal with the daily out-put, and thus add to the cost of production. In times of keen competition even a penny extra cost on the tonnage price in raising the coal may place a colliery at a disadvantage as compared with a neighbouring one, and result in loss of contracts; with greatly increased cost prices, it may be impossible to work the colliery at all.

Mr. C. M. Percy, in his excellent work on *The Mechanical Engineering of Collieries*, gives the following rules for winding-engines. (1) To find the load which a given pair of engines will start. Multiply the area of one cylinder by the pressure of steam and twice the length of stroke. Divide this by circumference of drum and deduct ⅓ for friction, &c. The result is the load the engines can start. For instance, 2 — 20-inch diameter cylinders by 40-inch stroke, with a 12-foot diameter drum and the steam pressure at boilers 50 lbs. $\frac{314 \times 50 \times 80}{454 \text{ in.}} = 2,766 - \frac{1}{3}$ or $922 = 1,844$ lbs. the load. N.B.—The load referred to in this and the next rule comprises rope and coal, because cages and tubs balance each other. (2) Knowing the load and the diameter of drum and the length of stroke and the pressure of steam, to find the area and diameter of cylinders. Multiply the load by the circumference of the drum and add $\frac{1}{2}$ for friction, &c. Divide this by the steam pressure multiplied by twice the length of stroke, and the result is the area of the cylinder. Example, a drum 15 feet, stroke 6 feet, the steam pressure 60 lbs., and the load 7 tons = 15,680 lbs. $15{,}680 \times 47 = 736{,}960 + \frac{1}{2}$ or $368{,}480 = 1{,}105{,}440$. $\frac{1{,}105{,}440}{60 \times 6 \times 2} = 1{,}535$ square inches, area of piston, and to find the diameter we must divide the area by ·7854 and take the square root of the result, thus $\sqrt{\frac{1{,}535}{\cdot 7854}} = 44\frac{1}{4}$ inches diameter of cylinder. (3) To find the period of a winding approximately. Reckon the piston to travel at an average velocity of 400 feet per minute, and divide this by twice the length of the stroke, and multiply by circumference of drum. This gives speed of cage in feet per minute, and divide depth of pit by this, and the result gives period of a winding. Example, drum 47 feet circumference, stroke 6 feet, depth of pit 600 yards $= \frac{400}{6 \times 2} \times 47 = 1{,}565{\cdot}1$, speed of cages in shaft per minute $\frac{600 \times 3}{1{,}565} = 1{\cdot}15$ minute or about $1\frac{1}{7}$ minutes. (4) To find the *useful* horse-power during a winding. Multiply the depth of pit by weight of coal raised, and divide this by period in minutes occupied in winding, and divide again by 33,000. Example, coal 2 tons = 4,480 lbs., depth 1,500 feet, period of winding $\frac{5}{6}$ths of a minute; $\frac{1{,}500 \times 4{,}480}{\frac{5}{6} \times 33{,}000} = 244$ horse-power. (5) To find the approximate horse-power exerted by a pair of winding-engines during a winding. Multiply area of both cylinders by $\frac{2}{3}$rds of the boiler pressure of steam and by 400, and divide by 33,000. Example, cylinders 30 inches diameter = 706 square inches, boiler pressure 60 lbs., $\frac{2}{3}$rds of which is 40. $\frac{706 \times 2 \times 40 \times 400}{33{,}000} = 685$ horse-power. (6) To ascertain the piston-speed of a winding-engine. Divide the windings into periods of 3 revolutions each. Let two persons take alternately the time in seconds which each 3 revolutions occupy. Take the average from these as follows:—Suppose there are thirty revolutions divided into 10 periods, three revolutions each, the time occupied in seconds comes out 9, 7, 5, 3, 2, 2, 2, 3, 6, 9 = 48 seconds for the whole winding. Suppose the stroke 5 feet × 2 = 10 × 30 = 300 piston feet in 48 seconds and 300 × 60 = 18,000 ÷ 48 = 375 piston feet per minute.

If only the average piston speed throughout the wind is required, there is no useful object attained by dividing the winding into periods of a certain number of revolutions. The whole winding may in that case be timed, and the number of seconds occupied in the wind be divided into twice the length of stroke multiplied by the number of revolutions. Where a comparison of different rates of the varying piston speed throughout a wind is desired, it would, of course, be necessary to sub-divide the total revolutions.

It will be well to bear in mind the following rules for winding-engines:—

$$\text{Horse-power} = \frac{\text{area of cylinder} \times \text{effective pressure per square inch} \times \text{piston speed feet per min.}}{33{,}000}$$

$$\text{Area of cylinder} = \frac{\text{Horse-power} \times 33{,}000}{\text{Effective pressure per square inch} \times \text{piston speed in feet per minute.}}$$

$$\text{Effective pressure per square inch} = \frac{\text{Horse-power} \times 33{,}000}{\text{Area of cylinder} \times \text{piston speed in feet per minute.}}$$

$$\text{Piston speed in feet per minute} = \frac{\text{Horse-power} \times 33{,}000}{\text{Area of cylinder} \times \text{effective pressure per square inch.}}$$

Horse-power is the quantity of work a horse is assumed to be capable of performing, based upon experiments made by Watt. It is equivalent to the raising of a resistance of 33,000 lbs. through one foot in one minute, and is the unit employed in measuring the work of a steam engine.

A few questions on winding engines, &c., such as are given at the examinations, are now submitted, and the method of working out answers to them shown.

> *Question* 2.—What size and kind of engines would you fix to raise 1,000 tons of coal from a depth of 500 yards in 8 hours? Describe the size of ropes, description of cage, diameter of drum, pressure of steam, time of drawing and changing, the number of tubs wound at each time and weight of each?

1,000 tons of coal per day of 8 hours = $\frac{1{,}000}{8}$ = 125 tons per hour : $\frac{125}{60}$ = 2·08 tons per minute, and assume that for this large quantity the seam is a thick one, capable of taking large tubs, and that the cage will have an average running speed of 1,000 feet per minute inclusive of the stoppages to change tubs; then we have $\frac{500 \times 3}{1{,}000}$ = 1·5 minute as the time in which we must make each trip, that is, a cage comes to the surface every 1½ minutes; then as 2·08 tons per minute must be landed in order to get 1,000 tons in 8 hours, we have 2·08 × 1·5 = 3·12 tons to be drawn in each cage. Unless trams carrying over 30 cwt. be used, it would be necessary to have double-decked, or if very small trams are used, three-decked cages, but assume that double-decked cages are to be used, with 2 tubs in each deck, or 4 tubs per cage, which gives an average of $\frac{3{\cdot}12}{4}$ = ·78 ton or 15·6 cwt. of coal per tub. Tubs weigh from ⅓ to ½ their carrying capacity, and a tub to carry this weight need not weigh more than 7 cwts. As the depth is great and the weight for the engine to lift from the pit bottom must necessarily be considerable, steel should be used wherever possible in order to keep down the dead weight. The tubs and cages may be of steel, and the rope of steel wire.

Now to consider the strength of the rope required, the weight upon which at the start would be somewhat as follows:—

One steel cage and chains, say	2 tons,	5 cwt.
Coal in 4 tubs = ·78 ton × 4 =	3 „	2·4 „
4 empty steel tubs = 4 × 7 cwt. =	1 „	8 „
	6 „	15·4

beside its own weight. The weight of 250 fathoms (or really rather more as there is the height of the pulleys above the top of the pit) at say roughly 14 lbs. per fathom would be 1 ton 11 cwt., to add to 6 tons 15·4 cwt., making a total of 8 tons 6·4 cwt., so that by estimating for a rope having a safe working load of 8·4 tons, we shall be quite safe. By the formula already given to find the safe working load of a steel wire rope we have—Circumference $= \sqrt{2{\cdot}4 \times 8{\cdot}4} = 4\frac{1}{2}$ inches, which is the circumference of the steel rope required. Applying the formula given to find the size of drum, it gives 17 feet as the diameter for a plain cylindrical one. The engine must necessarily be powerful enough to lift 8 tons 6·4 cwt., therefore assume it has a 6-foot stroke, and a piston speed of 400 feet per minute. In that case $\frac{400}{6 \times 2} = 33{\cdot}3$ strokes per minute, and $33{\cdot}\dot{3} \times 17 \times 3{\cdot}14159 = 1{,}780$ feet per minute, the average speed of the cage whilst travelling from bottom to the top of pit. Therefore $\frac{500 \times 3 \times 60}{1{,}780} =$ say 51 seconds, nearly as the time occupied in the journey stated. It has already been ascertained that a cage must be drawn every 1·5 minute, or 90 seconds, in order to raise the desired quantity. There would therefore be $90 - 51 = 39$ seconds as the time in which to do the changing. To find the size of cylinder according to Mr. Percy's rule already given, assuming a boiler pressure of 60 lbs., we have as the circumference of a 17-foot diameter drum 53·4 feet, and the stroke being 6 feet, this gives 12 feet travel of piston. The weight of cages and tubs balance each other and are not to be considered, but the weight of coal is 3 tons 2·4 cwt., that of the rope 1 ton 11 cwt., making a total of 4 tons 13·4 cwt., or say $4\frac{3}{4}$ tons.

The weight or resistance of $4\frac{3}{4}$ tons at the end of a lever 53·4 feet long will be $4{\cdot}75 \times 53{\cdot}4 = 253{\cdot}65$ tons $= 568{,}176$ lbs., and the power to overcome this will be 12 (twice the length of stroke in feet) × 60 (the boiler pressure of steam) = 720 × something which will equal 568,176. Therefore $\frac{568{,}176}{720} = 789$, and this means the area of the piston in square inches. Add one-half for overcoming friction, $789 + 394{\cdot}5 = 1{,}183{\cdot}5$ square inches area of piston, and therefore $\sqrt{\frac{1{,}183{\cdot}5}{{\cdot}7854}} = 38{\cdot}82$ inches, or say 39 inches as the diameter. Inasmuch as at the lift only 1 cylinder may be applied, this size of engine is necessary. For reasons already given the engines should be placed horizontally, and have Cornish in preference to slide valves, and as has been shown with 2 cylinders of 39 inches diameter, 6 feet length of stroke, $33{\cdot}\dot{3}$ revolutions per minute (average velocity), effective steam pressure 40 lbs., and 17 feet diameter of drum. The cages to be double-decked and made of steel, to hold 2 tubs on each deck. Time of drawing 51 seconds, of changing 39 seconds, the weight of steel tubs 7 cwt. each when empty, and the circumference of the steel wire rope to be $4\frac{1}{2}$ inches.

Question 3.—What description and size of winding engines would you erect to raise 100 tons per hour from a coal-field 2,000 acres in extent, the seam being 5 feet 6 inches thick, and the depth of shaft 300 yards?

I should fix a double horizontal engine as in the last case, direct acting, and to find the size should proceed much in the same way. 100 tons per hour $= \frac{100}{60} = 1{\cdot}\dot{6}$ ton per minute, or $1\frac{2}{3}$ tons, and on the same principle we should make a trip $\frac{300 \times 3}{1{,}000} =$ every $\frac{9}{10}$ths of a minute. $\therefore 1\frac{2}{3} \times \frac{9}{10} = 1\frac{1}{2}$ tons to be drawn in each cage. We should have to choose between sending this $1\frac{1}{2}$ tons in 4 tubs of $7\frac{1}{2}$ cwt. each, or in 2 tubs holding 15 cwt. each, or in 1 tub holding the 30 cwt. The seam is high enough for all sizes, but assume that 4 tubs are sent in a double-decked cage. The tubs when empty would weigh about 4 cwt. each, for that is about one-half of their carrying capacity, and the cage and chains about a ton, or

24 cwt., assuming that iron tubs, an iron cage, and an iron wire rope be used. The weight the engine would have to lift would be—

Cage and chains, say	1 ton	4 cwt.
Coal in 4 tubs	1 „	10 „
4 empty tubs = 4 × 4 cwt.	0 „	16 „
	3 „	10 „

besides the weight of rope. The weight of 150 fathoms of iron wire rope at say roughly 14 lbs. per fathom, would be nearly 1 ton, and 1 ton + 3 tons 10 cwt. = $4\frac{1}{2}$ tons as a safe working load for the rope. Circumference = $\sqrt{4 \times 4\frac{1}{2}} = 4\frac{1}{4}$ inches as the circumference of iron wire rope necessary.

Assume a 16-foot diameter drum (Mr. André's rule would give 16 feet 6 inches), and also that the engine has a 5-foot stroke, with a boiler pressure of steam of 60 lbs., the piston speed being 400 feet per minute, $\frac{400}{5 \times 2} = 40$ strokes per minute. $40 \times 16 \times 3{\cdot}14159 = 2{,}010$ feet per minute as the average speed of the cage whilst travelling from bottom to top of pit. $\frac{300 \times 3 \times 60}{2{,}010} =$ nearly 27 seconds. As a trip must be made every $\frac{9}{10}$ths of a minute = 54 seconds, this leaves $54 - 27 = 27$ seconds in which to do the changing.

To find the size of cylinder we have the weight of coal and rope at $2\frac{1}{2}$ tons. $\frac{50{\cdot}26 \times 2\frac{1}{2} \times 2{,}240}{5 \times 2 \times 60} = 469{\cdot}09 + 234{\cdot}54$ for friction $= 703{\cdot}63$ and $\sqrt{\frac{703{\cdot}63}{{\cdot}7854}} =$ 29·93, or say 30 inches as the diameter of the cylinders.

Question 4.—What is meant by such expressions in machinery as horse-power, units of work, and back-pressure of the steam?

The English unit of work is the work done in overcoming a resistance of one pound avoirdupois through a space of one foot, and is spoken of as the foot-pound. It is assumed that a horse will raise 33,000 lbs. 1 foot high in 1 minute, therefore we call one horse-power, 33,000 lbs. raised 1 foot high in 1 minute. Back pressure of the steam is the resistance the piston of an engine meets with from the spent steam as it is forced into the exhaust port, and its tendency is to lessen the effect of the engine.

Question 5.—What is meant by the duty of an engine?

The amount of work yielded by that engine, or the number of lbs. raised 1 foot high by the combustion of a given quantity of coals.

Question 6.—Why is a machine put in motion by steam?

Because the pressure of the steam on being applied to the piston of the engine is greater than the combined resistance offered by the weight attached and the friction of the engine itself.

Question 7.—What is steam and how is it obtained? What do we mean by such expressions as high pressure and low pressure steam?

Steam is an invisible elastic fluid, generated from water by the application of heat. It is measured by pressure gauges, and we speak of it as being so many pounds pressure to the square inch. We also speak of steam having a pressure of 15 lbs. to the square inch as one atmospheric pressure; 30 lbs. as two atmospheres, 45 lbs. as three atmospheres, and so on. Steam of two atmospheres and above is called high pressure steam, and below two atmospheres, low pressure steam.

> *Question* 8.—Describe an experiment for ascertaining approximately the relation between the pressure and temperature of steam at a moderate pressure above that of the atmosphere.

This experiment can be made by means of a Marcet's boiler, see Fig. 81. The apparatus consists of a vessel in which is first placed a little mercury, G, and above the mercury a small quantity of water, F; above the surface of the water a space is left for the steam. A piece of glass tube, A, about a yard long, and open at both ends, is passed steam tight through the top of the boiler, and reaches nearly, but not quite, to the bottom, so as to be well into the mercury. It is placed in an upright position. Similarly, a thermometer, C, is passed through the top of the boiler, but its bulb does not quite reach the surface of the water. The glass tube is furnished with a graduated scale, B, placed outside the boiler, by means of which the height of mercury in the tube above that in the boiler can be read off approximately. A cock, D, is also placed outside the boiler, above the surface of the water, so that when it is opened the enclosed air may be driven out through it by the steam. The boiler being placed conveniently for the purpose, a Bunsen flame, K, is applied under it, causing the water to boil and generate steam.

On this steam attaining a greater pressure than the pressure of the external air, the mercury will begin to rise in the tube. By opening the cock for a short time the steam will drive out the enclosed air from the boiler, and the cock may be shut again. As the pressure of the steam above the atmosphere increases, it will be noticed that the mercury rises in the tube, and at the same time that the temperature of the steam also rises above 212° F., as seen on the thermometer.

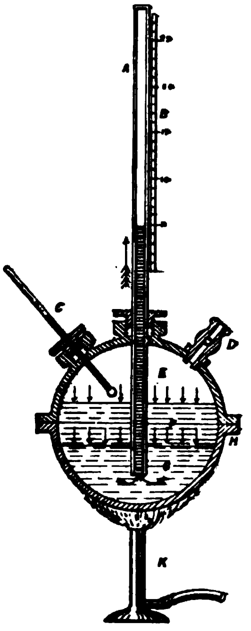


Fig. 81.—MARCET'S BOILER.

The cock must now be carefully adjusted so as to allow the exact quantity of steam being generated in a certain time to pass through it to the outer air, in the same time.

The temperature and the pressure corresponding to that temperature will then be constant, and may be read off and recorded from their respective scales.

By closing the cock again, the temperature may be allowed to rise, say, 10°, upon reaching which the cock is again adjusted as before, and the temperature and pressure again noted. This operation may be repeated for any reasonable number of times, and it will be seen that for a given temperature there is a corresponding pressure, or vice versâ.

When the temperature is 233° F. the corresponding pressure will be 7·4 lbs., half an atmosphere, or 22 lbs. absolute, for the mercury will have risen to 15 inches.

At a temperature of 250° F. the mercury will have risen to about 30 inches, corresponding to a pressure of 14·7 lbs. on the square inch, or 1 atmosphere, or 29·4 lbs. absolute.

Further experiment would be impracticable, as the height of the glass tube, and the amount of mercury in the boiler limit the proceeding. Mercurial gauges on this principle were at one time used to register the pressure of the steam in boilers, before Bourdon's gauge was introduced.

> *Question* 9.—What are *saturated*, *superheated*, and *wet* steam respectively? Define a thermal unit, and explain what is meant by the latent heat of steam.

Saturated steam is steam in contact with the water from which it is generated. It is at its greatest density corresponding to the temperature and pressure at which it is formed. Its physical condition is such, that at the slightest increase of pressure or decrease of temperature it is ready to yield some portion of liquid. For a given pressure there is one temperature and one density. The steam usually supplied to engines from ordinary boilers is what is called *saturated* steam.

Superheated steam is saturated steam which has received an additional quantity of heat after it is free from contact with the water. It has a higher temperature than that corresponding to its pressure, consequently, when used in the cylinders of an engine, no liquefaction takes place, for although superheated steam may lose heat in the cylinder, its temperature never falls so low as the temperature of saturation. Superheated steam is not now used in the cylinder, however, owing to many practical objections.

Wet steam is steam containing water. Steam in this condition is sent to the cylinders of an engine through "priming" at the boilers. Some boilers are liable to "prime," the chief causes giving rise to it being, small steam space in the boilers, and also a bad circulation of water. The loss of efficiency in using it arises from the fact that the water carried into the cylinder produces no pressure on the piston, and does no work. Whilst in the cylinder it condenses the freshly entering steam, so that there is a loss from waste of heat.

The British thermal unit is the heat necessary to raise 1 lb. of water by 1° F., when at its maximum density, *i.e.*, from 39·1° to 40·1° F.

Latent heat is the quantity of heat communicated to a substance in a given state in order to convert it to another state without changing its temperature. The latent heat of steam is the heat necessary to convert water at the boiling point (which varies with the pressure) into steam without changing its temperature. Latent heat is not sensible to the thermometer, and is measured in a special manner. The latent heat of steam at atmospheric pressure is found to be 966·6. That is, it requires 966·6 thermal units of heat to convert 1 lb. of water at 212° F. into steam at the same temperature, or 1 lb. of steam at 212° F. gives out 966·6 thermal units of heat in being condensed into water at the same temperature.

> *Question* 10.—If one pound of coal develops 12,000 units of heat by its complete combustion, how much water at 60° F. should be converted into steam at 212° by the consumption of one cwt. of such fuel, assuming that there is no loss of heat during the operation?

The total heat of evaporation is the sum of the sensible and latent heats of evaporation. The heat necessary to raise the temperature of 1 lb. of water from the freezing point, 32° F., to the temperature of evaporation is called the sensible heat, and the additional heat necessary to evaporate it is termed the latent heat.

The total heat of evaporation for water is the number of thermal units required to raise 1 lb. of water from the freezing point, 32° F., to the stated temperature at which it is being evaporated, and to evaporate it at that temperature.

Hence the total heat in 1 lb. of steam = 966·6 + (212° − 32°) = 1,146·6°.

As the water is at 60° F., the units of heat required to convert it into steam will be 1,146·6° − (60° − 32°) = 1,118·6° F. Or 966·6° + (212° − 60°).

If one pound of coal gives 12,000 units of heat, one cwt. will give 12,000 × 112 = 1,344,000 units.

Therefore the water converted into steam will be $\frac{1,344,000}{1,118·6}$ = 1,201·5 lbs.

Question 11.—Describe condensing and non-condensing, high-pressure and low-pressure, horizontal and vertical, expansive and non-expansive, direct-acting and geared, single and double engines?

A high-pressure engine is one working with steam of 30 lbs. absolute pressure to the square inch or above, but as its exhaust is open to the atmosphere, it is opposed by the atmospheric pressure of 15 lbs. to the square inch, and its effective pressure is thereby reduced 15 lbs. to the square inch.

A low-pressure engine works with steam below 30 lbs. pressure absolute (or 15 lbs. above atmospheric pressure) to the square inch, and may or may not have a condenser for receiving the exhaust steam. A condensing engine has a condenser and air-pump connected with it, and the steam having performed its work in the cylinder, is admitted to the condenser, where it is condensed by coming into contact with a scattered jet of cold water which meets it there, and the condensed steam and water then fall to the bottom of the condenser. All water contains air more or less, and the heat of the steam disengages the air from the condensing water, which would rise through the exhaust pipe and so prevent the proper escape of steam as well as counteract its pressure, if provision were not made for its disposal otherwise. This disposal is by means of the air-pump, which pumps out the condensed water and air, thus preventing the air from accumulating in the condenser, where it would obstruct the engine. But the air-pump does not effectually remove the condensed water and air; if it did, and all the exhaust steam from the cylinder were condensed, a perfect vacuum would be formed: as a matter of fact this is never attained in practice, but there always remains in the cylinder a portion of uncondensed steam; the resistance from this is frequently equal to 4 lbs., and is termed the back pressure. With 4 lbs. of back pressure we should have 11 lbs. of vacuum, because 15 lbs. (the atmospheric pressure) − 4 lbs. = 11 lbs. So that with a given pressure of steam in the boiler we should have 11 lbs. more effective pressure per square inch on the piston of a condensing engine as compared with that on the piston of a high-pressure engine.

A non-condensing engine has no condenser, but exhausts its steam direct from the cylinder by means of pipes into the atmosphere. Horizontal and vertical engines, as the names imply, indicate whether the cylinders are placed vertically or horizontally. An expansive engine is one having an arrangement whereby the steam is cut off when a certain point of the stroke is reached, the stroke being completed by the elasticity of the steam already in the cylinder. In a non-expansive engine the initial pressure of the steam is maintained throughout the stroke, except in so far as it is not reduced by the throttle or steam valve, which reduction is termed "wire drawing." A direct-acting or first motion engine conveys motion to the main shaft direct by means of the crank, but in a geared engine spur and pinion wheels are used, which have the effect of reducing the speed of the drum, but increase the load the engine will lift. Sometimes a single cylinder is used, sometimes two working on the same shaft, and they are said to be single and double respectively.

Question 12.—Does the pressure on the piston increase or decrease during expansion? Explain the effect of using steam expansively? How is the mean pressure of the steam throughout the stroke ascertained?

The pressure of steam decreases on the piston during expansion. Mariotte's law with regard to the expansion of steam is as follows:—If a given weight of steam be made to vary its volume without changing its temperature, the elastic force of the steam will vary in the inverse ratio of the volume it is made to occupy.

Although there is a great economy in steam, and consequently in fuel, when using steam expansively, there is not actually more work done in a steam-engine working expansively, than where the same pressure is allowed to remain throughout the stroke. The advantage gained by expansion is the fact of getting more work out of a given quantity of steam. This may be explained by working out examples. The following is from *The Steam Engine*, by T. Baker. The rule to find the work done by expansion without the use of logarithms is this. Divide that part of the stroke through which expansion takes place into any even number of equal parts, and calculate the pressure per square inch upon the piston at each division of the stroke by Mariotte's law; take the sum of the extreme pressure in pounds per square inch, four times the sum of the even pressures, and twice the sum of the odd pressures; multiply the sum of all these by one-third of the common distance between the positions of the piston and the result will be the work done upon each square inch of the piston after expansion begins. The work done before expansion begins, being evidently equal to the pressure per square inch multiplied by the number of feet moved before expansion; and the whole work done during a single stroke is equal to the sum of the work done before and after expansion.

Example.—The pressure of steam upon the piston is 60 lbs. per square inch, the resistance arising from imperfect condensation is 4 lbs. per square inch, the length of the stroke is 12 feet, and the steam is cut off at $\frac{1}{6}$th of the stroke; it is required to determine the number of units of work done upon each square inch of the piston, the number of units of work gained by working expansively, and the load per square inch.

By dividing the remaining portion of the stroke after the steam is cut off, viz., $12 - 2 = 10$ feet into 10 equal parts, each will be a foot. Let the pressures at these different divisions be represented by p, p_1, p_2, p_3, &c., and then by Mariotte's law already given.

$$\begin{array}{rl}
\text{As } 3:2::60:p_1 & = 40 \\
4:2::60:p_2 & = 30 \\
5:2::60:p_3 & = 24 \\
6:2::60:p_4 & = 20 \\
7:2::60:p_5 & = 17{\cdot}142 \\
8:2::60:p_6 & = 15 \\
9:2::60:p_7 & = 13{\cdot}333 \\
10:2::60:p_8 & = 12 \\
11:2::60:p_9 & = 10{\cdot}909 \\
12:2::60:p_{10} & = 10
\end{array}$$

Then $60 + 10 = 70$, the sum of extreme pressures.

$$\begin{array}{rl}
40 & \\
24 & \\
17{\cdot}142 & \\
13{\cdot}333 & \\
10{\cdot}909 & \\
\hline
105{\cdot}384 & = \text{the sum of the even pressures.} \\
4 & \\
\hline
421{\cdot}536 & = 4 \text{ times the sum of the even pressures.}
\end{array}$$

30
20
15
12
77 = the sum of the odd pressures.
2
154 = twice the sum of the odd pressures.
421·536 = 4 times the sum of the even pressures.
70 = sum of the extreme pressures.
3)645·536
215·178
120 = 60 × 2 = work done before expansion.
335·178 = whole work done upon each square inch.

Since the resistance from the uncondensed vapour is 4 lbs. per square inch, then 12 × 4 = 48 lbs. = whole resistance, and by subtracting this from 335·178, the whole work done per square inch there will remain 335·178 − 48 = 287·178 for the effective work.

Now to find the advantage from working the steam expansively:—

When the steam works without expansion then 12 × 60 = 720 = work done upon each square inch, but as the steam is cut off at $\frac{1}{6}$th of the stroke in working it expansively, there is only $\frac{1}{6}$th of the quantity of steam used in this case, or $\frac{720}{6}$ = 120 lbs. per square inch, and 335·178 − 120 = 215·178 lbs. gained in this case, or $\frac{335·178}{120}$ = nearly 2·8 times as much work done by the same quantity of steam when worked expansively.

To ascertain the mean pressure throughout the stroke it is usual to divide its length into 10 even spaces and ascertain what the pressure would be at each division. An average of these gives the mean pressure on the piston throughout the stroke.

The even divisions will be $\frac{12}{10} = 1·2$ foot apart. We may find the pressure at any point of expansion by Mariotte's law. Thus, the pressure of steam at 0 is 60 lbs.; at 1·2 foot it is 60 lbs.; at 2·4 feet it is $\frac{60 \times 2}{2·4} = 50$ lbs.; at 3·6 feet it is $\frac{60 \times 2}{3·6} = 33\frac{1}{3}$ lbs.; at 4·8 feet it is $\frac{60 \times 2}{4·8} = 25$ lbs.; at 6 feet it is $\frac{60 \times 2}{6} = 20$ lbs.; at 7·2 feet it is $\frac{60 \times 2}{7·2} = 16\frac{2}{3}$ lbs.; at 8·4 feet it is $\frac{60 \times 2}{8·4} = 14\frac{2}{7}$ lbs.; at 9·6 feet it is $\frac{60 \times 2}{9·6} = 12\frac{1}{2}$ lbs.; at 10·8 feet it is $\frac{60 \times 2}{10·8} = 11\frac{1}{9}$ lbs.; and at 12 feet it is $\frac{60 \times 2}{12} = 10$ lbs.

Then $\frac{60 + 60 + 50 + 33\frac{1}{3} + 25 + 20 + 16\frac{2}{3} + 14\frac{2}{7} + 12\frac{1}{2} + 11\frac{1}{9} + 10}{11}$ = 28·4 lbs. as the average pressure of the steam approximately, from which must be deducted the average back pressure to get the mean effective pressure. Engineers usually ascertain the mean pressure by means of indicator diagrams, on which, after being taken, ten ordinates are drawn and measured by means of a suitable scale, as described in Chapter XVI. of this work. The ordinates represent the pressure at the points of division throughout the stroke similar to those which we have here considered.

> *Question* 13.—If 1 cubic inch of water at 39° F. gives 1,700 cubic inches of steam, what will be the expansion at 225° F.?

If we take a vessel of water and apply sufficient heat to it, the temperature will gradually rise till it reaches 212° F., the volume being increased from 1 at 39° F. to 1·04315 at the boiling point. At this point if the vessel be an open one the temperature of the water becomes stationary, and bubbles of invisible vapour or steam are formed at the surface exposed to the source of heat. These rise through the liquid causing ebullition and escape into the air where they soon become partially condensed and are then rendered visible. If the water be contained in a close vessel, the pressure of the steam generated gradually increases, until at last if no escape be provided, it will burst the vessel. If steam be allowed to enter an empty vessel we find that it occupies a very large space as compared with the water from which it is produced; the increase in volume being about 1,700 times. Roughly, as being a rule easily remembered we say, a cubic inch of water when converted into steam at the ordinary pressure of the atmosphere (14·7 lbs. per square inch) occupies the space of a cubic foot. Strictly, the specific volume of steam at 212° F. from water at 212° F., and at ordinary atmospheric pressure, is 1,642. If the pressure be increased the volume will be diminished in a corresponding degree, the steam produced from a cubic inch of water will only occupy about half a cubic foot when at a pressure of two atmospheres or 29·4 lbs.

The relation between the pressure and temperature of steam is of great importance, and numerous experiments have been made to determine the temperature when the pressure is known. An important property of elastic fluids was discovered by Charles, and further experimented on by Gay-Lussac, *i.e.*, that if the temperature of a given weight of any elastic fluid under constant pressure be made to vary, it will acquire augmentations of volume exactly proportional to the augmentations of temperature, and for every increase of 1° F. of temperature, there will be produced an increase of ·00203 of the volume of the fluid from the temperature of 32° F.

The value of this constant is sometimes taken at ·002036.

If v be the volume of any given weight of elastic fluid under any pressure and at 32° F., the volume v_1 which it will occupy under the same pressure and at any other temperature t of F. will be $v_1 = v + v \times {\cdot}0020\dot{3}(t - 32)$. This will hold good if we put the ratio of the relative volumes u and u_1 instead of the ratio of the absolute volumes v and v_1, thus $\frac{u}{u_1} = \frac{1 + {\cdot}0020\dot{3}\,(t - 32)}{1 + {\cdot}0020\dot{3}\,(t_1 - 32)}$.

As stated in the question the value of u_1 is 1700, of t_1 212° and of t 225° and we desire to find the value of u. Substituting these known values in the above equation we have $\frac{u}{1{,}700} = \frac{1 + {\cdot}0020\dot{3}\,(225 - 32)}{1 + {\cdot}0020\dot{3}\,(212 - 32)} \therefore \frac{u}{1{,}700} = 1{\cdot}019326$ and $u = 1{,}700 \times 1{\cdot}019326 = 1{,}733$ which is the relative volume required. That is a volume of steam at 212° F. represented by 1,700 would have a volume of 1,733 at 225° F., at the same pressure, or we may say that any volume of steam at 212° will have a volume at 225°, under the same pressure 1·019326 times greater.

But if this steam is generated in a boiler with the steam in contact with the water, the pressure as a matter of fact would not remain the same, and the volume it would occupy in the boiler instead of being greater at 225° would be less than at 212°, but the pressure would increase.

Saturated steam is not a perfect gas, and there are several formulæ for ascertaining the relationship subsisting between its temperature and absolute pressure. The more simple of these are only applicable, and that approximately, to certain ranges of change.

Tredgold's formula for pressures from 1 to 4 atmospheres is $p = \left(\frac{103 + t}{201{\cdot}18}\right)^6$. Applying this first to the pressure at 212° we shall find that it complies with the 14·7 lbs. already stated, thus $p = \left(\frac{103 + 212}{201{\cdot}18}\right)^6 = 14{\cdot}73$; and for the pressure at 225° we have $p = \left(\frac{103 + 225}{201{\cdot}18}\right)^6 = 18{\cdot}7816$ lbs., showing as stated an increased pressure.

From the formula to find the relative volumes when both temperature and pressure change at the same time $u = 1{,}700 \times \frac{p_1}{p} \times \frac{1 + {\cdot}0020\dot{3}(t - 32)}{1 + {\cdot}0020\dot{3}(t_1 - 32)}$ and substituting the known values in the formula $u = 1{,}700 \times \frac{14{\cdot}7}{18{\cdot}7816} \times \frac{1 + {\cdot}0020\dot{3}(225-32)}{1 + {\cdot}0020\dot{3}(212-32)}$, but as already worked out the last item in the above was found to equal 1·019326, it will render the work easier to substitute it, thus $u = 1{,}700 \times \frac{14{\cdot}7}{18{\cdot}7816} \times 1{\cdot}019326$ and therefore $u = 1{,}359$ which is the volume 1,700 cubic inches at a pressure of 14·7 lbs. and at 212° F. will occupy at a pressure of 18·7816 lbs. and at 225° F.

> *Question* 14.—What weight would a pair of 22-inch cylinder horizontal engines with a 4½-foot stroke and an 8-foot cylindrical drum on the first motion raise from a pit 260 yards deep with a round wire-rope, the boiler pressure being 40 lbs. per square inch? The engine works expansively, the steam being cut off at ¾ stroke.

With a boiler pressure of 40 lbs. the initial pressure of the steam in the cylinder would be, say ⅔rds of this 27 lbs

To find the pressure at the end of the stroke or at any point during expansion proceed by the following formula :—

P = Initial pressure of steam in pounds per square inch including the pressure of the atmosphere.

l = Distance travelled by the piston before steam is cut off.

L = Distance travelled by the piston when the pressure of the steam = X.

X = Pressure of steam in the cylinder including the pressure of the atmosphere, when the piston has travelled a distance L.

$$X = \frac{Pl}{L}.$$

$$\therefore X = \frac{(27 + 15) \times 3}{4} = 31\tfrac{1}{2}$$ from which deduct 15, the atmospheric pressure $= 16\tfrac{1}{2}$.

The average pressure throughout the stroke then is $\frac{27 + 27 + 27 + 16\frac{1}{2}}{4} = 25$ nearly. For an average pressure of 25 lbs. on the piston throughout the stroke take a boiler pressure of $25 \times \frac{3}{2} =$ nearly 38 lbs.

To find the load these engines will lift from the stated depth, and the other particulars as given. The circumference of the drum is $8 \times 3{\cdot}1416 = 25{\cdot}1328$ feet, and the area of a 22-inch cylinder is $22^2 \times {\cdot}7854 = 380{\cdot}133$. Therefore $\frac{380{\cdot}133 \times 38 \text{ lbs.} \times 108}{25{\cdot}1328 \times 12} = 5{,}172$ from which must be deducted ⅓rd for friction 5,172 — 1,724 = 3,448 lbs. and this load which the engines will lift comprises the rope and coal. The cages and tubs balance each other and need not be considered. The round wire rope used must have a safe working load then of 3,448 lbs. plus the weight of cage and tubs, say for an iron wire rope a safe working load of 57 cwt. and a $3\frac{3}{8}$-inch circumference rope will be required weighing $9\frac{1}{2}$ lbs. per fathom for the winding. In a pit 260 yards or 130 fathoms deep the total weight of such rope would be $130 \times 9\frac{1}{2} = 1{,}235$ lbs., and therefore the actual weight of coal these engines will lift is $3{,}448 - 1{,}235 = 2{,}213$ lbs. or 19 cwts. 3 qrs. 1 lb., or rather under a ton.

Question 15.—What is meant by initial, mean, and terminal pressure of the steam?

Initial is the full pressure of steam per square inch acting on the piston over a portion of its stroke previous to the closing of the steam valve. Mean pressure of steam expresses the average pressure of steam throughout the stroke, an example of how to get which is given in the answer to Question 12. The terminal pressure is that acting on the piston at the close of its stroke. Thus if steam enters the cylinder of a non-condensing engine with a 4-foot stroke at a pressure of 60 lbs. to the square inch and the steam is cut off at a quarter stroke, the absolute pressure or the pressure including that of the atmosphere is 60 lbs. + 15 lbs. = 75 lbs. By the formula in Answer 14, the terminal pressure is $\frac{75 \times 1}{4} = 18{\cdot}75$ absolute $\therefore 18{\cdot}75 - 15 = 3{\cdot}75$ lbs. as the terminal pressure shown by the pressure-gauge. As the engine is non-condensing it is necessary to deduct 15 the atmospheric pressure from the 18·75.

Question 16.—What provision should be made for letting the condensed water out of horizontal winding engine cylinders? and is the condensed water liable to accumulate elsewhere?

A small pipe with drain-cock should lead from either end of the cylinder and from the under side of it. There will always be a little condensation after the engine has stood some time on admitting the steam to the cylinders. Sometimes these are covered with wood and sometimes with cement, or they may be steam-jacketed as a means of retaining their heat. The steam pipes leading to the cylinders of the engine, if long, present a large cooling surface causing condensation and therefore should also have a separator and steam trap in the bend at the lowest point to allow for the escape of the condensed vapour.

Question 17.—Where would you place the brake for a winding engine and which kind do you prefer?

I prefer a well designed foot brake for small or ordinary sized engines, because a steam brake requires a good deal of fitting up, and when used comes into action very abruptly and suddenly causing some shock to the machinery, and a good foot-brake meets all the requirements, comes gently into operation, and can be gradually or firmly applied at pleasure. In powerful engines with heavy drums of large size subjected to rapid winding, the momentum acquired by the drums renders a greater brake power necessary, and where these are used, a steam brake should be adopted, in order that the engine-man may have greater control over the engines. When the brake is applied to the drum, a ring is formed there which may have either one or two iron straps lined with wood or hemp rope forming the brake, and care should be taken that when the brake is "off" these straps clear the drum ring. The reason for placing the brake on the drum in preference to the fly wheel is the fact that in case of accident to any part of the machinery the engine-man could at once apply the brake on the drum, thus preventing the load from falling. A brake on the fly wheel would not have so much holding power as one on the drum ring, and in case of a broken main shaft between the drum and the fly wheel, or injury to the fly wheel, a brake on it would have no power to prevent the load from falling down the shaft.

Question 18.—What is the use of a fly wheel on a steam engine? How are its size and weight determined?

The object of the fly wheel is to render the motion of the engine uniform. If we take the case of a single cylindered horizontal engine, at the finish of the backward and forward movement of the piston in the cylinder, the piston-rod, connecting-rod and crank form one straight line, and the engine is said to be "on centre," and it is clear that any pressure on the piston when in these positions will be merely transferred to the axis of the shaft to which the crank is connected and to the bearings carrying that shaft, and therefore the pressure has no power or tendency to turn the shaft round. At these points then the crank loses all power and ceases to act. When two engines are connected to the same shaft, they are so arranged that their cranks form a right angle with each other. This prevents the engines ever being "on centre," because, when one engine is in that position, the force of the other is at its greatest, but even with a pair of engines the motion would be very irregular and uneven but for the fly wheel. When the crank is in its most advantageous position, the tendency is to increase the speed of the engine; owing, however, to the weight of the fly wheel providing a mass of matter capable of being acted upon, the fly wheel takes up a portion of the power, to be given out when the crank comes into its least effective position. Thus, the fly wheel stores up sufficient power to carry the engine over the dead points and assist in maintaining a uniform motion. The weight of the fly wheel should be so adjusted as to bear a due proportion to the power of the engine. If too heavy, it is a needless addition to the load of the engine, and if too light, the object of its use will not be gained. The practical rule is that the power stored up in it should be about equal to that produced by 6 half strokes. Thus, if the steam exert a pressure of a ton on the piston and the length of the stroke be 4 feet, the power thus generated is 6 × 4 × 1 = 24 tons. The weight and velocity of the wheel should therefore be so arranged that its momentum is about equal to this. If the weight of the rim then be 1 ton its velocity should be that which would be acquired by a body falling 24 feet; if it weigh $1\frac{1}{2}$ tons it should be that acquired in falling 16 feet, and so on in proportion.

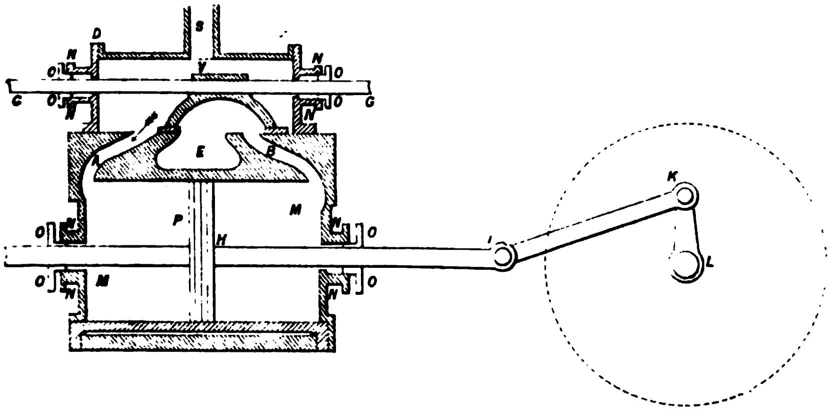


Fig. 82.—Illustrating the Action of a Winding-Engine.

Question 19.—Explain the working of the winding engine; show by a sketch its chief parts and give their names.

In the annexed sketch (Fig. 82), S is the steam pipe leading from the boilers; C D is the steam or valve chest; V is the slide valve; G G is the valve or slide rod; A & B are the steam ports; E is the exhaust port; P is the piston; H I is the piston rod; I K is the connecting rod; K L is the crank; M is the cylinder; N N are the stuffing boxes; O O are the glands.

The steam enters the valve chest D C through the steam pipe S. It then passes into one of the steam ports A or B, but if one is open the other is closed and the steam may be made to enter either first at the will of the engine-man who, by means of reversing gear, can move the slide valve V and so close either one of the steam ports and at the same time open the other.

Assuming it to have entered on the side of the piston P, shown by an arrow on the sketch, *i.e.*, through the steam port A, the piston will make a forward movement to its full extent, but whilst doing this a movement is conveyed in the opposite direction by means of the slide rod G G to the valve V, and this movement causes the valve to close over the steam port A, and at the same time it has cleared the steam port B, and placed the valve so that the steam may enter at B to drive the piston back, and also convey the exhaust steam which has just finished doing its duty through the exhaust port E leading to the condenser or the atmosphere. This backward and forward or reciprocating movement of the piston is conveyed, by means of the piston rod and connecting rod, to the crank K L which turns the main shaft on which is placed the drum, and the drum thus revolves.

Question 20.—What causes the motion spoken of in the slide rod, and what is the object of placing stuffing boxes and glands for it and for the piston rod to work in?

Motion is conveyed to the slide rod by means of two eccentrics placed on the main shaft, working in eccentric straps, and rods connect these by means of links to the slide rod, the two eccentrics being necessary for the reversing of the engine.

The object of stuffing boxes and glands is to allow the rods to move up and down without excessive friction, and yet at the same time prevent any leakage of the steam.

A well-lubricated packing is placed in the stuffing box, and a gland of brass is screwed down to keep this in place. Brass bearings cause less friction for iron to work in than iron.

Question 21.—Of what are the cylinders and pistons made, and in the continual motion of the piston in the cylinder, what provision is made for wear to the part exposed to the constant friction?

The cylinder is made of cast iron and is of considerable thickness. Its interior surface is bored with great care, so that it is of uniform diameter throughout and perfectly cylindrical. When the uniformity of diameter is destroyed through wear, the cylinder is re-bored. Covers are bolted on each end and the joints packed, so as to be perfectly steam-tight. Pistons are also made of cast iron, but sometimes they are made of brass. It is of the utmost importance that the piston be in perfect contact at all parts with the cylinder, so that no steam may pass by, and yet in that contact the friction must not be needlessly increased by too strong a pressure against the sides. For this purpose different kinds of packing are used. Formerly it was the custom to turn a deep groove round the edge of the piston disc, and into this groove was inserted well-lubricated hemp packing. The piston was made in pieces, and the top disc was attached to the rest by screws. These screws on being tightened forced the packing against the sides of the cylinder, thus preventing the steam from passing and preventing undue wear of the piston or cylinder. From time to time, as required, the cylinder cover would be removed, the piston withdrawn from the cylinder, and the packing renewed. The trouble and inconvenience resulting from this method are now, however, avoided by using metallic pistons of different construction. By using these no packing is required. For the metallic pistons the grooves formed are rectangular in form, and instead of the hemp, two or more packing rings are used which, at first, were made of brass or steel. These packing rings were flat, and had the same external diameter as the piston; they were made in several segments, the ends being tongued and grooved to keep them in position; and the joints were so arranged in each ring as to be intermediate to those of the other. These segments were constantly acted on by strong steel springs placed in the piston, the effect of which was to press them against the interior of the cylinder, and as they became gradually worn they were pressed out to exactly fit it. There were four or five springs, all tightened by screws placed for the purpose, and they were so adjusted as to keep the packing rings in contact with the interior of the cylinder without causing undue friction. The packing rings are now nearly always made of cast-iron. After turning they are left rather larger than the bore of cylinder. They are then cut across, so as to allow of compression sufficient for entering the cylinder with the piston. Where the rings are cut across they are connected by a thin plate, and the joints in the two rings are placed as far apart as possible and opposite each other, so as to ensure steam-tightness. Proper oil receptacles are placed on the cylinders, and these are kept continually replenished with good lubricating oil which passes into the interior, and very little wear then results to the metallic piston; but when necessary they can be easily repaired.

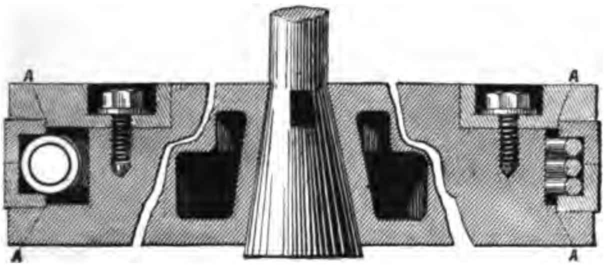


Fig. 83.—Lancaster Piston Block.

It is usual to screw the piston rod into the piston. Formerly, where no tail rod was used, the piston rod was passed through the piston and secured on the other side by a nut. After working some time the nut would wear loose and so give trouble. The object of having a tail rod on a horizontal winding-engine is to steady and guide the piston in its motion.

There are very many patent forms of pistons, in each of which is some novelty for which the inventor claims superiority in design and construction.

Messrs. Lancaster & Tonge, Engineers, Pendleton, near Manchester, make a patent piston block, which is shown in Fig. 83. It is designed to keep the body of the block from the cylinder and the rings from being driven inwards, thus obviating the uneven wearing of the cylinder by the block bearing on it. The grooves A A are turned so that the rings rest on them, but being free to expand, keep the piston perfectly steam-tight.

Figs. 84 and 85 show the piston rings and one arrangement of springs, while Figs. 86 and 87 also show the piston rings and another arrangement of springs for the Lancaster piston.

These two arrangements of springs are also shown in Fig. 83 on opposite sides of the piston. A straight spiral spring, Fig. 85, is bent into a circle, Fig. 84, and then has two actions, *first*, the continuous effort of the straight spiral spring when forced into the circular rings, to recover that form; and *second*, the straight spiral spring being diametrically compressed when forced into the rings, makes a continuous effort to recover its original diameter. The lateral pressure is thus obtained by forcing the rings against the sides of the cylinder, and the vertical pressure by the rings being grooved to the circle of the spring, thus forcing the rings against the block and junk ring. It is self-adjusting, perfectly steam-tight, with a minimum of friction.

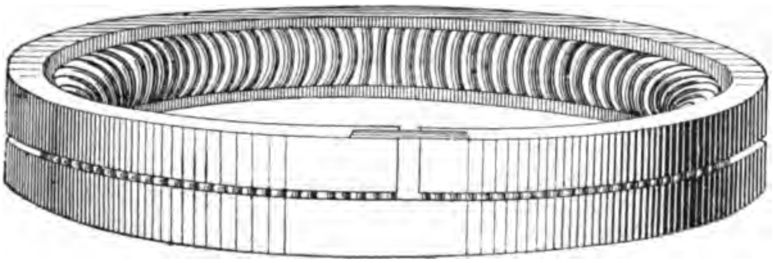


Fig. 84.

Fig. 85.

Lancaster Piston Rings and Spiral Spring.

Figs. 86 and 87 show the "Lancaster" serpent coil, which is a round section of tempered steel, and has therefore many advantages over those made from a flat section. The round coil against a flat surface cannot bind, but causes a revolving tendency in the packing rings. There is no difficulty in putting in; the rings being slightly rounded, the round coil slips in with the greatest ease.

Question 22.—Describe Joy's valve gear.

Joy's gear is an arrangement for working the slide valves of steam engines, and it is claimed for it that it is a more perfect appliance than the usual link motion worked by eccentrics on the main shaft, and can be adopted wherever the link gear is applied. In Joy's gear the necessary motion for the slide valve is obtained from the connecting rod.

The following sketch, Fig. 88, and description are taken from Percy's *Mechanical Engineering of Collieries*.

In the connecting rod from a point, A, preferably about the middle, motion is imparted to a vibrating link, B, constrained at its lower end to move vertically by the radius rod, C. From a point, D, on this vibrating link, horizontal motion is communicated to the lower end of the lever, E, from the upper end of which lever the motion is transmitted to the valve spindle by the link, G. The centre or fulcrum, F, of the lever, E, partakes also of the vertical movement of the connecting rod, to an equal extent to the amount of its vibration at the point, A; the centre, F, is for this purpose carried vertically in a slot, J, which is carried to a radius equal to the length of the link, G, connecting the lever, E, to the valve spindle. The slot itself is formed in a disc or sheave, K, which is concentric with the centre, F, of the lever, E, at the moment when that lever is in the position given by the piston being at either end of the cylinder. This disc is capable of being partially rotated on its centre, so as to incline the slot over to either side of the vertical, by means of the worm and hand wheel, M, thereby causing the curved path traversed by the centre, F, of the lever, E, to cross the vertical centre line, and diverge from it on either side at will. The forward or backward motion of the engine is governed by giving the slot this inclined position on one or other side of the vertical centre line; and the amount of expansion depends on the amount of the inclination, the exactly central or vertical position being "mid gear." In that position steam is admitted at each end of the stroke to the amount only of the lead; and this is done exactly equally on each side of the centre line, the amount of lead being constant for forward and backward motion, and for all degrees of expansion. Thus, when the crank is set at the end of the stroke either way, the centre, F, of the valve lever coincides with the centre of the slot, and therefore the slot may be moved over from forward to backward gear without affecting the valve at all. It will be seen at a glance, that, if the lower end, D, of the lever, E, were attached directly to the point, A, on the connecting rod, there would be imparted to the centre, F, of that lever an unequal vibration above and below the centre of the disc, K. The extent of inequality would be twice the versed sine of the arc described by the lower end, D, of the lever, E; and this would give an unequal port and unequal cut-off for the two ends of the stroke. But this error is corrected by attaching the lower end, D, of the lever, E, to the vibrating link, B; for while the point, A, on the connecting rod is performing a nearly true ellipse, the point, D, in the vibrating link, B, is moving in a figure like an ellipse bulged out at one side, and this irregularity is so set as to be equal in amount to the versed sine of the arc described by the lower end of the lever E, thus correcting the above error, and giving an equal travel to the centre, F, of the lever above and below the centre of the slot. At the same time, the

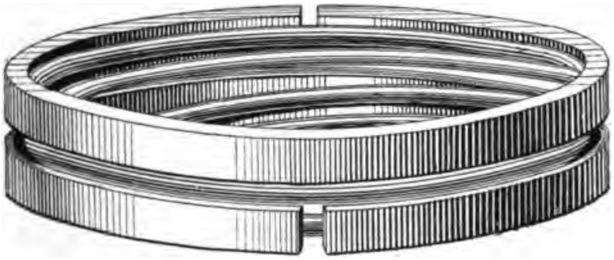


Fig. 86.

Fig. 87.

LANCASTER PISTON RINGS AND SERPENT COIL SPRING.

error introduced by the movement of the end of the valve link, G, is corrected by curving the slot, J, to a radius equal to the length of G.

These two errors may, however, be set against each other, and a compromise may be made by attaching the end of the lever, E, direct to the connecting rod at A, and allowing the centre, F, to slide in a straight slot. By a just balancing of the errors so produced, and by making the centre, F, of the lever, E, and the centre of the disc, K, to coincide at varying points in the travel of the former, a fair motion may be got for the forward gear of an overhead marine engine, giving a longer cut-off for the up stroke than for the down stroke. This is, of course, at the sacrifice of the backward gear, in which the reverse is the case, and the various degrees of exhaustion are between the two extreme conditions.

Referring again to the equalising of the traverse of the centre, F, of the lever, E, in the slot, J, the unequal traverse may be either under corrected or over corrected by shifting the point, D, in the vibrating link, B, nearer to or further from A; by this means a later point of cut-off may be given to either end of the cylinder at will, and the engine may thus have more steam admitted to one side of the piston than to the other if required. The same thing may be done for the lead.

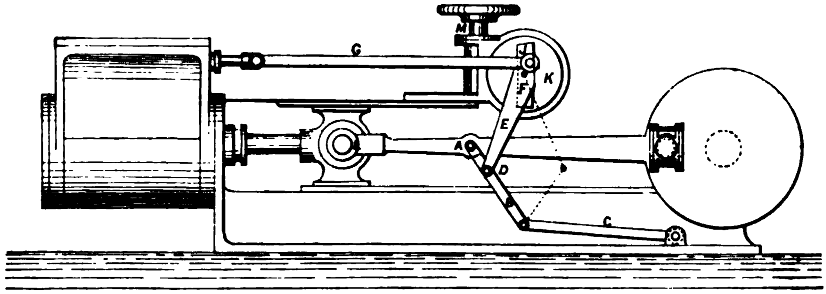


Fig. 88.—Joy's Valve Gear.

By altering the position of the crank, for which the lever centre, F, coincides with the centre of the slot, J, an increased or diminished lead may be given. The central positions and exact corrections are, however, in all cases standard and equal.

Question 23.—What is the object of a winding indicator, and where is it usually placed?

The object of any winding indicator is to show the position of the cages in their ascent and descent in the shaft, and it should be placed well within the view of the engineman in the engine-house whilst at his duty there. The indicator receives its motion from the crank shaft, to which, in the ordinary form of indicator, one end of a small chain is attached and round which it is wound. The other end is carried, by means of pulleys, overhead to a suitable place in the wall of the engine-house, and has there attached to it a well-defined pointer, which travels in a groove formed vertically in a frame resting against the wall. When the cages are at certain levels in the shaft, it is shown to the engine-man by the travelling pointer reaching certain marks formed in the indicator, in addition to which it is made to ring a bell each time the ascending cage approaches the surface, thereby giving warning to the engine-man to close the steam valve.

Another form of indicator, worked by gearing off the crank shaft, consists of a circular dial face with an index hand moving round it, which indicates the position of the cages by well-defined marks on the dial face.

Question 24.—Describe any system or systems of winding coal up shafts in which the use of the drum on the engine is dispensed with.

The use of the drum is dispensed with in Craven's improved winding gear, and also in the Kœpe system of winding. The following description of the former method appeared in the *Colliery Guardian* of June 9th, 1882:—

"Important improvements have lately been introduced by Mr. John Craven, of Wakefield, in the form and arrangement of winding gear for mines, for the invention of which letters patent have been granted. The improvements consist mainly in obviating the necessity for coiling ropes round the drums usually employed, and so removing the danger and expense arising from the great wear and frequent injury of the ropes by one coil chafing against the other. In order to effect these objects, the inventor employs a single winding rope, an upper set of grooved headgear pulleys, and a lower set of grooved winding pulleys, each consisting of two pulleys, and an intermediate grooved pulley between the two sets. The rope is attached at one end to one of the cages, passes over one of the headgear pulleys, under one of the winding pulleys (to which the motive power is applied), and back over the intermediate pulley, and then under the other winding pulley, and thence over the other headgear to the other cage, to which the end of the rope is attached. This arrangement is designed to give greater durability to the rope, and to obviate all tendency to slipping of the rope, as in proportion as the weight of the load is increased the adhesion of the rope is augmented. The bearings of the intermediate pulley may be carried in a movable frame, either inclined or otherwise, so as to admit of the pulley being adjusted as required, in order to maintain the rope taut.

"Fig. 89 represents in elevation and Fig. 90 in plan a winding gear constructed and arranged after the method above described. A set of two-grooved headgear pulleys is mounted in bearings in the headgear or framework, and another set of two-grooved winding pulleys is keyed on the crank shaft of the engines in the engine-house. The headgear pulleys are preferably set at an inclination inwards towards the winding pulleys in the engine-house, as shown in the plan. The intermediate grooved pulley is mounted in bearings carried by the framework. The single winding rope is attached at one end to one of the two cages and passes over one of the headgear pulleys, thence under one of the winding pulleys in the engine-house, and then back and over the intermediate pulley, and then under the other winding pulley in the engine-house, and to and over the other headgear pulley to the other cage, to which the end of the rope is attached. The winding engine is represented in the engraving, and its power is applied to the crank shaft of the engines, on which the winding pulleys are mounted, so as to drive the pulleys in the one or the other direction for raising the one cage and lowering the other by the one rope. The bearings of the intermediate pulley are in blocks which are capable of sliding upon guides carried by the framework. To these blocks are attached the rods represented in the elevation, the other ends of which are screwed and pass through lugs, and are provided with screw-nuts. By screwing up or slackening these nuts the bearings can be moved in one or the other direction to maintain the rope taut, or to slacken or tighten it as required. The following specific advantages are claimed for the invention:—No chafing of rope as in the ordinary system of drum, so that the ropes last longer; reduction of work for the engine to do in starting; reduction of strain upon the engine, &c., in stopping; speed, instead of being obtained by a large diameter of drum, is got from the engine running quickly; a smaller engine is required than with ordinary gear, owing to the comparative lightness of the winding pulley; the winding pulleys have only one groove each, and are of very small weight comparatively; saving in first cost, the engine-house being much narrower, and only one rope required instead of two; great adhesion, no slip occurring between the rope and the pulleys."

Mr. Percy, in his *Mechanical Engineering of Collieries*, describes the Kœpe system as follows:—

"The Kœpe system of colliery winding dispenses altogether with winding drums and substitutes a pulley. One winding rope answers for and is attached to both cages, instead of having a separate rope for each. This rope having a cage at each end simply passes about half round an ordinarily-constructed **V** pulley on the crank shaft. The balance rope under the cages is used in connection with the Kœpe system. The advantages claimed for the arrangement are manifold. First, massive winding drums are abolished, thus avoiding the enormous weight to start and stop each winding. Second, the pair of engines can be brought closer together, thus making the crank shaft shorter. Third, a smaller engine-house is required. Fourth, only one rope is actually used for winding. Fifth, the rope always coils round exactly the same diameter. Sixth, the rope always works in the same line. Seventh, the load is exactly uniform throughout the winding. Eighth, a smaller pair of engines are equal to the work. Objections have been taken to the arrangement that the rope is liable to slip upon the drum-

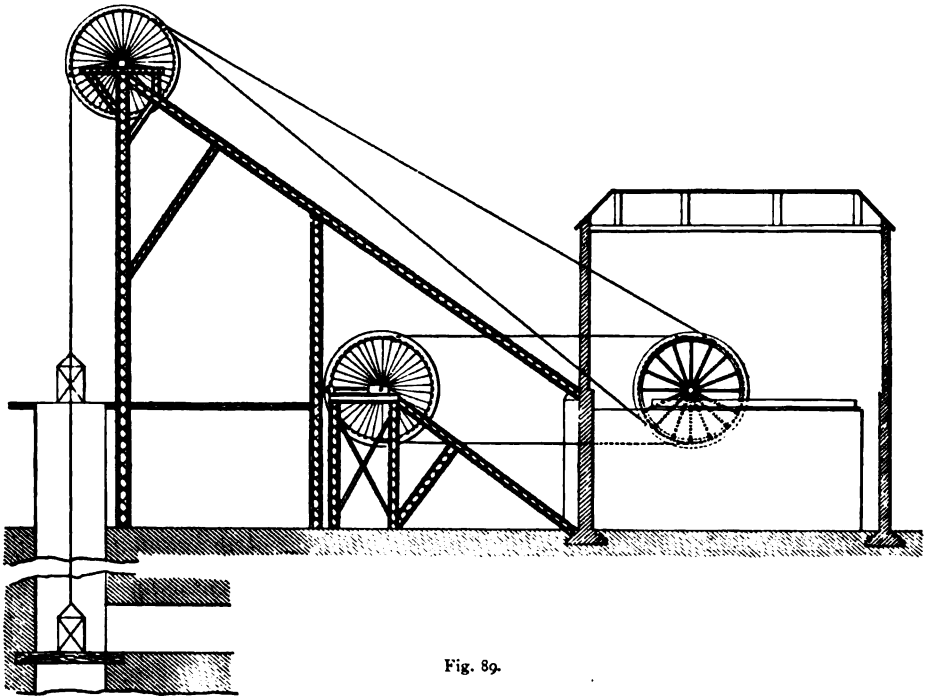


Fig. 89.

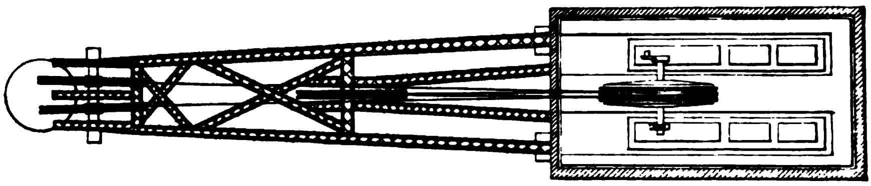


Fig. 90.

CRAVEN'S WINDING GEAR.

pulley; if the rope breaks the two cages fall down the pit; and the difficulty of re-capping. However, the Kœpe system has been fairly tried at the Bestwood Collieries and has been successful. There are two pits exactly the same depth and raising the same load in equal times, the steam for the engines coming from the same range of boilers. The Kœpe engines have 30-inch cylinders and 5½-foot stroke. The non-Kœpe engines have 36-inch cylinders and 6-foot stroke. Such a result is certainly encouraging for the advocates of the Kœpe system of colliery winding."

In the appendix to the same work, Mr. Percy, referring to the same subject, writes:—

"The Kœpe system as arranged at Bestwood is more elaborate than has been described. There is only one winding rope, but two lighter or safety ropes are also attached to both cages, and the object of these is that should the main rope break, the cages will not run down the pit. The manner of their operation is somewhat curious. The ends are attached to the respective cages, and the ropes pass over two pulleys in the head gear. The diameter of these pulleys is the distance from centre to centre of cages. So long as the main rope does not break, these safety ropes simply move about, but when the main rope breaks, the safety ropes have the weight thrown upon them, and pressing down upon their pulleys are brought to a stand, and prevent either cage from running back. This arrangement necessitates, in addition to the main rope, two safety ropes, and a balance rope under the cages equal in weight to the other three. It is really no part of the Kœpe system, introduces complication, and is not necessary. The Kœpe system proper consists as described, of one winding rope, one balance rope, and one drum-pulley."

Neither Mr. Craven's nor the Kœpe system appears to have become at all popular.

Question 25.—State any rules you know for finding the nominal horse-powers of engines.

Molesworth gives the following:—

"Nominal" horse-power means very little; the term is arbitrary and varying. The actual or indicated horse-power varying in stationary engines from 2½ to 3 times the nominal horse-power, and in marine engines it is sometimes 6 or 7 times the nominal horse-power. It is now becoming obsolete for marine engines.

V = mean velocity of piston in feet per minute.
D = diameter of cylinder in inches.
S = stroke of engine in feet.
H = horse-power of engine.

For high pressure engines.	*For condensing engines.*
$H = \frac{D^2 \sqrt[3]{S}}{15 \cdot 6}$	$H = \frac{D^2 \sqrt[3]{S}}{47}$
$D = \sqrt{\frac{15 \cdot 6 H}{\sqrt[3]{S}}}$	$D = \sqrt{\frac{47 H}{\sqrt[3]{S}}}$
$V = 128 \sqrt[3]{S}$	$V = 128 \sqrt[3]{S}$

Mr. Greenwell gives the following:—

For high pressure engines.	*For condensing engines.*
$H = \frac{D^2}{13 \cdot 6}$	$H = \frac{D^2}{30}$

CHAPTER VI.

SURFACE ARRANGEMENTS *(Continued)*.

STEAM BOILERS AND THEIR FITTINGS.

Ordinary Forms of Colliery Boilers, "Egg-ended," Cornish, and Lancashire—General Construction and Flue Arrangements—Galloway Tubes for Cornish and Lancashire Boilers—Details of Construction—Riveting, Punching, and Drilling—Caulking—Welded Shell Joints—Attachment of Flat Ends—Expansion Joints for Internal Tubes—Relative Strengths of Different Riveted Joints—Diagonal Seams—Complete Shell Rings—Means of Strengthening Apertures cut in the Shell—Seating a Lancashire Boiler—Faulty Methods of Seating—Seating Blocks and Crown Tiles—Means of Preventing Radiation of Heat—Position on the surface in which to place Boilers—Boiler Fittings—Mechanical Stokers—Hydraulic Test—Steam-pipe Connection—Steam-pipe Expansion Joints—Heating the Feed-water before it enters the Boilers—Feed-pumps and Injectors—Chimneys and Chimney Flues—Galloway Breeches-Flued Boilers—Babcock and Wilcox Water-tube Boilers—Arnold's Boiler—Vertical Boilers—Safeguards against Explosions—Grooving, &c.—Necessity for Care in Tending Boilers—Water Impurities—Analysing and Purifying Feed-water by the Addition of Chemicals—Archbutt & Deeley's Method of Purification—The Hotchkiss Boiler-cleaner—Seale's Patent Water Purifier—Sanderson's Patent Feed-water Purifier and Heater—Periodical Examination of Boilers by Experts—Causes of Explosion—Dangerous Practices when Cleaning Boilers—Warming Surface Buildings by Steam—Rules Relating to Boilers—Separators, Steam-traps, and Burnam's Steam Loop—Feed-water Heaters and Economisers—Warning Whistles, &c.

Boilers are necessary for the generation of steam to supply the engines with their motive power. They have been made of various forms at different times, but those now mostly used for colliery purposes are the common cylindrical boiler, the Cornish, and the Lancashire. The efficiency of boilers is expressed by their evaporative power of water per lb. of coal consumed on the fire-grate. Common boilers evaporate about 7 or 8 lbs. of water per lb. of coal consumed, and Cornish and Lancashire boilers from 10 to 12 lbs.

The plain or egg-ended boiler, shown in Fig. 91, is fired externally, the fire-grate being situated underneath the boiler, but not shown on the sketch. Boilers of this type are made in lengths varying from 20 to 35 feet, 30 feet being a common length. There is little or no gain by extending the length beyond 35 feet. The construction of this boiler is very simple; it does not require any stays whatever, either at the hemispherical ends or elsewhere. The disadvantages attending its use arise from the fact that it is a large coal consumer, and that all sediment collected in the bottom of the boiler is immediately over the greatest heat, and there is, consequently, a great risk of burned plates. Again, owing to its small amount of heating surface, steam cannot be so rapidly generated in it, nor obtained from it, at the high pressure so frequently required at collieries. Formerly, the products of combustion were conveyed round the boiler by what was termed a "wheel-flue." Having passed under the boiler to the end, the products of combustion entered a side flue, thence traversing along one side of the boiler, across the front end, back along the other side, continuing on to the chimney. The present practice is to dispense with this, and the boilers have what are known as "flash-flues," that is, the products of combustion pass under the boilers, and direct to the chimney.

A Cornish boiler has two flat ends and one internal flue passing from end to end. The object of the internal flue is to give a greater heating surface than that of the egg-ended boiler.

Fig. 92 shows a sketch of this boiler.

The products of combustion pass from the fire-grate through the internal tube to the back end of the boiler, then, dividing, they return to the front end along the two side-flues. Here they pass down to the bottom flue, and, re-uniting, pass underneath the boiler to the chimney.

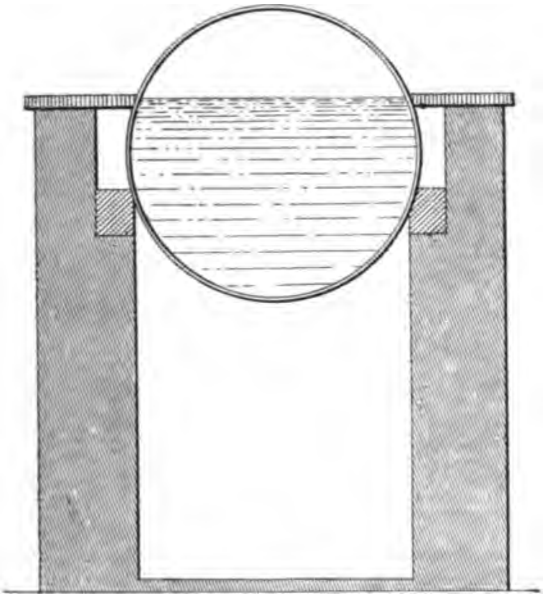


Fig. 91.—Common Cylindrical Boiler.

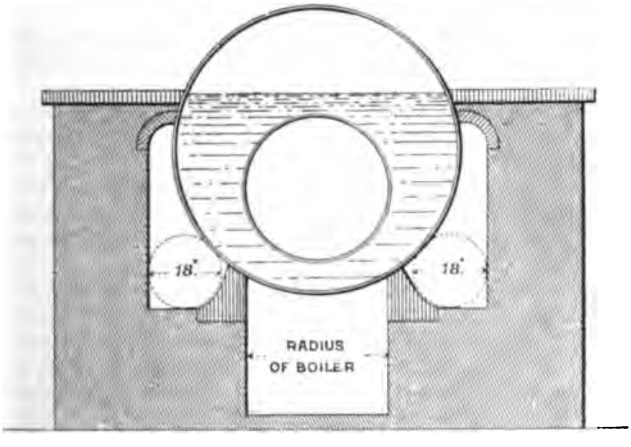


Fig. 92.—Cornish Boiler.

Where the flues are traversed thus, the gases are reduced in temperature before coming in contact with the bottom of the boiler, where all sediment collects, and all danger of burned plates on the under side of the boiler is avoided. Where the water used in the boilers is pure, or with boilers of such a length as to ensure the gases being considerably cooled before leaving the furnace flue, it is usual for the products of combustion to be brought first along the flue under the boiler to the fire end, where they divide and pass upwards into the two side-flues, which they traverse to the end of the boilers, and thence to the chimney.

It has been estimated that the temperature of the gases escaping from boilers of the Cornish and Lancashire type having a moderate draught is about 600° F.

The egg-ended and Cornish boilers are usually made 6 feet in diameter.

The Lancashire boiler, like the Cornish, has two flat ends, but it has two internal flues running from end to end, and is usually from 7 feet to 8 feet in diameter.

The Cornish and Lancashire are internally fired, a furnace being placed at the one end of the internal flue of the Cornish, and two furnaces at the one end in the two flues of the Lancashire.

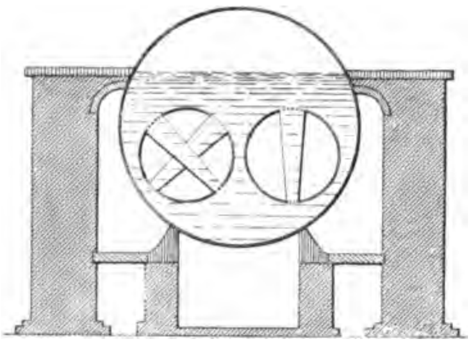


Fig. 93.—Lancashire Boiler with Galloway Tubes.

Fig. 93 is a sketch of a Lancashire boiler fitted with Galloway tubes.

The Cornish boiler is suitable for small powers. If boilers of great power were made of the Cornish type, an excessively large furnace flue would be required, in order to give sufficient grate surface. The length of grate is regulated by the reach of the fireman, and in practice is made from 5 to 7 feet. A flue of large diameter is weak to resist collapse, unless very thick plates are used in its construction, and thick plates are undesirable in steam boilers.

The Lancashire boiler is designed to meet the requirements of large power, the two flues of moderate size serving the same purpose as one large one. The arrangement of the external flues is the same as that described for the Cornish boiler.

In both the Cornish and Lancashire boilers the effective heating surface is increased, and a better circulation of water in the boilers promoted by the use of "Galloway" tubes. A further advantage of the "Galloway" tubes is the increased strength given to the flue in which they are placed. They are conical in shape, and pass through the central flues beyond the fire-bridge, some being placed vertically and others diagonally, those placed diagonally being at right angles to each other.

In a line of tubes a vertical right-hand diagonal and left-hand diagonal are placed in succession. The conical tubes are about 5 inches larger in diameter at the higher than at the lower end. Galloway tubes may have Gamgee spouting cowls applied to them. A cowl consists of a dome extending from the top of the tubes to about four inches above the water-line. The dome contains an inverted cone, the object of which is to cause the ascending hot currents of water to pass from the top of the Galloway tube into a gradually contracted space. The reduced area increases their velocity and causes them to be ejected into the steam space above the water-line. The colder water falls to the bottom of the boiler, and so the whole body of water is kept in more active circulation.

In constructing these boilers, iron or mild steel plates are used, the joints in the shells of which are riveted together by either a single or double row of rivets.

The holes formed in the plates for the rivets are either drilled or punched, and the rivets may be either oval, or, what is much more common, circular in form, and the riveting may be done by hand or mechanical power. The plates should all be of good quality, to stand the various processes of flanging, welding, punching, &c., to which they are subjected. The plates forming the furnace flues are more severely taxed by variations of temperature than any others in the boiler, those of the furnace-crown especially being subjected alternately to the fierce hot flames from the fire and the currents of cold air which rush into the furnace every time the firing-door is opened.

Steel is unquestionably a better material for the construction of boilers than iron, but great care must be paid to its special properties, and plates of a mild nature, possessing moderately high tensile strength but great ductility, should always be used.

Boiler makers test the plates used for boiler making in many ways. One test for ductility is to bend back the flange of a plate when cold. Another test for steel plates is to cut strips, which are heated red hot, then plunged into cold water, after which they are bent cold to a certain defined radius. The ductility is fully proved if the strip shows no sign of fracture.

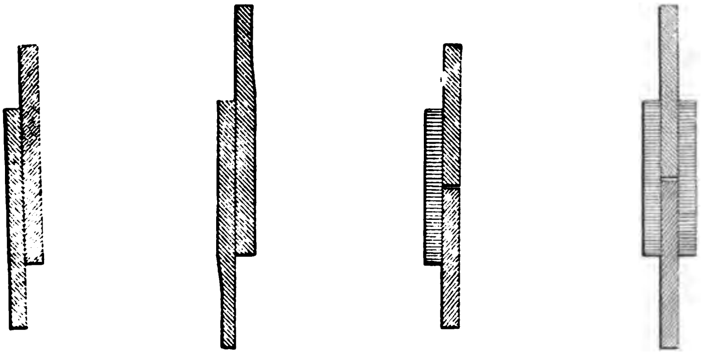


Fig. 94. Fig. 95. Fig. 96. Fig. 97.

Boiler-plate Joints.

Thinner plates of steel may be used than would be necessary with iron plates in boilers worked at a given pressure.

The joints of the flues are formed either by riveting or welding.

Riveted joints are either lap-joints or butt-joints.

In the ordinary lap-joint the edge of one plate projects, or laps over the edge of the next plate, as shown in Fig. 94. Sometimes the plates are slightly thickened at the portion forming the joint, as shown in Fig. 95.

In the butt-joint the two plates are placed level and uniform, edge to edge with each other, and the connection made by either a single cover-plate, as shown in Fig. 96, or by a double cover-plate, shown in Fig. 97. Rivets are then driven through the boiler and cover-plates.

Where lap-joints are made the plates are cut to overlap one another slightly, and then bent to the proper curvature.

All joints in the shell of a boiler are single, double, or triple riveted, but triple rivets are only used in the seams of marine boilers of large diameter, and working at high pressure.

The simplest form of joint is a single riveted lap-joint, but it is also the least efficient. The rivets are placed about two inches apart centre to centre, and are about $\frac{5}{8}$ths of an inch in diameter.

Double riveting is done in one of two ways. Where the rivets of one row are placed opposite to the spaces between the rivets of the other row, it is termed *sig-*

zag riveting, as shown in Fig. 98. Where the two rows of rivets are placed opposite each other, it is called *chain* riveting, as shown in Fig. 99. Zig-zag riveting requires less lap than chain riveting, besides making a tighter joint, but the plates are not so strong. The chain-riveted makes the stronger joint, and is coming more into use.

There are some advantages in punching the holes in the plates, and some in drilling them. The strongest argument against punching plates lies in the fact that the plates are punched when flat, and afterwards bent to shape, whereas the drilling is done after the plates have been bent.

A plate with punched holes receives damage along the row of holes in bending. To some extent, also, punching injures the texture of the metal immediately surrounding the hole. Again, where the punching is carelessly performed the holes in the plates do not correspond. What is called *drifting* is then resorted to. It consists of bringing the two plates into good alignment for receiving the rivets by either a pointed or barrelled drift. Where the contiguous holes of two plates are so much out of correspondence as to be nearly blind, the pointed drift is used.

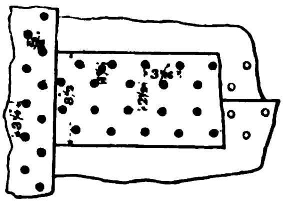


Fig. 98.—Zig-zag Riveting.

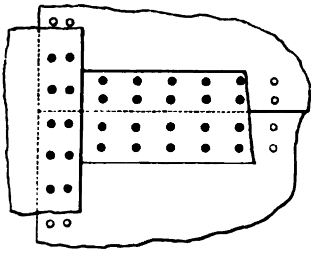


Fig. 99.—Chain Riveting.

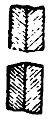


Fig. 100.—Punched Holes.

The practice of drifting is most objectionable, especially with the pointed drift, as the plates are injured by it. Where the holes do not quite coincide they should be drilled true. It is quite true that the plates when drilled separately present the same difficulty with respect to a want of agreement between the rows of holes.

Drilling machines are now in use whereby the plates are drilled when fixed in position, thus ensuring an absolute coincidence of the holes in the different plates.

The holes formed by drilling are of a uniform size throughout the plates, but punched holes are slightly tapered, as shown in Fig. 100. When the plates are placed together, the small ends of the holes are placed inside the joint, and the larger ends outside. There is in this a decided advantage in the punched holes, as when the rivets are driven in they hold the plates more firmly, and make a tighter joint than with the parallel holes of drilled plates.

The edges of drilled holes are sharp, and exercise a cutting action on the rivet; if slightly counter-sunk the strength of the rivets is increased, but this adds to the expense. No cutting action takes place at the outer edge of a punched hole.

From experiment it seems well established that where the plates are punched the rivets are stronger, but the plates are weakened to a greater degree; and altogether joints made with drilled holes are rather stronger than joints made with punched holes.

The rivets should be of the best Lowmoor iron or mild steel, and are heated in a common portable hearth before being hammered down. Greater care is required with steel rivets than with iron, as steel is a material which suffers even greater injury than iron from being over-heated. As the rivets cool they contract and draw the plates closer together. Formerly, all the joints were riveted by hand, but now wherever a machine can be applied it is used to do the riveting. Besides doing the work more quickly, good machine riveting is superior in strength to hand riveting. The hydraulic riveting machine is mostly used for the purpose. In machine riveting the pressure on the whole body of the rivet is more gradual than that resulting from the sharp, sudden blows of a hammer in hand riveting. This gradual pressure, resulting from powerful squeezes, forces the rivet into the hole, making its filling a matter of certainty before forming a head at all, and the joint is, therefore, more secure. The act of riveting with a mechanical riveter is momentary, and the point of the rivet is not rendered brittle by being repeatedly hammered when at a low heat or cold. Care should be taken to have the plates drawn closely together before riveting, or the compression of the body of the rivet into the hole may cause a slight shoulder to be formed between the plates, which will prevent the closing of the joint.

When the riveting of the boiler is completed, the joints should all be carefully caulked, so that they may be absolutely steam- and water-tight.

The usual practice is what is called split caulking. Fig. 101 shows this method as applied to a lap-joint, and Fig. 102 as applied to a butt-joint.

By means of a tool something like a chisel a score or split is cut as shown in the figures. This brings the extreme edge of the lap into close contact for about $\frac{1}{8}$th of an inch, but at the same time is objectionable, as in the case of lap-joints it is liable to open the plates between the extreme edge and the point where they are held tightly together by the rivets. For this reason, many makers have now largely given up split caulking.

The best practice consists in planing the edges with a slight bevel, and then by means of a proper caulking tool the surfaces are driven into close contact without injuring the plates.

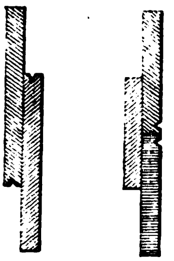


Fig. 101. Fig. 102.

SPLIT CAULKING.

A boiler shell consists of rings formed of plates from three to four feet six inches wide, rolled with the grain running circumferentially. Each ring is usually composed of two or three plates. Steel plates are now rolled large enough to admit of one plate forming an entire ring, so that there is only one longitudinal seam in a boiler made of such plates.

Within the last few years boilers have been made with welded joints in the shells. If the soundness of these joints were assured, a boiler made of complete rings would possess advantages over one built in the ordinary manner, in having fewer joints, which, besides being the weakest parts of a boiler, are the places where leakage most frequently occurs, and external corrosion arising therefrom. The soundness of welded joints is, however, still uncertain, and for this reason boilers built with these ring plates have not become popular.

The plates in the rings are connected to each other by lap- or butt-joints, as are also the rings to each other.

The circular seams form a continuous line round the boiler, but the horizontal or longitudinal seams are not continuous, the joint in each ring being intermediate to joints in the adjoining ring.

The flat end plates are each in one piece, the portion to receive the internal flues being bored out of each plate, and are connected to the shell in different ways.

Fig. 103 shows the usual method of attaching the front plate to the shell and to the internal tube, and Fig. 104 the method of joining the back plate to the outer shell and internal tube. In Fig. 103 a ring of angle iron is placed outside the shell, and riveted to the shell and to the front plate. The internal tube is usually similarly dealt with at both the front and back ends, having a ring of angle iron placed outside it at either end, the angle iron being riveted to the tube and to the end plate as shown in Figs. 103 and 104.

Occasionally both the front and back plates are attached to the shell by inside angle iron, but usually the back end plate is flanged and then riveted to the shell plates by an ordinary lap-joint, as shown in Fig. 104.

Sometimes the internal flue is attached to the end plates by flanging the end plates inwards or outwards.

The longitudinal seams of internal boiler tubes are either welded or butt-jointed. The reason why no lap-joints are made in the internal flue is that any departure from a truly circular section gives less resistance to collapse. The cylindrical shell of a lap-jointed boiler is not perfect in form, but the *internal* pressure to which it is subjected has a tendency to rectify the defect and to bring the shell to the form of an exact cylinder. The effect of *external* pressure on an imperfectly cylindrical tube, however, increases the defect and causes a greater departure from the true circular section. By welding the longitudinal seams or by butt-jointing them, the truly cylindrical form is approached as nearly as practice allows.

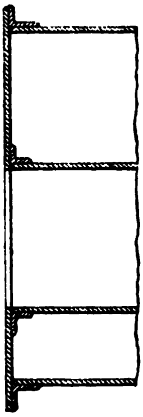


Fig. 103.—ATTACHMENT OF FRONT-END PLATE.

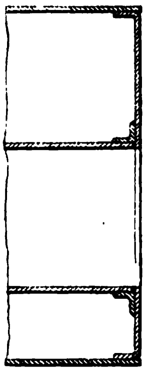


Fig. 104.—ATTACHMENT OF BACK-END PLATE.

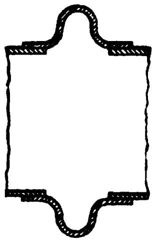


Fig. 105.—BOWLING HOOP EXPANSION JOINT.

In addition, the internal tubes require strengthening and also to have some provision for expansion and contraction, as they are subjected to great heat and sudden admissions of cold air. It is customary now to secure the different rings forming the flue to each other by an expansion joint. At first a ring of T iron was used for strengthening the tubes. It was riveted round the joints of each ring of plates, and although found to give sufficient strength it held the flue too rigidly and did not allow free expansion and contraction to take place.

Fig. 105 shows the expansion or bowling hoop, which has been much used for flue joints. It is weldless, and can be made in iron or steel. It is as strong as the T iron ring, with the advantage of allowing free expansion of the tube. The objection to it is that it exposes two rows of rivets and a double thickness of plates to the intense heat of the furnace, and these are therefore liable to be burned.

Another form of expansion joint is known as Adamson's flanged seam, and is shown in Fig. 106.

In it the ends of the flue plates are flanged and connected by means of rivets with a ring placed between. The object of the ring is to give a caulking edge on each side of the lap. This joint is very elastic, and allows free expansion and contraction to take place.

By its use no double thicknesses of plate and no rivets are exposed to the action of the fire. All plates which require flanging must, however, be of excellent quality, and even then, if not skilfully done, the joint gives a considerable amount of trouble.

Fig. 107 shows another method of strengthening flues and of allowing for their expansion and contraction, viz. Paxman's flue joint. It consists of welded rings of iron or steel, which are rolled out accurately in a machine to the shape shown on sketch, the connection being made by a simple lap-joint. This joint allows for expansion. The rivet heads and double thicknesses of plate, although not removed from the action of the fire, are out of immediate contact with it.

Foxe's corrugated furnace flues are stronger than the plain flue fitted with any of the strengthening rings mentioned, and yet their shape allows every facility for expansion, whilst giving greater heating surface than the ordinary flue. An objection to their use, however, lies in the facility offered for sediment and salt incrustation to gather and form in the hollows of the corrugations at the top of the flue, whilst at the bottom the corresponding hollows under the fire are filled with dead ashes, not easy to remove.

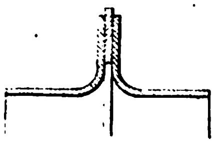


Fig. 106.—Adamson's Flanged Seam.

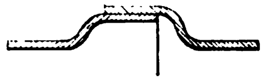


Fig. 107.—Paxman's Flue Joint.

The method of fitting Galloway tubes in the flues consists in making a hole in the upper side of the flue of such a size as to allow the flange of the small end of the tube to be passed through. The hole cut at the bottom of the flue is the net size of the bottom of the Galloway tube. A row of rivets is then driven all round the flanges through the flue. The lower flange of the Galloway tube is inside and the upper one outside of the flue, as shown at H, in Fig. 131. Sometimes the Galloway tubes are welded into the flues, and this is very effectual in preventing leakages at the joints. An objection to this plan is the large hole which must afterwards be cut in the flue, if it is found necessary to remove the tube, as the welded part must be cut away with it.

As shown in Fig. 131, the last two rings of an internal tube in a Lancashire boiler are rather smaller than the others, the last but one being tapered in form, the diameter at the small end being some six inches less than at the larger end.

The egg-ended boiler, as before stated, requires no stays whatever, owing to its shape at the ends. The pressure acting on flat surfaces, however, causes those surfaces to bulge out. All flat surfaces in boilers therefore require to be stayed.

In the Cornish and Lancashire boilers the only flat surfaces are the ends, and these require to be stayed with gusset stays, or with longitudinal stays passing from end to end of the boiler, or it may be with both.

Gusset stays are usually made of a single plate of iron, and this is fixed to the end plate and to the shell by means of angle irons on each side of the strengthening plate, as shown in Fig. 131 at E. The rows of rivets at the front end gusset stays may also be seen in Fig. 122.

There are usually five gusset stays over the flue and two under at each end.

Longitudinal stays are rods of iron or steel secured at the ends by nuts and washers. One is shown at R in Fig. 131. When these stays exceed 20 feet in length they have a tendency to droop in the centre, and must, therefore, be supported there by small brackets riveted to the shell.

In staying the ends, the object desired is to strengthen the plates and yet preserve a certain amount of elasticity. If made absolutely rigid the flues would not have sufficient freedom to expand.

The following table, taken from Sir John Anderson's *Strength of Materials*, enables us to compare the relative strengths of the different forms of riveted joint.

The strength of the solid plate is taken at 100.

Riveted Joints.

Description of Joint.	Riveting.	Rivet Holes.	Percentage of Strength of the Solid Plate possessed by the Joint.
Lap	Single	Punched Drilled	55 62
Lap	Double	Punched Drilled	69 75
Butt, 1 Cover	Single	Punched Drilled	55 62
Butt, 1 Cover	Double	Punched Drilled	69 75
Butt, 2 Covers	Single	Punched Drilled	57 67
Butt, 2 Covers	Double	Punched Drilled	72 79

From this table it appears that the single-riveted lap-joint is the weakest, that butt-joints with one cover are no stronger than lap-joints, but with two covers the percentage of plate strength is a little more than lap-joints.

In the construction of boiler shells it has been proposed to substitute diagonal seams for the longitudinal, but an objection to this is the waste of plates resulting from cutting the ends to the chosen angle. Doubtless, a cylindrical boiler constructed with seams in this manner would be stronger than with ordinary longitudinal seams. A cylinder is twice as strong transversely as longitudinally.

In any boiler shell having the circular and longitudinal seams similarly riveted the former has twice the resisting power of the latter. A single-riveted joint, unless badly designed and clumsily made, is rather more than half as strong as the solid plate. A single-riveted ring seam is, therefore, calculated to resist pressure equally as well as the solid plate of the same thickness in the longitudinal section. In whatever way the longitudinal seams are riveted they cannot be made as strong as the solid plate, and consequently the strength of any boiler to resist internal pressure is limited by the strength of its longitudinal seams. If, instead of forming any longitudinal seams at all, the plates are cut so as to form an angle between the circular and the longitudinal the strength of the joint must be increased, the amount of increase being greater the more nearly the circular is approached. It is true that the effect of a diagonal seam replacing a straight longitudinal one would, while adding to the resisting power longitudinally, also weaken it transversely.

As the ring seam is so much stronger than the straight longitudinal, however, a boiler formed with diagonal seams would be, on the whole, stronger to resist internal pressure, and taken altogether the alteration would be a gain.

Boiler-plates are now made of such a size as to admit of one forming a ring, and possibly the amount of waste in cutting a diagonal seam on it would not be of so much consequence as the increased gain in strength obtained by it.

The diagonal seams would be formed by double riveting and be placed above the water-line, thus ensuring a considerable increase in the total strength of the boiler, if the angle formed between the ring and longitudinal seam be well chosen.

Some efforts have been made, too, to roll short, complete rings by special machinery, for boilers, but there appears to be considerable difficulty in practically carrying out this process.

Wherever an opening is cut in a boiler for a steam-pipe or any other fitting, an internal block is riveted round the opening to compensate, in some measure, for the diminished strength.

The man-hole is the largest opening in the boiler, and as this is usually elliptical in form, 16 inches by 12 inches, it necessitates cutting a large aperture in the boiler crown. At one time these man-holes were left without any strengthening collar, notwithstanding the danger arising from the diminished loss in strength of the plate. Afterwards it became the practice to rivet a broad strip of wrought-iron round the hole on the outside. The lid or cover consisted of a flat plate rolled to the curvature of the boiler. It was fitted to the inside of the shell and held up by bridge bars, bolts and nuts. With this arrangement it was very difficult to get a good fitting joint, as a great strain was brought upon the shell-crown by screwing the cover-bolts so tightly, the effect of which, increased by the steam pressure inside, was liable to bulge the cover sufficiently to fracture the edge of the man-hole. Rings of cast-iron were then tried, made to fit boilers of any radius, and these answered very well for moderate pressures. With increased pressures, however, these man-holes began to fail, and where the pressure is great the man-holes are now made of wrought-iron or steel. Besides the rigidity of cast-iron, which does not permit the man-hole to stretch as the more elastic wrought-iron or steel will do, cast-iron man-hole mouth-pieces may contain hidden faults such as blow-holes in the body of the metal. The present practice is to strengthen the aperture by riveting a broad strip of wrought-iron or steel round the inside of the hole, and fitting above it a short cast-iron, or, if for high-pressures, a wrought-iron or steel-flanged neck. This neck or mouth-piece is made flat on the top and has a slightly curved cover, secured to the neck by bolts and nuts passing through the flanges.

Boilers should be placed absolutely level in their seatings, or be very slightly inclined to the front. Usually they are fixed with an inclination of 1 or 2 inches in the length, the fire end being the lower, so as to have more water there than at the other end when at work, and to ensure the boiler being thoroughly emptied through the blow-off cock when desired.

This plan allows for any slight settlement which may afterwards take place in the boiler, causing the back end to drop. If the boiler be set exactly level and the back end afterwards settle down slightly, all the water would not flow out of the boiler through the blow-off cock when desired.

The objects to be aimed at in setting an ordinary Cornish or Lancashire boiler are:—

1. The circulation of the gases in contact with the shell, so that the heat of the fuel is utilised to its full extent.*

2. The distribution of these gases in such manner as to equalize the temperature of the shell as far as is practicable.

3. The arrangement of flues to give free access for a person to make an inspection of every part of the outside of the boiler.

4. The prevention of loss of heat by radiation from the surfaces not exposed to the hot gases.

Drawings Nos. 117—121 show the details of seating for an ordinary Lancashire boiler designed to meet these requirements. The products of combustion pass from the internal flues, as shown by the arrows, under the bottom of the boiler, and return by the side-flues to the stack. In constructing the side-flues great care must be taken that only the portion of boiler-plate which is below the level of the water inside the boiler is exposed to the direct action of the flames or heated gases. If the side-flues are so constructed as to allow the heated gases to come in contact with the plates above the water-line, there is danger of the plates being burned.

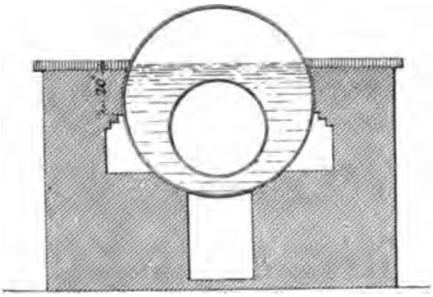


Fig. 108.—FAULTILY SEATED CORNISH BOILER.

The external flues are built of ordinary bricks, but it is essential that all parts of the seating exposed to the action of the fire should be made of good fire-brick, and the external flues are always lined inside with a thickness of 4½ inches of fire-brick. In a range of boilers, the walls must be sufficiently thick between them to allow of one boiler being removed without injury to another.

It is of great importance to reduce as much as possible the actual contact of the brick-work with the bottom or sides of the boiler, as these points of contact are the most liable to deterioration. In some modes of setting there is no contact at all below the water-line, the boilers being suspended.

Care must also be taken in fixing boilers in the saddles on which they rest that some arrangement is made to allow for their expansion and contraction. For this purpose rollers are sometimes placed in the saddles.

There are a number of ways in which Cornish and Lancashire boilers have been at different times improperly seated.† Fig. 108 is a method of faultily seating a Cornish boiler, while Fig. 92 shows the most approved method of seating a similarly constructed boiler. In it properly designed seating blocks are used for the boiler to rest on, the point of contact between the two being over an extremely small area. To ascertain the supporting width in transverse section for the seating blocks where in contact with the boiler, allow one inch per foot diameter of the boiler; thus the seating blocks for a boiler 6 feet in diameter should be 3 inches on each side where in contact with the boiler, and for a boiler 7 feet in diameter 3½ inches on each side. The crowns of the side flues should be built with proper quarter-circle tiles, and these as well as the seating blocks should be hard and of a non-porous nature, so that they may not absorb moisture and secrete corrosion. Above the crowns the brickwork should not exceed 6 or 9 inches.

Fig. 108 shows the boiler seated without either seating blocks or crown tiles, and the flue space formed by the brickwork is limited in extent and faulty in design. The seatings are built of ordinary fire-brickwork, are a foot broad, while the same kind of work is in contact with the boiler above the crowns of the side flues for 18 inches on each side.

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* See Report of the Chief Engineer to the Boiler Insurance and Steam Power Co., Manchester, June, 1881.

† For this comparison we are indebted to very able articles in the columns of the *Practical Engineer.*

The effect of so much brickwork in contact with the boiler renders the latter more difficult of examination, and being porous, it is calculated to absorb moisture and keep out of sight corrosion at those portions of the boiler shell covered by the brickwork.

Besides these objections to such masses of brickwork, there is the further one that the seatings formed in Fig. 108 deprive the boiler of a considerable amount of heating surface. In a boiler 28 feet long, the heating surface lost at the seatings would be (1 ft.+1 ft.)−(3″+3″)×28=42 square feet as compared with Fig. 92. Allowing 10 square feet of heating surface per nominal horse-power the deprivation of heating surface results in a loss of 4·2 nominal horse-power in a boiler of the length given as compared with Fig. 92, and would be still more in one of greater length.

Again, the side flues of Fig. 92 being of ample size and suitable shape, offer no impediment to a full and searching examination when the boiler is not at work, and at other times ensures an efficient draught. The deposits of soot formed on the sides of the shell will fall to the floor, which being level or but slightly below the level of the lowest point of the boiler shell, allows of a certain amount of accumulation without greatly interfering with the draught, whereas in Fig. 108 thick deposits of soot still further reduce the capacity of the flues, and necessitate more frequent cleaning out than in Fig. 92.

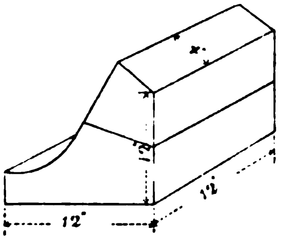

Fig. 109.—SEATING BLOCK.

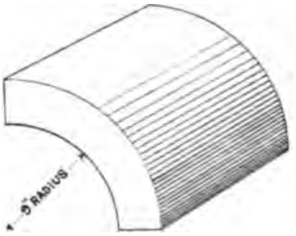

Fig. 110.—CLOSING-IN TILE.

To ensure a sufficient depth in the side flues further attention must be given to the form of seating block used. It is not sufficient that this presents a small bearing surface for the boiler to rest on, for if made shallow and used for boilers of comparatively small diameter, the side flues formed as a result will be restricted in area. The seating blocks then must be of sufficient depth, as well as made to give narrow bearing surfaces for the boiler to rest on.

Fig. 109 shows a seating block suitable for a large boiler. Its inner vertical side next the boiler is 12 inches deep, by 12 inches wide at the bottom, and 3 inches thick in its outer edge, presenting a bearing surface of 4 inches at the portion in contact with the boiler plates. Opposite the ring seams, these blocks should be made in two parts as shown in Fig. 109, and the upper part left loose. This will facilitate examination for leakages at the seams of that portion of the shell resting on the blocks, as the loose portions of the blocks may be taken out, and after the examination is completed, re-placed.

The use of lime-mortar in any brickwork or blocks, actually in contact with the plates must be carefully avoided. The effect of using it in those positions is to corrode the iron, and therefore ground fire-clay is substituted to cement the fire-bricks and blocks in setting steam boilers. Fire-clay of good quality, such as that of Stourbridge, stands heat and sets well, and it has no injurious effect on boiler plates.

A suitable quarter-circle closing-in tile for the crowns of side flues is shown in Fig. 110. It has a radius of 9 inches, so that the width of the side flue at the top should be 9 inches, whilst at the bottom it should be constructed of such size as would enable a circle of 9 inches radius to be struck so that its circumference may touch the floor and side of the side flue, and also the shell of the boiler, as shown by dotted lines in Fig. 92.

Many other instances of improperly seated boilers besides that shown in Fig. 108 may be given.

Fig. 111 shows a case where, owing to limited space, a Cornish boiler was set without side flues.* Here the shell is completely covered by brickwork, allowing of no opportunity of examination, and inducing corrosion to a dangerous extent without possibility of its being discovered.

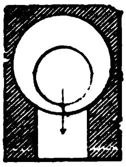

Fig. 111.

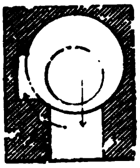

Fig. 112.

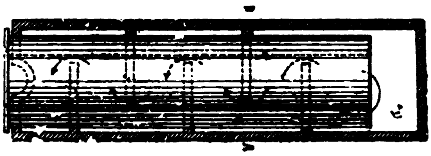

Fig. 113.

Improperly Seated Boilers.

Fig. 112 shows a Cornish boiler improperly seated with one small side flue and a bottom flue.

The effect of this on a long boiler is to cause springing and leakage of the seams from the heat being applied to one side of the boiler only.

Figs. 113 and 114 show a less objectionable way of seating a Cornish boiler in places where sufficient room cannot be obtained for side flues, but it is a wrong mode of seating. Here the boiler is carried on cross walls, and the products of combustion coursed as shown by the arrows.

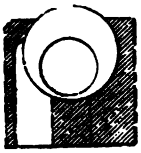

Fig. 114.

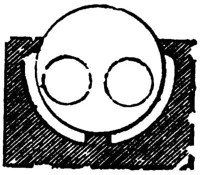

Fig. 115.

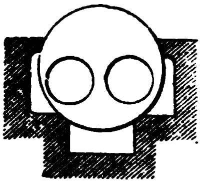

Fig. 116.

Improperly Seated Boilers.

Figs. 115 and 116 show faultily seated Lancashire boilers where there has been sufficient room for the side flues, but no regard has been paid to giving access outside the shell.

Errors of this kind, shown in Figs. 115 and 116, may be corrected by removing parts of the brickwork, so as to enlarge the flues.

To prevent radiation of heat from the upper part of the shell, some kind of external protection must be provided. For this purpose boilers are too often entirely covered with brickwork, but this practice is very objectionable. It renders the upper portion of the shell inaccessible for examination, and should a slight leakage take place at a joint of one of the fittings, most probably it would not be detected until the moisture resulting from it had spread over a considerable surface of the shell, owing to the absorbent porous brickwork.

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* See Report of the Chief Engineer to the Boiler Insurance and Steam Power Co., Manchester, June, 1881.

Scale. 6 Feet to 1 Inch

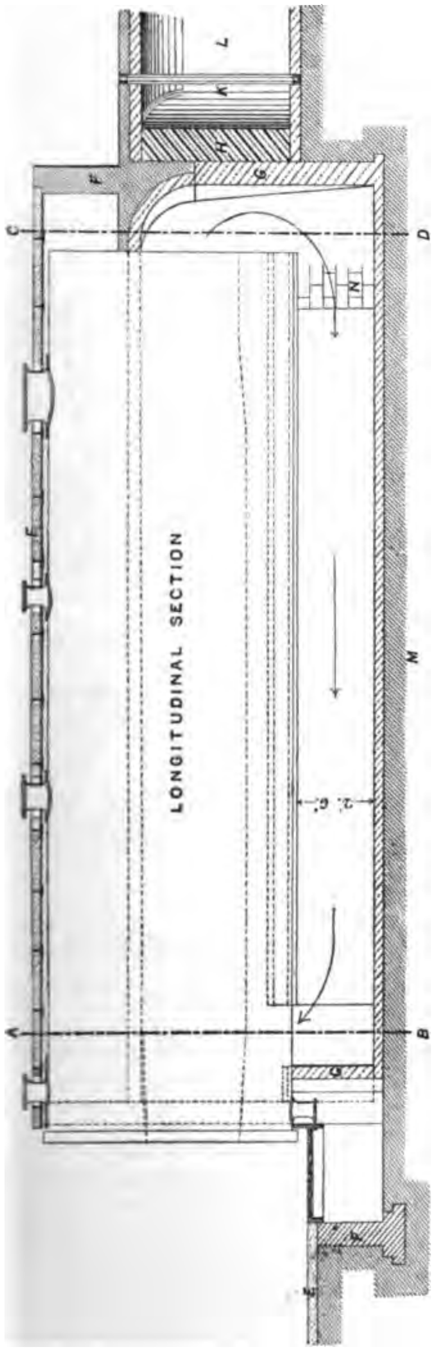

Fig. 117.

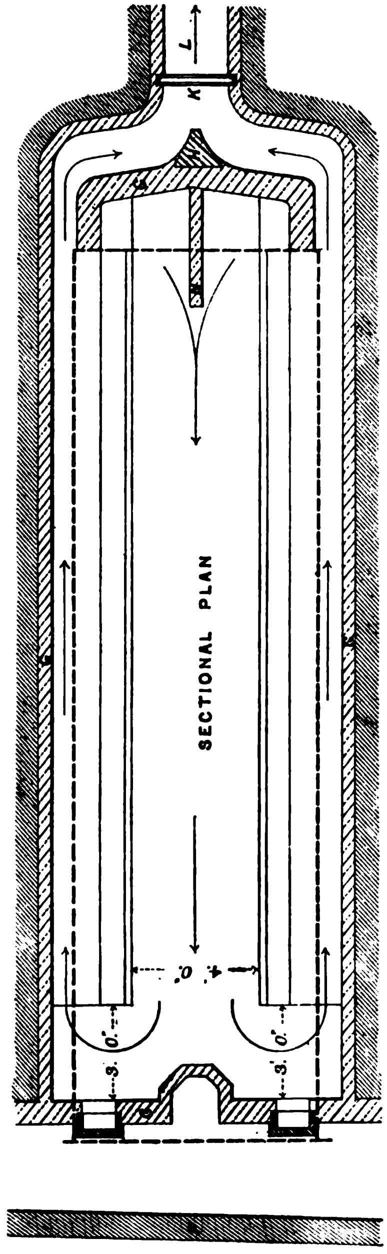

Fig. 118.

The Seating of an Ordinary Lancashire Boiler.

The effect of damp brickwork in contact with the boiler plates is to cause damage to the plates by external corrosion. It is quite possible for this to remain undiscovered until it causes an explosion of the boiler.

Scale. 6 Feet to 1 Inch.

SECTION THROUGH LINE A.B

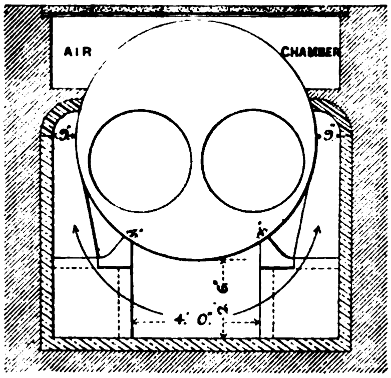

Fig. 119.

SECTION THROUGH LINE C.D.

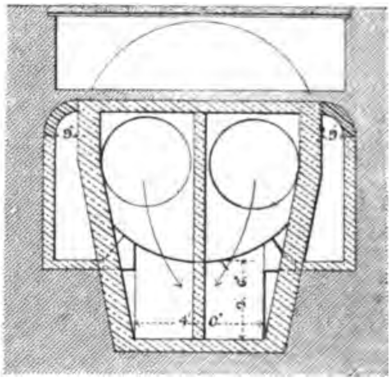

Fig. 120.

ALTERNATIVE ARRANGEMENT OF COVERING

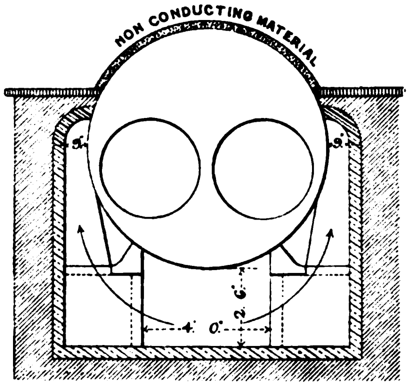

Fig. 121.

FRONT ELEVATION OF LANCASHIRE BOILER

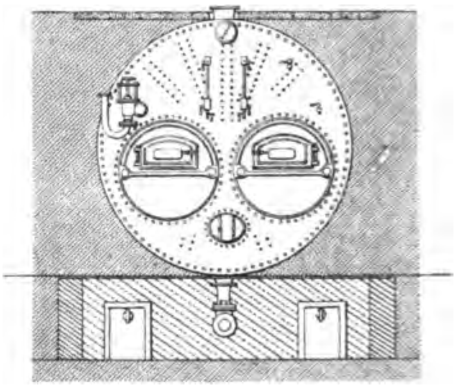

Fig. 122.

The Seating of an Ordinary Lancashire Boiler.

Sometimes the upper part of the shell is covered by a "lagging" of wood stuffed with sawdust, but this practice is no more to be admired than a brickwork covering. Coverings of patent felt and fibrous substances, which are bad conductors of heat, are much to be preferred to the previous-mentioned methods of external protection. An objection, however, to the use of these coverings is, that they do not admit of the detection of leaks so readily as in uncovered boilers. The best and most approved arrangement of external protection is that of providing a chamber closed in by thick tiles as shown in Figs. 117—120.*

This method renders the upper portion of the shell quite accessible at any time for examination. Fig. 121 shows an alternate method of covering with some good non-conducting material, but it is not nearly so good a plan as that shown in Figs. 117—120.

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* See Report of the Chief Engineer to the Boiler Insurance and Steam Power Co., Manchester, June, 1881.

Besides a separate covering for each boiler, a general roof covering should always be provided over boilers, to protect the whole range of boilers and fire-holes from the weather.

In explanation of Figs. 117 and 118,—

E shows the chamber-tiles, and floor-tiles at fire-hole.
F Ordinary brickwork.
G Firebrick walls or linings.
H Fireclay shaped blocks.
K The damper suspended by a chain passed over a pulley, at the other end of which is a damper counter-weight not shown in the drawing.
L The chimney flue, or a flue leading to a main chimney-flue.
M Concrete bed or ordinary brickwork.
N A short 4½-inch partition-wall dividing the downtake into two compartments. Its object is to prevent baffling of the draught where the gases from the two internal flue-tubes unite. The absence of this wall may cause annoyance.

It will be observed (Fig. 117) that a proper recess is formed giving access to the blow-off cock and facility for its inspection. The practice of bedding the blow-out pipe in the wall is to be condemned. The upper course of brickwork, G, at the front end is shown 4½ inches thick, or it may be brought up 9 inches thick to within a few inches of the boiler plates and there reduced to 4½ inches, so as not to increase the area of brickwork in contact with the plates beyond what is absolutely necessary.

Care should be taken in setting the boiler to allow a clear space of one or two inches between the shell angle ring and the floor-plates at the front end; at the same time the shell angle ring should itself project about an inch beyond the front cross-wall, so as to be clear of the brickwork and open to inspection. In a range of boilers the main flue should be such a size as to afford a sectional area of not less than 5 square feet for each boiler of the range. In the case of a single boiler the width should not be reduced to less than 2 feet, on account of the difficulty of cleaning it, &c.

In some cases, the downtake, instead of being covered in with the quarter-circle tiles shown in Fig. 117, has a flat cast-iron cover plate for the purpose. It is of sufficient size to reach across the diameter of the boiler and the two side-flues. In boilers having a good draught the products of combustion issue from the flue-tubes at a high temperature, which the crown tiles are unable to resist beyond a limited time, after which they are fractured and allow the downtake to fall in. The cast-iron plate is a more expensive covering as regards first cost, but where the temperature of the escaping gases is high, is much more suitable.

With regard to the position in which boilers should be placed on the surface, circumstances must decide. In selecting the situation, it will be well to bear in mind one or two points.

Low and damp situations cause moisture in the flue brickwork, and where these situations cannot be avoided, the importance of preventing any damp from reaching the plates resting on the seatings is manifest.

A comparison of Figs. 92 and 108 shows without much reflection that the former is much better calculated to resist ill-effect to the boiler shell resulting from a damp situation, than the latter. In Fig. 108 the broad seat receives any leakage water from the boiler side above, which it readily absorbs, and the moisture yielded from the damp situation is very likely to find its way to the same point, where it lodges against the shell and causes external corrosion, which is hidden from view. In Fig. 92, leakages from the boiler sides would not be so liable to remain in contact with the shell at the seating blocks, nor would moisture from the damp situation be likely to reach the shell at all, as the top of the blocks is considerably above the side flue floors.

The front of the boilers should be parallel with and close to a railway or siding, so that coal may be brought and ashes removed conveniently, and to prevent loss of heat in pipes, the nearer the boilers are to the engines the better.

Boiler fittings.—A *man-hole* large enough to admit of a person getting into the boiler to inspect its condition and do any necessary cleaning or repairing. A *blow-off cock* usually placed in front of and below the level of the boiler; a pipe leads from the bottom of the boiler, the end of the pipe being closed by a valve able to withstand the pressure of steam. The blow-off cock, on being turned, allows the water from the boiler to run through this valve. The *water-gauge*, Fig. 123, is a thick glass tube placed in the end of the boiler, one end of which communicates with the steam space above the level of the water in the boiler, the other end below the water level. It thus indicates the level of the water in the boiler. It is usually provided with a cock at each end by means of which it may be cut off from connection with the boiler, so that in case of a broken glass, the steam and water can at once be prevented from escaping and a fresh glass inserted. There is another cock at the lower end of the tube by which means the water in the tube can be allowed to escape from it. Another plan for ascertaining the level of the water is to place two *gauge-cocks* in the end of the boiler, the one being two or three inches higher than the other, the one being above and the other below the water level in the boiler. If the water is at its proper height steam should issue from the upper one on opening it and water from the lower one. These gauge-cocks should always be placed on the boiler in addition to the water-gauge. Still another method of ascertaining the height of the water in the boiler is by means of a *float* resting on it and communicating with a chain passed round a pulley above the boiler, at the end of which is attached a weight moving up and down with the float in the boiler. One of the most important of the fittings is the *safety-valve*, Fig. 124. The necessity for this is evident, for an engine may stand a considerable time, during which the pressure inside the boiler goes on increasing, and if some provision were not made for its escape an explosion must occur. The safety-valve should be placed directly on some convenient portion of the surface of the boiler, in preference to being attached to the pipe leading from the steam-valve, as often seen. The most common form is the lever. The valve is kept close by means of a lever fast at one end and having a sliding weight at the other, and near the fixed end a spindle passing through the valve, resting in its seat, communicates with the boiler. The

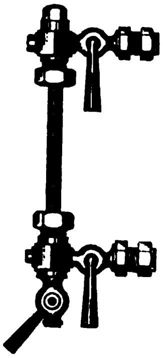


Fig. 123.—Water-gauge for Boilers.

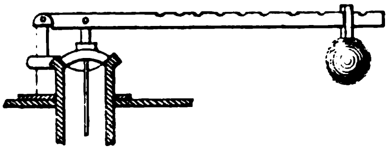


Fig. 124.—Lever Safety-Valve.

objection to its use lies in the fact that it sometimes gets corroded and sticks in its seat, and it is being supplemented by the use of dead weight and spring-valves.

Fig. 125 is a sketch of Hopkinson's 1890 patent compound safety-valve, designed to guard against either excess of pressure or deficiency of water.

The valve here shown is intended for use only with pressures below 120 lbs. per square inch, that for higher pressures being different.

The drawing shows a section of a boiler fitted with one of these valves. The exterior valve is held to its seat by the combined effect of the external weight on the lever and the internal weight on the rod, while the internal valve is held down by the weight suspended under it only. On the steam attaining too great a pressure in the boiler the external valve rises and discharges round its seat, the internal valve remaining closed. If the water is allowed to get too low, the balanced float drops with the falling water level, and this causes the projection seen on the balance-lever to come into contact with the screw-adjusted collar on the weight suspension rod, thereupon raising the inner valve and allowing steam to escape. Whether the pressure is too high or the water too low steam is thus blown off at the valve. The external fixed cage is designed to guard against the risk of the outer valve being blown out should the internal suspending rod break.

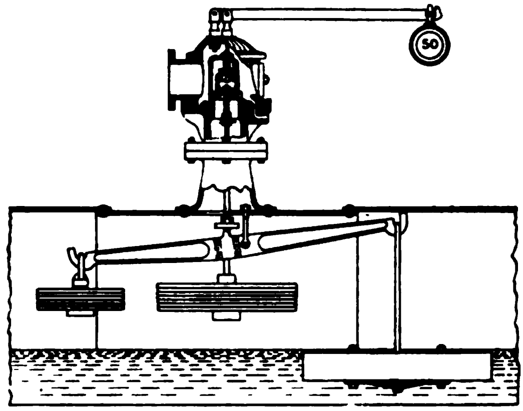


Fig. 125.—Hopkinson's, 1890, Patent Compound Safety Valve.

The valve designed and made by Messrs. Hopkinson for pressures above 120 lbs. to the square inch is termed the "Hipress." In it the high-steam and low-water valves are separated. The steam valve is spring-loaded; the low-water steam blow-off valve rests on a separate fixed seat and both valves blow off steam to the same space.

All boilers should have two safety-valves, or a safety-valve and an *escape-valve*, the latter allowing the steam to escape on attaining a certain pressure.

A good plan is to fit a boiler with a Hopkinson's patent compound safety-valve and a Hopkinson or Cowburn dead-weight safety-valve.

The *feed water-valve* is so arranged as to allow the water to flow into the boiler, at the same time preventing any from returning. It is used to regulate the supply of water to the boiler.

The *steam, crown, or stop-valve* regulates the supply of steam from the boiler to the steam-engine, and it may be closed or opened by means of a screw worked by a hand-wheel.

McDougall's patent *anti-primer* is a very interesting and novel appliance for preventing priming in steam boilers. After separating the water from the steam, the former is conveyed to the water space of the boiler by channels, in which there is no rush of steam to impede its return. The apparatus is therefore able to return the priming water to the boiler, leaving the dry steam to pass out. Figs. 126 (A) and 126 (B) show the apparatus applied to a Lancashire boiler, the former being a longitudinal and the latter a transverse section. A casing, D, contains a series of water-pockets, C, which have slots at the upper sides, B, through which the steam issuing from the boiler is compelled to pass, there being no other outlet.

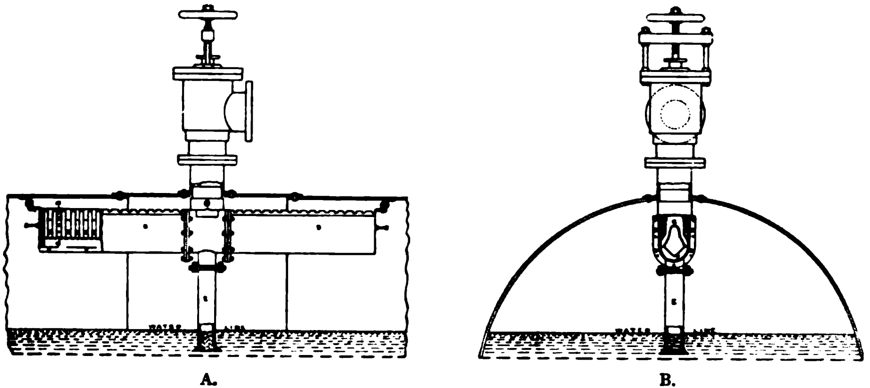


Fig. 126.—McDougall's Patent Anti-Primer.

The pockets are somewhat arched, the steam and priming water entering the slot or opening, passes on both sides of the arch, the water is conveyed along the bottom of the channels, C, to the lower chamber, F, connected with the dip pipe, E, the lower end of which passes into the water in the boiler, and thus, the dip pipe and water passages are free from any current of steam. The apparatus prevents the dirt and scum from being carried into the working parts of steam engines, &c., as they flow along with the water, are intercepted with it and returned to the boiler. The apparatus can be fitted to all types of land boilers.

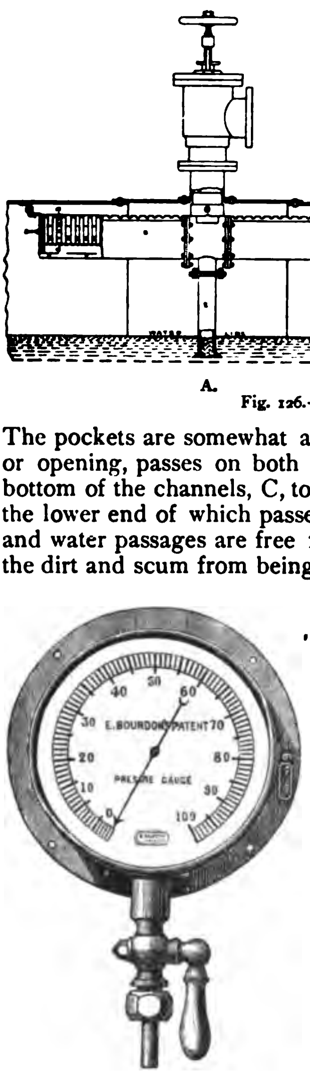


Fig. 127.—Bourdon Pressure-Gauge.

Dampers are placed in the flues to regulate the draft on the furnace, and these are hung by chains passing over pulleys with balance-weights attached.

A *fusible plug* is sometimes employed to guard against the risk of explosion. A short nozzle is attached to the boiler just below the lowest level at which the water may safely stand. On this there is screwed a cap, the central portion of which is composed of an alloy which melts at a temperature not very much exceeding the boiling point. So long as this plug is kept covered with water it remains firm, the heat being carried away from it by the water, but should the level of the water fall so low as to expose it, the centre at once melts, and allows the steam and part of the water to escape into the flues and furnace, damping or extinguishing the fire, and at the same time removing the undue pressure. An objection to its use is, that it cannot be relied on after being in some time, as it becomes incrusted or injured by the heat and loses its efficacy. The *pressure* or *steam-gauge* is the means by which we ascertain the pressure of the steam. Mercurial gauges were at one time employed, but now "Bourdon's" is chiefly used. In this there is a dial-plate with a hand on it pointing to the pressure (see Fig. 127). The steam acts upon a spring of peculiar construction, causing the index hand to move according to the steam pressure, and the figure at which the hand points denotes that the pressure inside the pipe to which it is fixed is that much above atmospheric pressure. Thus, if the hand points to 20, we know that the pressure of steam is 20 lbs. above that of the atmosphere. A pressure-gauge should be fixed to each boiler, and one also to the main steam-pipe common to all the boilers of a range. In Chapter XVI. of this work pressure-gauges are more particularly referred to.

A ***mud-hole*** is placed near the lowest part of the boilers. In the Lancashire boiler it is placed in the front end-plate; the orifice is oval in shape, and its cover is occasionally removed for the discharge of sediment.

An instrument largely used by engineers to test the accuracy of the pressure-gauges fixed on boilers is shown in Fig. 128. This ***pressure test indicator*** is manufactured by Mr. Joseph Casartelli, of Manchester, whose springs for instruments are skilfully and accurately made so as to be trustworthy throughout their range of action. The indicator consists of a barrel covered with wood containing a cylinder and piston with a spiral spring of either 25 or 50 lbs. to the inch. The scales on either side of the spring are graduated to correspond, and the pointer at the side moves up and down with the compression and tension of the spring. The springs range up to 100 and 200 lbs. respectively, but can be made stronger if required for higher pressures. The instrument is carried about with the engineer, who, when he wishes to test the accuracy of a working pressure-gauge, notes its indication. The steam is then shut off in the pipe to which it is attached and which communicates with the boiler. The pressure-gauge is then unscrewed and replaced by the pressure test indicator. On turning the cock to admit steam to the instrument the piston is forced up against the spring and the pointer shows the steam pressure on the scale. The reading thus indicated is noted and compared with that of the gauge; the indicator is then unscrewed and the pressure-gauge returned to its position on the pipe, unless its indication has been proved to be inaccurate, in which case one giving accurate readings must take its place. On many boilers now there is fixed a special cock, or tap, made to receive the nipple of the pressure test, so as to do away with the necessity of removing the ordinary gauges.

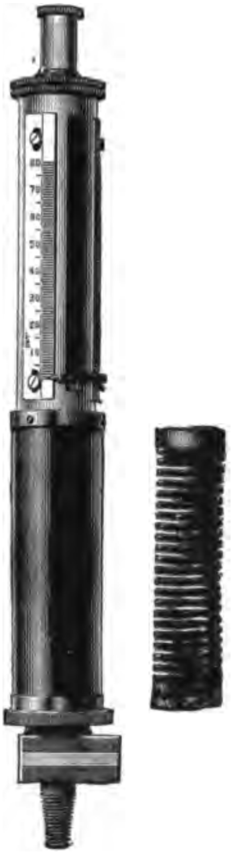


Fig. 128.—Pressure Test Indicator.

The furnace of a Cornish or Lancashire boiler consists of the mouthpiece, having doors provided with a sliding grid, shown in Fig. 122. The furnace bars are made in two lengths as shown in Fig. 131. At the front end these bars rest upon the dead plate, and at the back at a slightly lower level so that the bars may incline inwards. They are supported by the fire-brick bridge, either on a ledge formed on the bridge for the purpose, or a bearer built in it. In the middle at the joint of the two lengths the bars are supported by a cross-bearer. The bridge is usually built entirely of fire-brick to within about 20 inches of the crown of the internal flue, but sometimes a cast-iron stool is used to carry both the furnace bars and the fire-brick. The stool is provided with a sliding door, by means of which the admission of air to the furnace flue is regulated from the furnace mouth.

Some furnaces have a self-feeding arrangement. The coal in these is placed in a hopper above the furnace at the front end of the boiler, and a small revolving scoop driven by an engine constantly and slowly sprinkles the coal into it. A slow motion is also imparted to the furnace bars, so that the burning fuel is gradually carried to the back of the furnace as fresh coal is supplied in front. The objection to this plan is the complicated nature of the mechanism and the power required.

In the Bennis and also in Proctor's mechanical stokers the coal is distributed over the fire by flaps or shovels.

The economical working of steam-boilers requires not only that they shall be of proper design and maintained in efficient repair, but that strict attention shall be given to all details affecting the consumption of fuel with the view of obtaining its complete combustion. The fire-grate is, therefore, a highly important factor in the economy of boilers, and especially so where fuel can only be obtained at a high price.

It has been proved by the researches of Wye Williams, Sir William Fairbairn, and others, that a considerable saving is to be effected by the admission of air at the back bridge. Fairbairn has stated, as a result of experiments in this matter, that a clear gain of 4 per cent. was effected, and that it was thoroughly effectual in preventing smoke emission.

Caddy's patent tubular fire-bars have now been before the public since 1886, and have been adopted by the Admiralty, H.M. Office of Works, public corporations, and the leading boiler-makers throughout the world. They are not only used for ordinary furnaces, but have been applied with increasing success to most of the mechanical stokers in the market.

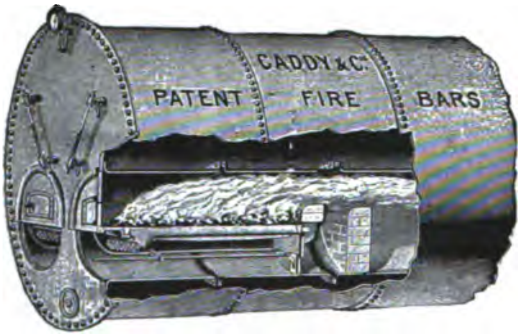


Fig. 129.—Caddy's Patent Tubular Fire-bars.

The patent consists of a hollow bar (see Fig. 129), cast face downwards, round an inner flattened tube of wrought-iron, and has a face-breadth of one inch, making it suitable for slack coals. Being cast in chill, face downwards, the surface is made smooth, and less liable to gather clinker.

The internal wrought-iron tube affords a passage for air from the front, to be delivered at a split bridge at the rear of the grate as an aid to smoke prevention. The air becomes heated and the bar cooled thereby; this tends to prevent the adherence of clinker.

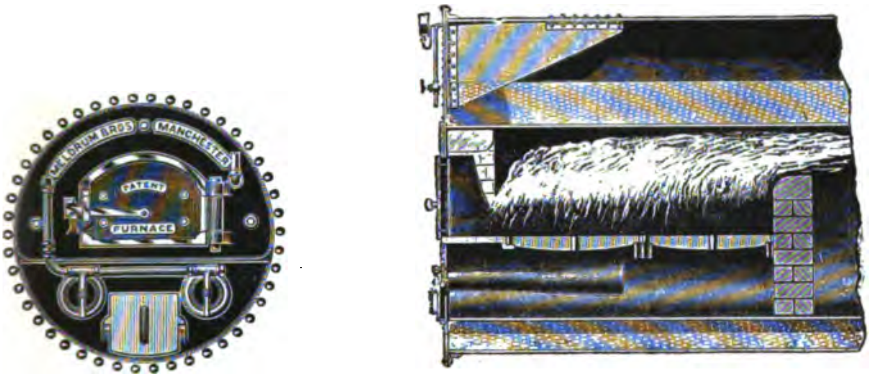
The face is somewhat rounded, and the total depth of bar is 4½ inches. Each bar has an air-way of about three-fourths of a square inch. Hence, with air-spaces of ¼ inch, the air-way through the bars amounts to about 7 square inches per foot of breadth, or about an inch and a half per square foot of an ordinary grate.

To show the advantage of the adoption of these bars, a large number of tests have been made, under the Alkali Works Regulation Act, full details of which can be obtained from the Government Blue Book. A ten hours' test of Caddy's patent bars, made with Lancashire boiler, 34 feet × 8 feet 6 inches, with 9 tubes, gave the following results:—Water evaporated per lb. of coal from and at 212° F. =7·97 lbs. Water evaporated per lb. of carbon value from and at 212° F.=

9·74 lbs. Percentage of ashes = 13·25 per cent. Temperature of gases leaving boiler = 215° F. No smoke. The gas analysis showed of carbon dioxide, 9·43 per cent.; carbonic oxide, *nil;* oxygen, 8·81 per cent.; nitrogen, 81·76 per cent. The coal used was Bridgewater rough slack.

If desired, the patent fire-bars can be fitted with a rocking attachment, not shown in the illustration, patented by the same firm. By means of this gear a dirty fire may be easily and quickly cleaned, without opening the furnace-door, or using the pricker-bar, which in careless hands causes injury by pricking the bars up. The motion is so arranged that by pressing a lever every alternate bar moves longitudinally about three inches. During this backward and forward traverse the front end of the bars in motion rises and falls an inch and a half, while the other half-set of bars remains in its normal position. This compound motion cleanses and renders the fires bright and assists in the reduction of smoke.

In chimneys of ordinary height the draught, which seldom exceeds that equal to about ¾ inch of water gauge, is insufficient for the combustion under boilers of any but the better class of fuel, and in order to burn inferior kinds it has been found advantageous to employ, as an auxiliary, what is termed forced draught, and in this way combustion may be supported in a fire fed by a close-lying fuel.



A.—Front Elevation. B.—Longitudinal Section.

Fig. 130.—The Meldrum Furnace applied to a Cornish Boiler.

The Perret system for burning coal-dust and other rejected fuels is an example of an apparatus for producing forced draught. In it specially constructed fire-bars are used, which dip into a trough or tank containing water placed in the ash-pit. As the water in the trough is evaporated, the steam tends to cool the bars and reduce their rate of waste by burning. By means of a fan air is forced through pipes to each grate, so that the space between the water-line in the trough and the coal above the bars is under pressure. This tends to prevent the formation of clinkers. The vertical currents of air between the bars tend to cool them, and also to prevent dust from falling into the tank.

The Meldrum furnace is an ingenious and carefully designed system of forced draught, by means of which refuse fuel may be utilised for steam-raising; and it is stated that by using ordinary breeze or coke-dust containing 20 per cent. of moisture, 15 per cent. of clinker, and 65 per cent. combustible, 6½ lbs. of water per lb. of fuel have been evaporated with these furnaces.

Fig. 130 shows the furnace as employed in a Cornish boiler, A being a front elevation, and B a longitudinal section. The thin bars are placed very close together, the air-spaces being only from $\frac{1}{16}$ to $\frac{1}{8}$ inch wide, so that only the finest ash can drop between. The front of the ash-pit is closed and blowers (usually two in number) are fixed into the plate which closes the ash-pit. The blower consists of a long cast-iron tube of uniform size, for a certain portion of its length, after which it widens out into a trumpet shape at its inner end, as seen in the longitudinal section. The inlet portion of the tube is curved to a special form. A very small nozzle is fixed centrally at the outer end of the tube, and through it a jet of steam may be injected, to give motion to the column of air within the tube, the momentum of which is sufficient to cause a considerable amount of pressure under the bars if necessary. This pressure is equally distributed over the whole area of the grate, and is ordinarily equal to about $\frac{1}{2}$ or $\frac{5}{8}$ inch of water column, but a much greater pressure has occasionally been produced. External air immediately afterwards enters the tube and a large volume in proportion to the steam used is forced through the tube. After passing over the bridge, the flame and hot gases travel through the boiler flues at the ordinary rate of chimney draught.

The blast is brought into action and may be regulated by means of the valve supplying steam. This valve is under the control of the fireman, and may be opened to produce a strong blast, if immediately after stoking, or closed so as to stop the blast, as the fire burns thin, or the steam may be allowed to issue at the nozzles to produce a blast of any required force.

Where the load is very variable, and when the fireman has other duties besides firing, the valve is made to work automatically, so that when the steam is at blowing-off point the valve closes and the draught ceases; when the necessity arises it comes into operation again. This patent contrivance for automatically working the valve is simple yet ingenious.

Forced draught is of great service when, as frequently occurs at collieries, more boilers are added and connected to an existing stack. The life of a boiler does not appear to be shortened in consequence of the adoption of forced draught.

Fig. 131 shows a longitudinal section, Fig. 132, a side elevation, and Fig. 122, a front elevation of a Lancashire boiler, the seating for which is shown in Figs. 117–121.

In Fig. 131, A and C are blocks for the safety valves, a dead-weight safety valve to be placed at A and a Hopkinson's double safety valve at C.

B is the block for the stop valve, to which is also fitted a perforated or anti-priming pipe inside the boiler, but not shown in the drawing.

D is the manhole block. It is usually oval but sometimes circular in shape, and is large enough to admit a man to the interior of the boiler to clean or repair it. It is placed in any convenient position at the top of the boiler.

E	are gusset stays.
R	longitudinal stay.
WL	the water-level in the boiler. Modern practice fixes the low water line at 4 inches and the working level at 9 inches above the furnace crowns.
H	Galloway tubes.
K	furnace bars.
G	the fire-brick bridge.
P	the internal flue, the rings being welded and connected to each other by the Bowling hoop expansion joint.
M	the blow-off block.

The feed pipe F extends about 8 or 10 feet into the boiler, and has the inside portion perforated to allow of a gentle distribution of water all round it.

N is a mud-hole for the discharge of sediment from the boiler, and is shown also in the front elevation, Fig. 122. In Fig. 122 will also be seen the feed-water-pipe and valve, two glass water-gauges, two gauge cocks, the furnace doors, blow-off cock, and the cleaning flue doors.

Lancashire boilers to work at 110 lbs. per square inch are at present made of the following dimensions:—

Length 30 feet. Diameter of shell 8 feet. Diameter of internal tubes 3 feet 3 inches. Number of shell rings 9, made of mild steel having a tensile strength of 30 tons to the square inch. Two plates in each ring $\frac{5}{8}$ inch thick; circular seams, lap-jointed double zig-zag riveted, $3\frac{1}{4}$ inches pitch, $1\frac{1}{4}$ inches line to line; horizontal seams butt-jointed with inside and outside strap plates $\frac{1}{2}$ inch thick, pitch $3\frac{1}{16}''$, $1\frac{7}{8}$ inches line to line in the outer lines, $2\frac{1}{2}$ inches line to line in the centre line. Rivet holes $\frac{15}{16}$ inch, all drilled in position. Diameter of rivets $\frac{7}{8}$ inch, of mild steel machine-riveted. End plates of mild steel $\frac{11}{16}$ inch thick, the back-plate flanged, the front single riveted to shell angle iron 5 inches by 3 inches by $\frac{3}{4}$ inch. Internal tubes of mild steel $\frac{7}{16}$ inch thick, 3 feet 3 inches diameter, the plates having a tensile strength of 22 tons per square inch. Internal tube rings hand-welded, flanged and attached by $3\frac{1}{2}$ inches by $3\frac{1}{4}$ inches, by $\frac{5}{8}$ inch angle irons to the end plates. Five gusset stays at the front and back end plate above the tubes and two below, and one centre gusset. The boiler subjected to a hydraulic test of 250 lbs. to the square inch.

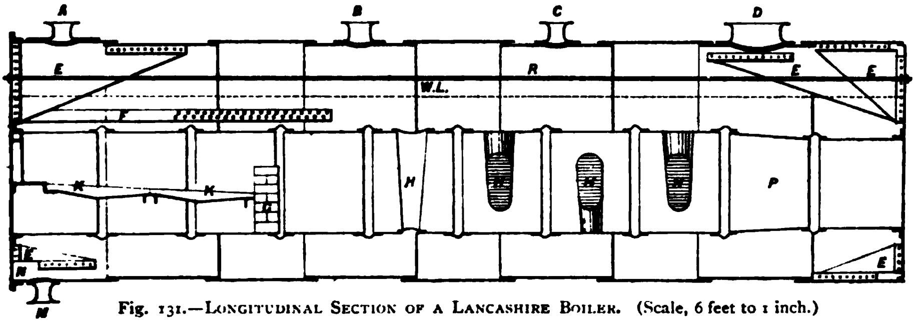


Fig. 131.—Longitudinal Section of a Lancashire Boiler. (Scale, 6 feet to 1 inch.)

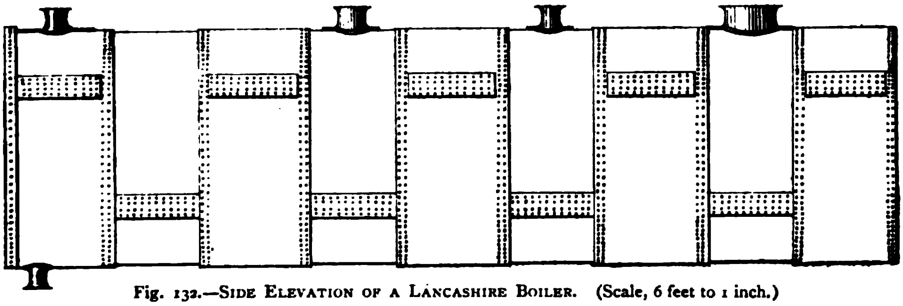


Fig. 132.—Side Elevation of a Lancashire Boiler. (Scale, 6 feet to 1 inch.)

A boiler should be tested to at least double the pressure it is intended to work at before being sent from the works, a hydraulic pump being used for the purpose, and a close examination for leakages maintained throughout the test.

Fig. 133 shows Casartelli's *portable hydraulic pump* for testing boilers up to any pressure. The stand and legs are all case-hardened iron, the working parts being of best gun-metal. It is fixed near the boiler to be tested and is provided with a suitable length of suction hose, one end of which is fitted with a brass rose to prevent the entry of chips, &c. The water may be drawn from a bucket or from some other source, for which purpose the suction hose is led to it. The end of the delivery hose is fitted with a suitable joint and screw for connection with the boiler. The boiler to be tested is first filled with water, and then the pump is brought into operation. On working the lever, after the connections are made, the pump it works forces water into the boiler through the delivery hose. The gauge is graduated up to 350 or 400 lbs., or higher still if necessary, and the pumping is continued until the pointer on the gauge reaches the number of lbs. pressure on the graduated scale it is desired to test to. The legs can be taken off when desired, as well as the two hoses and lever, and the whole apparatus may then be packed away in a portable case 25 × 12½ × 16½ inches.

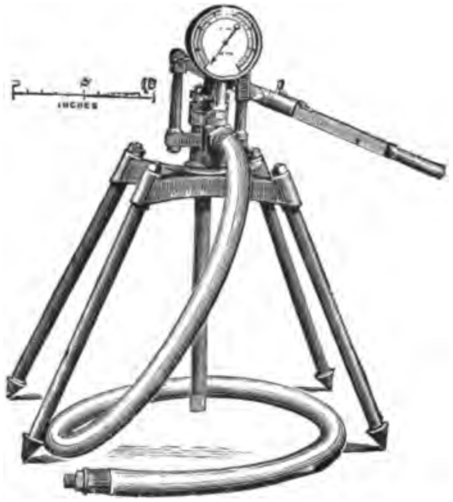
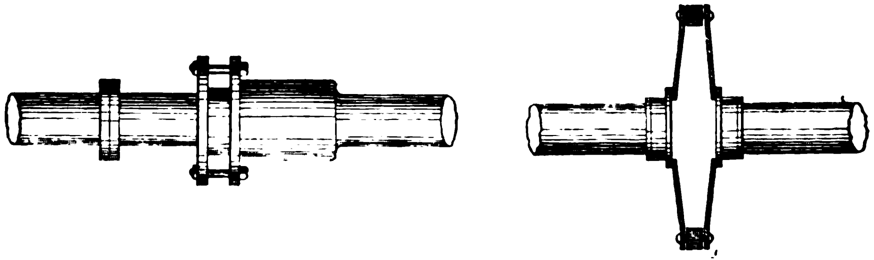


Fig. 133.—Portable Hydraulic Pump.

Insufficient attention in many cases is given to the various pipe connections.* The principal connection of a steam boiler is the steam pipe. The steam from the boilers should pass into the main pipe direct, or into a receiver of large size from the top of which the main pipe afterwards leads. The receiver or main pipe crosses over the boilers at right angles to their length, and each boiler of a range should have a separate connection with the receiver. The receiver should have no inclination either way, and some provision must be made for expansion and contraction of the pipes. The steam pipes should be protected from loss of heat by radiation, and for this purpose a covering of patent felt may be placed over them. Sometimes the receiver is supported by pillars from the boiler walls, rollers being placed on the pillars for the receiver to rest on and to permit of expansion and contraction.

Fig. 134 (A) shows the usual form of expansion joint used for steam pipes. It consists of an ordinary stuffing box and gland, and provides for longitudinal motion in the pipes by allowing them to slide in and out to an extent depending on the expansion and contraction which take place, and at the same time remain steam tight. In any long length of straight pipes which are not subject externally to upward or downward pressure or strain, and having no connection with them



A.—Stuffing-Box Form. B.—Diaphragm Form.

Fig. 134.—Steam-Pipe Expansion Joints.

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* The following observations respecting steam-pipe connections and expansion joints are, to a large extent, drawn from the columns of the *Practical Engineer*.

except at the two extremities, this expansion joint renders good service, the pipes being free to expand and contract without interference by side strains. These expansion joints, nevertheless, require attention, as the material with which they are packed wears, and so causes leakages unless renewed when necessary.

Fig. 134 (B) shows another form of expansion joint, which it was thought would be an improvement on that of Fig. 134 (A). In it two thin diaphragms are secured at their outer circumference to a block ring which is about 2 inches thick. The central portion of each diaphragm receives the steam pipe at its inner circumference, and as the diaphragms are thin and considerably larger in diameter than the pipes on either side, they form an elastic joint. As the steam pipes expand or contract the diaphragms approach or recede from each other, at that portion which is joined to the steam pipes.

It is very questionable whether this form of expansion joint is any improvement on that which it was designed to supersede. The steam in its passage along the pipes enters the space formed between the diaphragms with considerable force, and the pressure arising therefrom tends to separate the diaphragms, which may at the same time be subject to other motion resulting from expansion and contraction of the pipes. To ensure the necessary elasticity, the diaphragms must be made thin; if too thin they are unable to stand the force of the different pressures acting on them, and so give way.

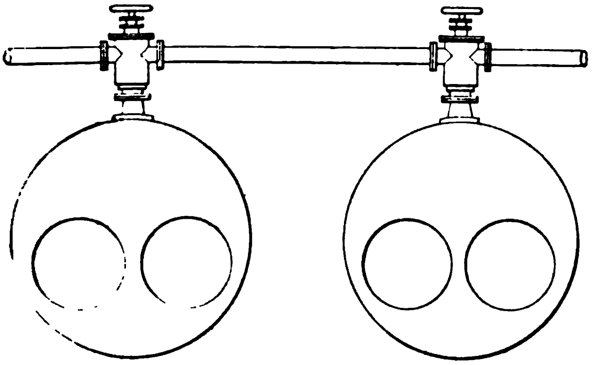

Fig. 135.

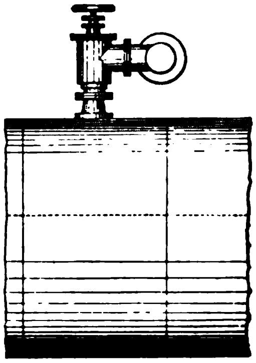

Fig. 136.

Undesirable Methods of Connecting Steam Pipes with Boilers.

In arranging a system of steam pipes from the range of colliery boilers several points call for consideration. Frequently the main pipe is connected with the boilers as shown in Fig. 135, but it is an arrangement not to be recommended. The motion to which a boiler is subjected is transmitted to the connections made with it, and when at work there is continual expansion and contraction taking place, which will change in accordance with the firing of the boiler and the consequent change in temperature and steam pressure. In a range of boilers these changes will be irregular at all times, and be most marked on laying the boilers off for cleaning, or repairs, and on re-starting them. Any tendency on the part of the boilers to revolve in their seats, following from expansion and contraction, causes a strain on the pipe joints and flanges, which, if severe enough, fractures them.

The boilers of a range are also subject to unequal settlement in their foundations, which is followed by a strain on the steam pipes; this will be more or less injurious to the joints according to the distance between boilers.

In a range of boilers having the steam pipes connected with them as shown in Fig. 135, the effects of expansion and contraction on the steam pipes may be slightly guarded against by fixing an expansion joint between every two boilers of a range. But this will not be sufficient to remedy the evils of all possible strains, and in a range composed of many boilers introduces a constant source of anxiety to the boiler attendant, who will have considerable difficulty in keeping so many stuffing boxes in order and steam tight. On account of the attention required to prevent leakages at expansion joints, the aim should be as far as possible to dispense with their use.

Fig. 136 shows another arrangement of steam pipes which in this case are carried across the range of boilers close to the stop valve, and the objections urged against Fig. 135 apply with equal force to the arrangement shown in Fig. 136.

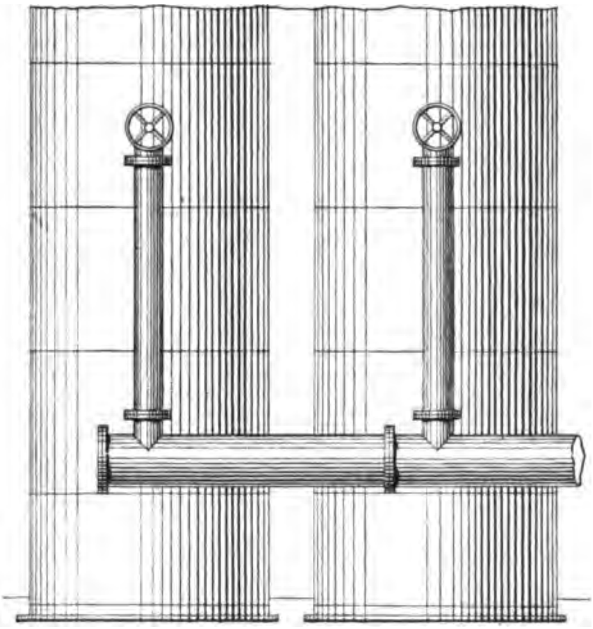

Fig. 137.—Approved Method of Connecting Steam Pipes with Boilers.

A method of connecting the steam pipes to a range of boilers free from the objections given to those just described is shown in plan, in Fig. 137, and, in elevation, in Fig. 138. Here, the main steam pipes cross the range of boilers at right angles at a distance of 7 or 8 feet from the stop valves, and a branch pipe at right angles to the main pipe is laid from each stop valve to the main pipe. To a very large extent this provides for unequal settlement of the boilers at their foundations. The motion caused by the settlement of one boiler is transmitted to the branch pipe, but while the full effect of the settlement is felt at the stop valve end of the branch pipe, the strain resulting at its other end which is fixed to the steam pipe will be slight, and may be provided for by not carrying the main pipes on fixed, unyielding supports from the boiler walls. These should be carried on rollers placed on springs which, while allowing for longitudinal expansion and contraction in the pipes, would also yield to the motion transmitted by the branch pipe.

and in turn convey it to the branch pipe on either side of that first receiving motion.

The use of expansion joints may be altogether dispensed with by providing springing lengths in the pipes. Where practicable, bends may be placed at right angles to act in this way, either along the course of pipes or at the two extremities. Where this method cannot be pursued, a horizontally laid U-shaped bend may be introduced between the two boilers at both extremities of the range. As variations in temperature proceed, the legs of the U will be acted on so that the two ends move slightly nearer to, or further from each other.

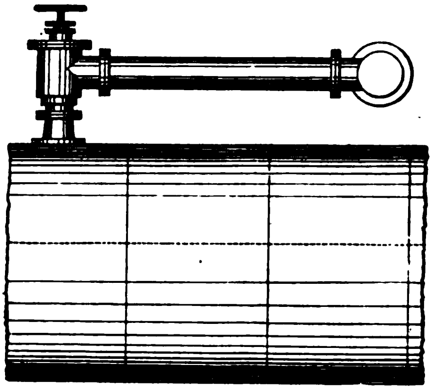
Fig. 138.—Approved Method of Steam-Pipe Connection.

Fig. 139 shows another method of laying the steam pipes from the boilers which is nearly as effective as that shown in Figs. 137 and 138, and is suitable under circumstances that prevent the arrangement there shown, such as irregularity of line in the stop valves resulting from boilers of a range being added to from time to time, or obstructions crossing a line which would otherwise be chosen at the stated distance from the stop valves.

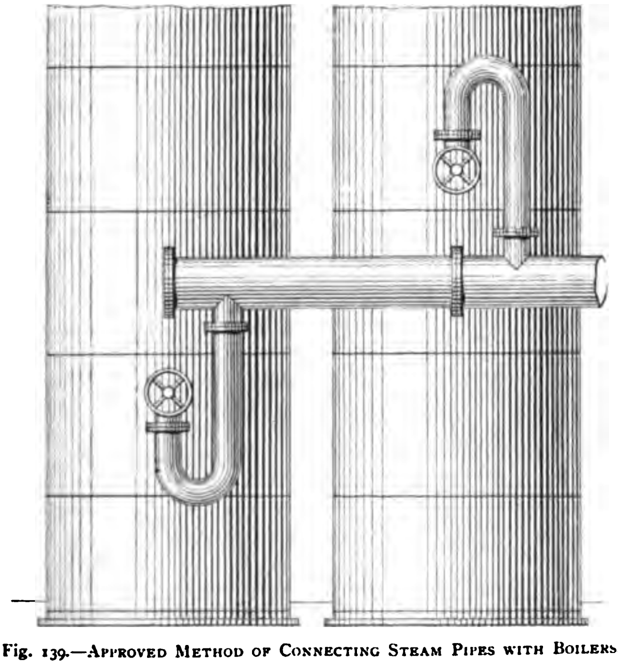
Fig. 139.—Approved Method of Connecting Steam Pipes with Boilers.

In this case the branch pipes forming the connection to the main pipe, instead of being straight, are curved and doubled back, as shown in the figure.

Whichever system is adopted, the bends of the U's where placed at the two extremities of the steam pipes, and the steam pipes themselves across the boilers, should be laid horizontally, in order to avoid all lodgment of water in the pipes. If the steam pipes cannot be continued in a horizontal position to the engines, they should be slightly inclined towards them, and any water formed through condensation be removed by separators, or steam traps.

Where the steam pipes are not laid on a level with the stop valves, some means have to be devised to overcome the difference in level, but it can never be so good an arrangement as that which places the stop valves and steam pipes on the same level.

Fig. 140 shows a plan sometimes adopted to overcome this difference of level. Under ordinary working conditions it may work effectually, but when one boiler of the range is idle the branch pipe becomes a receptacle for condensed water, necessitating the use of a drain pipe and tap, which must be fixed under the branch pipe, and so add to the cares of the already fully burdened boiler-attendant.

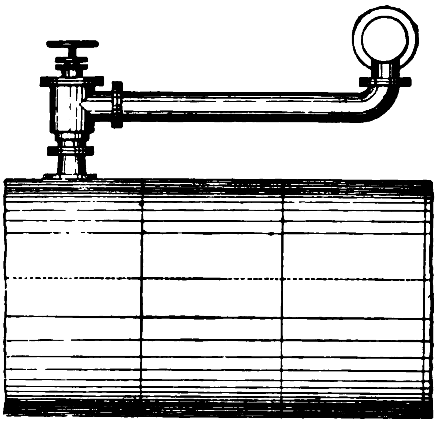

Fig. 140.—Means resorted to through neglecting to place the Main Steam Pipe on a level with the Stop Valve.

Where steam is generated on the surface and conveyed in pipes down the shaft, expansion-joints may be placed at suitable intervals—ordinarily about 50 yards—in the shaft. As the pipes form a long straight line in the pit, the ordinary expansion-joint is usually adopted and found to answer well.

Fig. 141 shows the plan adopted at the Elemore Colliery, Durham, for the prevention of ill-effect from expansion, without the use of expansion joints, a description of which is given in *The Coal and Iron Trades Review* of July 31, 1891. Here the column of pipes is supported at the pit bottom by a pair of large girders. The 40 feet of horizontal pipes laid in the drift near the surface act as a springing length, and allow for sufficient movement of the vertical and horizontal pipes. The pipe, B, facilitates the motion as it is a knuckle-piece having two right-angle bends in it. The weight, A, is attached by means of a chain passed over two pulleys to the horizontal pipes, and serves to check a too sudden fall of the pipes in the shaft when contracting. The pipes are 9 feet long, and at intervals of every third pipe, support-girders, E, and C, D, at right angles to E, are fixed between flanges to allow of the pipes sliding up and down. These girders prevent the pipes from bulging whilst expansion proceeds. The upper girders, C, D, are bolted to the lower which are let into the shaft, or where there is tubbing are secured to brackets on the tubbing.

If the pipes have to be continued from the bottom of the shaft into the underground workings along roadways which are subject to disturbance, either from lateral pressure or upward or downward variations of the floor and roof, the ordinary expansion-joint is unsuitable, because it does not yield to the road disturbances sufficiently to prevent injury or breakage to the pipes and consequent blowing at the joints.

Fig. 142 shows an expansion-joint especially adapted for allowing air,* steam or water pipes to accommodate themselves to the irregularities of underground roadways. A ring or cylindrical casting, K, is made with internal projections, F, at each end, of such size as to pass freely over the outside of the pipe B B. These projections are about $\frac{5}{8}$ or $\frac{3}{4}$ of an inch, and allow the ends of the pipe to move to the inside of the ring at J J, and by so doing the pipe, if 9 feet long, can be placed from 9 inches to 18 inches out of a straight line, as shown in dotted lines at A A, without at all affecting the security of the joints at D D. The joints D D are made by placing india-rubber rings over each end of the pipe B B, and to ensure their having a thoroughly tight grip of the pipe, the india-rubber rings must be smaller than the external part of the pipe B B. The ends of the cylindrical ring K are grooved out at F, and so also are the inside of the loose flanges E E, so that the india-rubber rings C C are kept firmly in place and cannot be blown out by pressure.

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* See Transactions, South Wales Institute of Mining Engineers, vol. xvi. pp. 188, 189.

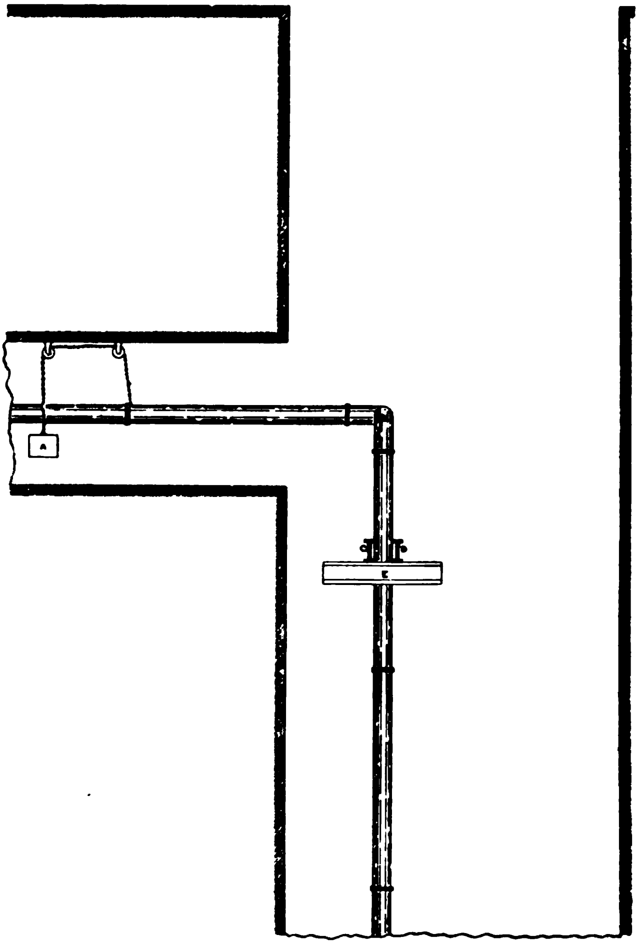

Fig. 141.—Sketch showing Arrangement of Steam Pipes in a Shaft where the use of Expansion Joints is dispensed with.

After all parts are put in their places, the through-bolts G G are screwed tight, when the joint is completed, ready for use.

Any pipe may be used for putting on the expansion-joint by cutting 5 or 6 inches out of the middle, or two old or broken pipes may be taken if one flange of each is good.

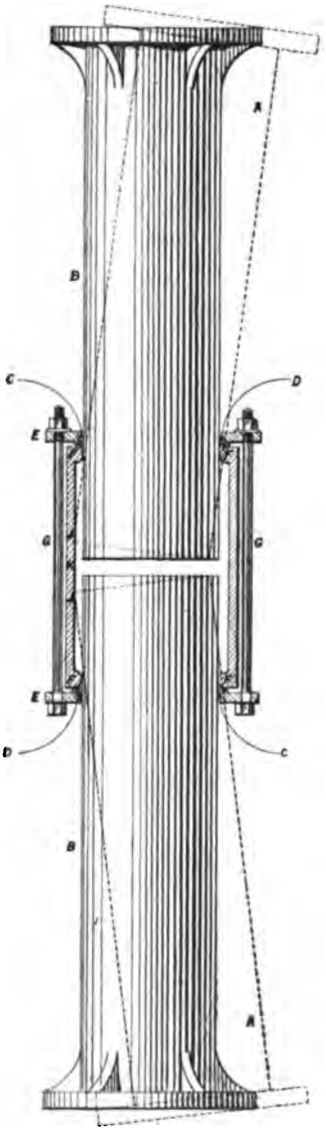

Fig. 142.—Steam-Pipe Expansion Joint for Underground Roadways.

The joint being simple is quickly made, as it requires no special connecting-pipes when joining new pipes, and when replacing an old one it need not be exactly the same length as the one removed; the length of the pipe may be varied a few inches by pushing it in or drawing it out of the expansion-ring K.

The feed-water should be conveyed to the boilers hot. The temperature of the water in its passage along the pipes is not high enough to cause inconvenience from expansion and contraction of the pipes, and no expansion-joints are needed. The exhaust steam is often used to heat the feed-water before the latter passes into the boilers, but unless this is done without allowing the steam to mix with the feed-water it is objectionable, as the grease and dirt pass from the cylinders into the boilers and so into the cylinders again and again. There are various methods of utilising the heat in the exhaust steam, most of which consist of passing it through pipes surrounded by the water-tank out of which the feed is supplied or by reversing the arrangement. Formerly feed-pumps were used to force the water into the boilers, and this was frequently done without raising the temperature of the water before doing so. The water entered the boiler through a pipe at the top and extended downwards a little, terminating in an open-mouthed pipe. Where the water has to be brought from a distance or from any point below the level of the boilers, pumps worked by small engines may still do most useful work, but they are not much used to pump the water direct into the boilers. The usual practice is for the pump to deliver the water into a tank placed a little above the level of the boilers, unless the water will reach the tank by gravitation. After the temperature of the water has been raised in the tank, it is conveyed by means of injectors to the boilers; or, if an exhaust injector is used, the temperature of the water is not raised in the tank, but enters this form of injector at 90° F. or lower.

The injector is lighter, occupies less space, and absorbs less power than a feed-pump, and is quite as reliable in action, with the further advantage of coming into action only when required.

Injectors may be divided into two classes, viz., live-steam injectors and exhaust-steam injectors. The action of both is somewhat strange, and has puzzled many engineers. In the former the water is forced into the boiler by the pressure of live steam direct from the boiler acting on the surface of the water in the injector, under certain conditions. In the latter the same object is attained by the use of exhaust steam, also under certain conditions.

M. Giffard invented the injector many years ago, and it was introduced into this country by Messrs. Sharp, Stewart & Co., whose works were formerly at Manchester, but have been removed to Glasgow. This form of injector is still the most generally used, but it is not applicable to all purposes. Although Giffard's injectors are stated to draw water from the supply, they all, but more especially the small sizes, work better when placed below it, so that the water may run to the injectors by gravitation. The pressure of steam to work the injector must not be under 5 lbs., nor must the pressure against the water being forced into the boiler be more than twice the pressure working the injector. It cannot be employed when the temperature of the feed-water is above 135° F. for low pressures and 105° F. for the highest pressures, as a part of its efficacy arises from the condensation of the steam. If therefore these temperatures are exceeded, a large volume of water is required to condense the steam, and the effect of this is to reduce the velocity of the steam in driving forward the large body of water. To work efficiently under the conditions named, the regulators require very careful adjustment.

The following particulars are furnished by the Patent Exhaust Steam Injector Co., Limited:—

"The Giffard injector was the first ever made, and the improved form of the original instrument, shown in Fig. 143, has been proved by experience to be a trustworthy apparatus, simple to work, and not liable to get out of order. Having self-contained steam and water regulators, it will work at a greater range of steam pressure than any other class. It may be fixed either above or below the level of its supply water, and the fixing can be done by any local engineer or mechanic.

"Among its advantages are the following:—

1.—Small first cost, the absence of all working parts in motion, and the consequent small repairs.
2.—It supplies water to the boiler without the necessity of working the steam engine, and in a continuous stream instead of intermittently as with a pump.
3.—The steam admitted to the injector is condensed and re-enters the boiler with the feed-water, thereby raising its temperature, so that only hot water enters the boiler, and the straining of plates consequent on unequal expansion when pumping in water at a low temperature is avoided.
4.—The consequent saving of the power required to work pumps, as well as of their wear and tear, which, in the case of donkey-engines especially, is considerable.
5.—The freedom from risk of damage or stoppage by frost.

"Very many thousands of these improved injectors are at work all over the world, and the makers confidently recommend them as being in all respects most complete and efficient, and certain to give the greatest satisfaction to steam users in all countries. Having all the progressive number books in their possession, exact duplicate injectors of these and all other classes made by Sharp, Stewart & Co., Limited, can only be obtained from their sole successors, The Patent Exhaust Steam Injector Co., Limited, 4, St. Ann's Square, Manchester.

"*Instructions for Fixing.*—The injectors may be placed horizontally or vertically, with connecting pipes. There must be a continuous supply of dry steam, with a cock or valve on the steam-pipe to the injector. The water-supply must be continuous, and should not be hotter than 135° Fahrenheit with low pressure, or 105° Fahrenheit with the highest pressures. The delivery to boiler is of course much hotter than this. The injector may be placed either above or below the level of the water-supply; the small sizes will draw from 2 feet to 5 feet, and the

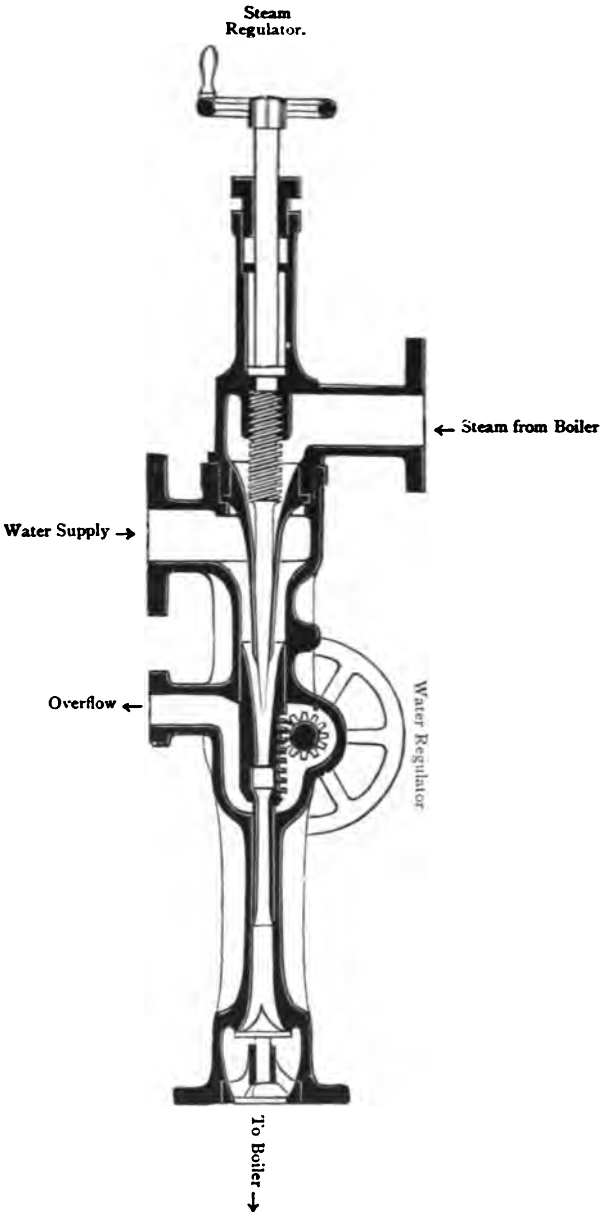

Fig. 143.—The Giffard Injector.—Class A.

larger ones 12 feet. There must be a back-pressure valve on the delivery pipe to the boiler. The water-supply pipe must be perfectly air-tight. A cock or valve is only required on this pipe when the water flows to the injector, and has to be shut off when not working.

"*Instructions for Working.*—Open the cocks on the steam- and water-supply pipes. Open the water regulator and slightly open the steam regulator until water issues from the overflow, after which, open it fully and the water will be forced into the boiler. If all the overflow is not taken up when the steam regulator is fully opened, adjust the water regulator to stop this. The water regulator should be turned to its full extent occasionally, to prevent its setting fast. The quantity of water delivered to the boiler may be reduced when the injector is working, by turning down the steam regulator until there is an overflow, and taking this up as usual by the water regulator.

" Steam from the boiler passes through the top branch pipe where shown on the drawing, and is admitted to the injector through a conical nozzle. The lower end of a vertical spindle fits accurately into the nozzle, and an ascending or descending motion may be given to this spindle by means of the screw actuated by a hand-wheel at the top. The admission of steam may be thus regulated. The feed-water enters the injector on the opposite side from the steam and flows through a branch pipe at a slightly lower level to the steam-pipe branch, as shown in the Fig. It passes round the outside of the conical nozzle to meet the steam which rushes in with great velocity; the steam is condensed and a partial vacuum formed, into which the following steam rushes with great velocity, to be in its turn condensed, but the water is carried along with it, and the energy is so great that the water is forced into the boiler notwithstanding the pressure against it. The water supply is regulated by the hand-wheel shown at the side. It works a small pinion inside the injector, and, on being turned, moves a tube up and down. Provision is made by means of a branch pipe placed below the level of the feed-water branch, for the overflow water. A back-pressure valve is placed at the bottom of the injector, and on the side of the boiler not shown in the Fig. a check-valve is placed, which prevents the escape of water when the injector is not in action."

The difficulty of understanding the action of the injector arises from the fact that the steam is able to drive the water it meets in the steam-cone into the boiler against a pressure equal to or greater than its own.

The explanation given in Prof. Jamieson's *Text-book on Steam and Steam-Engines* is as follows:—If two jets issue under the same pressure, one of steam and the other of water, the velocity of the former will be many times greater than that of the latter, and if steam, in its passage from a boiler, is condensed to water, but not reduced in velocity to that of water issuing under the same pressure, it is then able to overcome the pressure of water in its own boiler. In the Giffard injector, the meeting of the steam and water in the nozzle causes condensation of the steam without its losing its velocity, except that due to the friction of the passages. The partial vacuum, caused by the condensation, is followed by more steam, which rushes in with great velocity to meet more water in the nozzle; condensation follows, and the feed-water in advance is carried on by the force of the condensed steam-jet into the boiler. The velocity of the steam-jet is necessarily reduced by imparting a high velocity to this volume of water, but it is not reduced nearly so low as that of a jet of water issuing under the same pressure, and hence it is capable of overpowering and forcing back the water in the boiler.

There are many forms of live steam injectors, in which improvements have been attempted, the greatest of which is for the apparatus to work when placed above the level of the water-supply. The most that has been attained hitherto in this direction does not exceed 20 feet.

The Exhaust Injector is the invention of Messrs. Hamer, Metcalfe & Davies. The feed-water tank must be placed at a slightly higher level to that of the injector. The latter may be anywhere connected to the main exhaust-pipes, but the nearer to the boilers the better. An elbow in the exhaust-pipes makes a convenient point of connection. As the exhaust steam passes along the pipes, the injector takes sufficient to supply its own wants without interfering with the course of the rest. It is capable of feeding any boiler working at not more than 60 lbs. pressure, so that when the exhaust steam is at atmospheric pressure, or 15 lbs. absolute, the feed-water opposes and overcomes a pressure against it of 60 lbs. per square inch.

The exhaust injector in no way affects the working of the steam engines. To start it, communication is opened with the exhaust steam and with the feed-water supply. It may be arranged to work as an ordinary injector so as to feed the boiler when the engine is standing. The temperature of the feed-water must not exceed 90° F.

The construction of the exhaust injector is different from the injector. In the former the exhaust steam enters the top of the injector and passes into a conical nozzle. This nozzle is fitted with a fixed conical spindle, the object of which is to concentrate and direct the flowing exhaust into a circular jet by the time it meets the cold feed-water, and so precipitate condensation. The feed-water enters a branch at the side of the injector near the top, and on flowing into the conical chamber surrounds the steam in its passage through the nozzle. Condensation begins as the exhaust and feed-water meet in the condensing chamber, which forms the upper part of the hinged nozzle. The hinged nozzle imparts to the exhaust injector its automatic action. When not at work the nozzle hangs back, by this means giving space for the water and steam to meet. Immediately after meeting, condensation takes place, a partial vacuum is formed, and the hinged nozzle assumes a position which reduces the space through which the combined exhaust and feed-water pass.

The exhaust injector requires no attention, and is automatic in its action, starting and stopping with the engine to which it is connected. It is very efficient where engines work without stoppages, and is said to give good results even where engines do not run continuously, but whose motion is without interruption for several minutes at a time, and if a colliery winding engine runs sufficiently long during a period of winding, it may work satisfactorily with such engine. The waste steam, in passing through the injector, heats the feed-water to a temperature of about 190°. A saving in fuel from 15 to 20 per cent. is effected by the application of this injector as compared to pumping the water into boilers.

There must be the usual back-pressure valve on the boiler; and if the delivery-pipe is long, an additional back-pressure valve must be fixed two or three feet away from the injector.

It has been successfully applied to winding engines, fan engines, and hauling engines at collieries.

The main flue into which the smoke from the boiler is discharged leads to the chimney, which is necessary to create a draught. The main flue should have an area equal to that of the chimney at its base, unless more than one main flue discharge into it. The chimneys are larger at the bottom than at the top, the sides having a taper or batter of about 1 inch per yard. They are lined with firebrick for a considerable distance from the base. To determine the size necessary for the chimney, allow 5 square feet for each boiler, and let the height be equal to 25 times the internal diameter. Chimneys are built of many forms, but the circular seems to be the best.

The following table of round chimneys is taken from *Power Steam*:—

TABLE OF ROUND CHIMNEYS. Height equal 25 times diameter.

Diameter of Flue in Inches.	Height in Feet.	Rated H.P.	Lbs. of Coal per Hour.
24	50	49	245
27	56	70	350
30	63	94	470
33	69	124	620
36	75	158	790
39	81	197	985
42	88	242	1,210
48	100	348	1,740
54	113	478	2,390
60	125	632	3,120
66	138	810	4,050
72	150	1,023	5,115
78	163	1,265	6,325
84	175	1,531	7,655
90	188	1,830	9,150
96	200	2,167	10,835

For 10 boilers it would be necessary to have the diameter of the chimney at its base about 8 feet, and 200 feet high.

Besides the boilers already described which have hitherto been most in use at collieries, may be briefly mentioned a few other forms. The breeches-flued or duplex furnace is a modification of the Lancashire boiler. In it the two furnace-tubes, instead of being continuous from end to end of the boiler, unite into one large cylindrical flue immediately behind the bridges.

The "Galloway" is a form of breeches-flued boiler. In it the large flue from the point of juncture with the furnace-tubes to the back-end plate is somewhat elliptical in section as shown in Fig. 144.

Galloway tubes are fixed three abreast and two abreast alternately throughout the length of the elliptical tube as shown in Fig. 146. As now made the central cone tubes are placed quite upright, as shown in Fig. 144, the others forming a slight angle with the perpendicular. Galloway boilers are made of steel 30 feet long and 8 feet in diameter to work at 100 lbs. steam pressure.

The particulars below are furnished by Messrs. Galloways, Limited, Manchester:—

"The following drawings from photos will explain the general arrangement and construction of the patent Galloway boiler, containing all the most recent improvements in design. This system of boiler has been in use for about 30 years, having been improved in many points of detail from time to time—the latest advance being the upward curve of the under surface of the back flue, and the radial arrangement of the cone tubes. The alteration thus effected gives more space for cleaning and examining the lower portion of the boiler, and allows the insertion of an increased number of cone tubes. The tubes themselves, moreover, are now made entirely by machinery, and with the flanges square to the centre line, which causes them to be interchangeable, and reduces the strain upon the iron in the manufacture.

"Reliable tests of the efficiency of steam boilers conducted under independent and competent superintendence are difficult to obtain, but on many occasions the Galloway boiler has been proved to give a higher result than any other in the market. We may, however, instance the official tests at the Philadelphia Exhibition of 1876, which were carried out by a special committee, and entirely without any interference or control being possible on the part of the makers of the

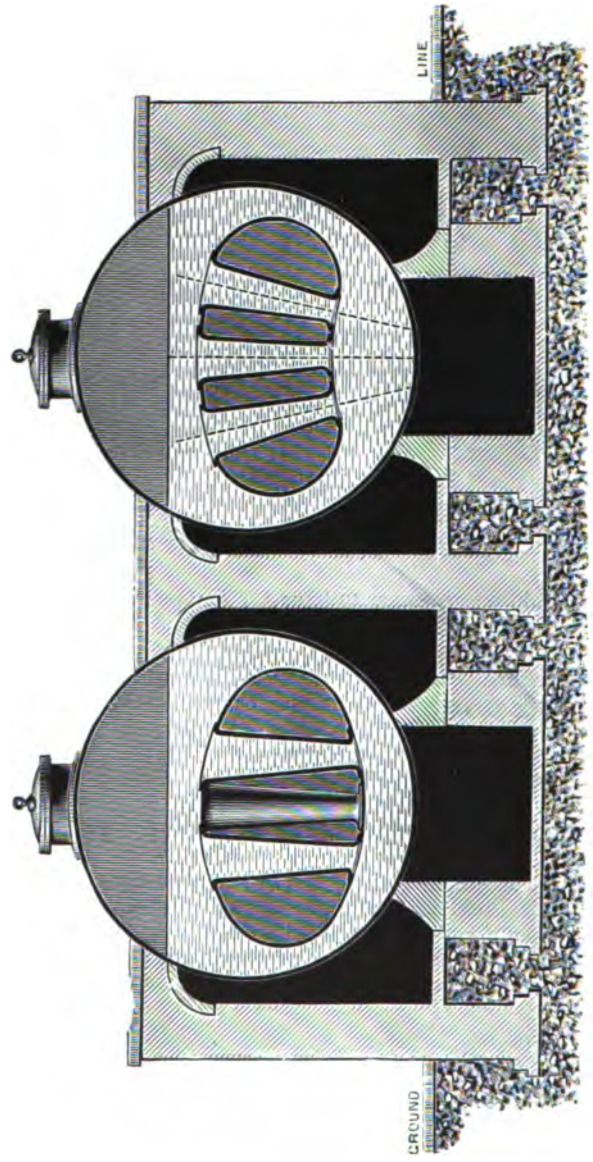

Fig. 144.—The Galloway Boiler.

As formerly made. On the latest patent.

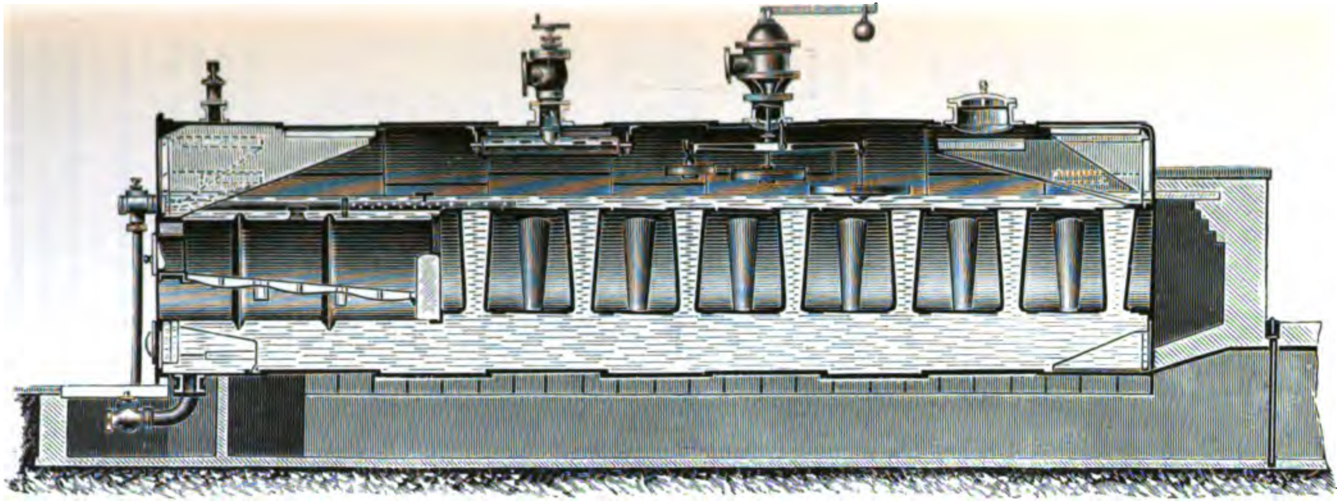

Fig. 145.—The Galloway Boiler. Longitudinal Section, with Fittings and Furnace Apparatus in Position.

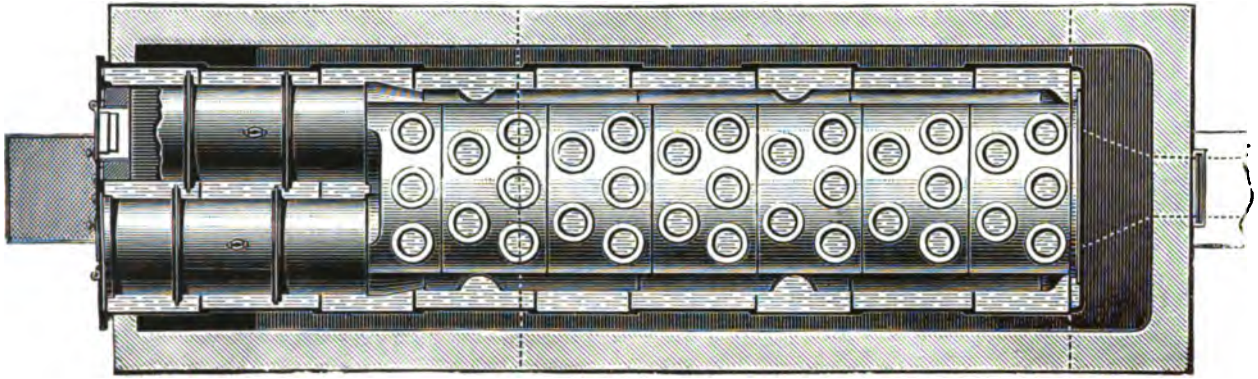

Fig. 146.—The Galloway Boiler. Plan showing Disposition of Tubes and Brickwork Bed.

boiler, where the Galloway boiler evaporated 11·72 lbs. of water, at a temperature of 212° per pound of coal; and also a test carried out at the mill of C. R. Collins, Esq., Hele, Devonshire, under the immediate supervision of the editor of *Engineering*, at which a result equal to 12·83 lbs. of water at the same temperature per pound of Welsh coal was attained. At Philadelphia, moreover, the quality of the steam generated was carefully tested, and, in addition to the highest rate of evaporation, the Galloway boiler was ascertained to give the driest steam. This is a matter of the utmost importance, as with perfectly dry steam there can be no risk of break-down occurring to an engine from that most frequent cause, priming.

"It has been ascertained in practice that a Galloway boiler is considerably more powerful than an ordinary two-flued or Lancashire one of equal dimensions, and as it burns the same amount of fuel it is correspondingly economical. A Galloway boiler 28 feet long by 7 feet diameter, when driving a condensing engine in fair order, is fully equal to 300 indicated horse-power, but in many cases one such boiler has regularly driven over 350 horse-power with compound engine, at a working pressure of 80 lbs. per square inch; this is considerably higher than can be obtained from a boiler of the ordinary construction."

Messrs. Galloways supply also the subjoined summary of official tests of boilers at the Philadelphia Exhibition, 1876, where thirteen boilers were tested for eight hours each, at a pressure of 70 lbs. to the square inch, "when the Galloway boiler gave the most economical result, and furnished the driest steam."

Description of Boiler.	Heating Surface in Square feet.	Horse-Power at 1 cubic ft. Water evaporated per Hour.	Lbs. of Water evaporated.			Per Centage of Water in Steam.	Lbs. of Coal Burnt.		Temperature of Gases leaving Boiler.	Cubic Feet of Water Space per Horse-Power.	Cubic Feet of Steam Space per Horse Power.
			Total.	Per Hour.	At 212° per lb. combustible.		Per Hour.	Per Square Foot of Grate Per Hour.			
Galloway ...	973	14·64	20824	2603	11·72	·57	283	7·269	324	14·10	4·04
Root	1590	54·29	27146	3393	11·565	not taken	381	9·09	393	2·29	·89
Fermenich ...	1078	26·46	13233	1654	11·53	not taken	185	11·79	415	4·08	2·63
Lowe	774	21·45	10729	1341	11·489	not taken	153	6·805	332	9·02	2·63
Babcock ...	1680	62·70	31358	3919	11·489	3·24	444	9·77	295	3·74	2·20
Andrews ...	540	18·94	9469	1183	10·513	not taken	148	not taken	419	4·14	1·29
Wiegand ...	1355	68·08	34042	4255	10·461	not taken	517	12·32	523	2·66	·64
Anderson ...	1135	44·44	22230	2778	10·255	not taken	350	9·747	417	1·43	1·28
Kelly	662	37·41	18710	2338	10·099	5·97	291	10·82	not taken	1·91	·68
Harrison ...	900	36·57	18285	2285	10·022	1·11	284	12·36	517	1·93	·80
Pierce	200	23·74	11876	1485	9·818	5·53	199	7·99	373	·65	1·64
Exeter ...	1525	32·65	16334	2041	9·765	4·63	280	9·35	429	2·66	1·43
Rogers & Black	399	21·13	10564	1320	9·31	2·68	181	8·05	571	1·71	1·17

In the Water-tube Boilers the flame and hot gases from the furnace act directly on rows of parallel tubes of small diameter through which the water is passed, and a receiver is placed over the tubes for the steam given off in them. They are very rapid steam generators, and like other boilers have their advantages and disadvantages.

The following description of the Babcock and Wilcox water-tube boiler is taken from the *Practical Engineer* of June 22, 1888:—

"The exhibit of the Babcock and Wilcox Co., of New York and Glasgow, consists of one of their patent water-tube boilers, as illustrated above. It has 64 tubes, 4 inches diameter and 18 feet long, with a steam receiver 4 feet diameter by 22 feet 4 inches long, and is suitable for a working pressure of 180 lbs. per square inch. It is rated at 126 H.P., and is estimated to be equal to a Lancashire boiler 30 feet by 7 feet 6 inches at 150 lbs. pressure. The rated horse-power here means 30 lbs. of water evaporated from 212°, at 70 lbs. pressure for each horse-power.

"It will be seen that the boiler consists of a series of lap-welded wrought-iron tubes, placed zig-zag one over the other, and connected at each end by vertical headers to a steam and water-receiver placed above in a horizontal position; while below, at the back end, there is a cast-iron mud chamber or sediment-collector, to which is attached the blow-off. These four different parts, when fitted together, constitute the boiler, and it is by the ingenious combination of these that the present state of perfection has been arrived at.

"The tubes are connected with the headers by being expanded into accurately bored and tapered holes, the headers again being connected by tubes to a substantial riveted block on the under-side of the water and steam-receiver. This method is stated to give every satisfaction under all pressures and conditions.

"The receiver itself is a cylindrical shell of Siemens-Martin steel, double-riveted in the longitudinal seams, and single-riveted circumferentially. The end plates form the segment of a sphere, or are what is usually termed 'dished.' This being the strongest form in which they can be made they do not require any further staying. The manhole, for internal examination, is placed in one end, and is formed by flanging the plate inwards and facing the edge, so that a metal-to-metal joint can be made.

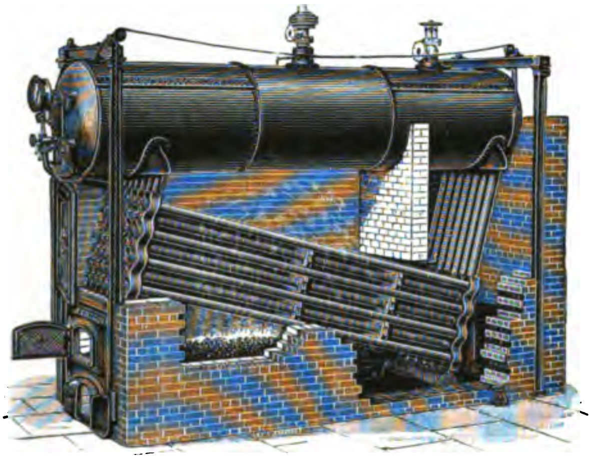

Fig. 147.—The Babcock and Wilcox Water-tube Boiler.

"The vertical headers have been usually made of cast-iron, but the firm have on exhibition a wrought-iron one, similar in design to those in use, as shown in Fig. 147. It is a beautiful piece of work, of great strength, and will considerably add to the confidence already placed in the safety of the boiler.

"The fire is placed under the tubes at the front end; the flames and heated gases rise to the under-side of the horizontal water drum, and thence pass downwards among the back portion of the tubes, and on to the chimney. There is no fear of any evil effects from this arrangement, as the boiler is suspended from above by means of hoops round each end of the steam and water receiver, and is thus entirely free to move and accommodate itself to the variable movements consequent on expansion and contraction.

"The water level is about the centre of the receiver, and by virtue of the greatest heat raising the temperature at the higher end of the tubes it circulates rapidly upwards at the front, along the chamber, down at the back, and on through all the tubes.

"Owing to the rapid circulation, the greater part of the sediment has not time to deposit itself in the tubes or headers, but is carried onward until by gravitation it reaches the lowest part of the boiler at the back, where it rests in the collector, and is blown out at intervals through the cock placed there for that purpose.

"There will, of course, be some deposit in the tubes, and this fact has not been overlooked. Opposite each tube, at either end in the vertical headers already mentioned, there is a small door, which is easily removed, and through which a scraper of improved construction is applied to the inside of the tube. This scraper, which we understand is the subject of a patent, accommodates itself to any unevenness, and is used from the front end, while the deposit is removed from the back or lower end of the tubes.

"These doors, which are also used in the construction of the boiler for expanding the tubes, have double joints, to ensure lightness and safety. The inside one has an indiarubber ring, and in case of the nut or stud breaking, or in any way becoming deranged, the pressure on the door presses it against the inside face, and so prevents leakage until a new door is got ready.

"For cleaning the outside of the tubes ample means have also been provided. In the side wall there are three long narrow doors, communicating with the different sections into which the flame space is divided. A flexible rubber hose, capable of withstanding 300 lbs. per square inch, is connected to the steam-receiver, and the wrought-iron perforated pipe on the other end is put through the doors and among the tubes; the steam jets rapidly bring down the soot. For cleaning out there are large doors, as shown in the illustration; the one at the back is also for getting at the sediment collector. The fittings consist of a stop-valve, a safety-valve, two sets of water-gauges, a feed check-valve, a blow-off valve, and an improved pressure-gauge, on the Schäffer and Budenberg principle. Great attention has been given to the delivery of dry steam to the engine—a point of importance well known to all engineers. The absence of priming in this boiler makes the use of any special apparatus unnecessary; the only precaution to be taken is to place the stop-valve on the opposite end of the steam-receiver to that over the fire. The steam is, as it were, made at one end and discharged at the other, so that by the time it has traversed the receiver all particles of water in suspension have fallen out of it, and the steam is in a fit state to be conveyed to the engine without any further drying. We may say while referring to the steam space that, excepting under exceptional circumstances, the old idea of having a dome, or other storage for the steam, other than that contained in the boiler-shell proper, is quite wrong, and so long as ample water surface is provided for the steam to disengage itself easily, nothing further is required.

"So much for the construction and design of the boiler, and from what we have said it will be seen that the manufacturers have paid every attention to these particulars, even in the minutest details, and from the success that has attended their efforts to produce an efficient and safe steam generator, they have evidently not overlooked the many important features inseparable from efficiency and economy. The ratio of heating surface to fire-grate, and these two combined to the steam space, is based on actual experiments and nearly 20 years' experience. The result, as embodied in practice, is to be found in the continual efficiency of the boilers at work.

"The following figures are taken from records of trials that have been made:—

"In 1879 a trial of two boilers, 360 H.P., was made in America, when, from careful observation, the rate of evaporation was found to be at the rate of 9·2 lbs of water per lb. of coal at a pressure of 70 lbs., and from a temperature of 180° with 12 per cent. of refuse.

"In 1883, at another trial in Pittsburgh, 10·9 lbs. of water were evaporated from each pound of bituminous coal—less 11 per cent. of ash—at a pressure of 70 lbs. from and at 212°.

"Again, during trials at San Francisco, with three different kinds of coal—viz., British Columbia, Welsh, and Washington—there were 11·1 lbs., 11·8 lbs., and 10·4 lbs. of water evaporated per pound of pure coal respectively. Efficiency trials have also been made in this country, and we extract the following from the company's report, dated July and August, 1885, regarding tests at the Gaythorn works of Messrs. J. and J. M. Worrall, dyers, Manchester, with a 140 H.P. boiler:—

	No. 1.		No. 2.
Heating surface	1,616 sq. ft.	...	1,616 sq. ft.
Grate surface, 6 ft. 6 in. by 4 ft. 6 in.	29 sq. ft.	...	29 sq. ft.
Ratio of heat to grate surface	55 to 1	...	55 to 1
Duration of test	27 hours	...	18½ hours
Average observed steam pressure	75 lbs.	...	95 lbs.
Average temperature of water fed to boiler by injector	135°	...	275°
Pounds of coal fired	25,536	...	11,704
Coal consumed per square foot of grate per hour	31·7	...	21·6
Total water evaporated	216,000 lbs.	...	118,400 lbs.
Water evaporated per hour	8,007 lbs.	...	6,400 lbs.
Water evaporated per pound of coal—actual conditions	8·46	...	9·85
Rated H.P. (one H.P. = 30 lbs. of water evaporated from 212° at 70 lbs. pressure).	140	...	140
Temperature of flue gases	700°	...	680°

"In trial No. 1 the 27 hours include three stoppages, aggregating 2½ hours, in which but little steaming could have taken place, so that the figures represent ordinary working conditions.

"From the careful manner in which the reports of the various tests of this boiler have been drawn up, it would appear that great care has been exercised to arrive at correct and reliable results."

Arnold's boiler resembles an ordinary Lancashire or Cornish boiler, the chief points of difference being in the construction of the internal flue and the substitution of a long tube for the Galloway tubes, the tube extending from the inside of the firebridge, with a slight inclination inwards, to a point near the back end of the boiler. The long tube is connected to the internal flue at the top near the fire-bridge and at the bottom near its other extremity, and so forms a passage for the water. This, it is said, improves the circulation and adds to the heating surface of the boiler. Each ring forming a part of the flue, instead of being of a uniform diameter, is bulged into the shape of a barrel, the object being to impart greater strength, and increase the heating surfaces of the fire-boxes of steam boilers.

Vertical boilers are useful where only a small amount of power is required, but they are not economical steam producers. There are many varieties of vertical boilers; they are generally fitted with either Galloway tubes or parallel tubes to improve the water circulation in them, and also to facilitate the escape of steam.

The truest economy results from in the first place obtaining boilers of the very best construction, and due attention having been paid to their seating, then giving constant attention to their preservation. It has been suggested that the working pressure of boilers should be reduced at regular intervals of time in proportion to their advanced age, and that some limit of age should be fixed beyond which they should not work at all. There are many objections to such an arbitrary course of procedure. The great difference in the quality of material used in the construction, and in the mode of setting a boiler as well as the subsequent care or neglect, render it impossible to frame rules which would be applicable to all boiler users without injustice to some. Boilers no doubt deteriorate with age, but in many instances practices are allowed to proceed which seriously increase the rate of deterioration and ensure a speedy destruction of the boiler. Where well constructed, well seated, and tended by careful and skilful men they have been known to work as long as 30 years at but trifling costs for repairs.

Again, it has been urged that as a safeguard against explosions, boilers actually at work should receive at regular intervals a similar hydraulic test to that applied when new. In some cases it may be beneficial to use the hydraulic test to a working boiler, as for instance immediately after it has undergone repairs and before re-starting work. All defects that are visible should be found by periodical inspection to which undoubtedly boilers should be subjected, but it is questionable if good results would follow an application of the hydraulic test used to search for defects which exist but are hidden. Its effect *may* be only to increase the defects without detecting them, and consequently the boiler on being set to work again with its weaknesses intensified, is in a worse state than before, and may have suffered so much from the treatment as to be unable to stand the usual working pressure and so explode on its being attained.

Only a good maker using the best materials will ensure the production of a reliable boiler, designed so as not to have too much rigidity in one place, or too great elasticity in another. When the fire is first lighted in the furnaces the front end may be seen to come out as much as ¾ of an inch from its position when cold. The furnace crowns also rise up as much as ½ an inch from the expansion, and if proper provision has not been made in a Lancashire or Cornish boiler for the elasticity of the ends, what is known as *grooving*, *channelling*, *furrowing* or *guttering* round the furnace-tube angle-irons may begin.

Grooving is the eating away of boiler-plates in lines or furrows, and is caused by the mechanical action proceeding at the point of grooving, assisted by the chemical action of the feed-water. In the lap-joints (and at the angle-irons) which are subject to much bending and straining motion, grooving frequently takes place, the plates being corroded in a line parallel to the joint and just at the beginning of the lap. The effect of grooving is to weaken the plates, and frequently to fracture their ends about the rivet-holes.

Expansion and contraction are constantly present in a working boiler, and as they cannot be controlled, provision must be made in the construction of the boiler to meet the varying temperatures, or the mechanical action resulting therefrom may cause fractures.

If Lancashire boilers are seated and protected against radiation of heat in the manner already suggested it will tend materially to their preservation.

Then the manner in which a boiler is worked and tended, the quality of the feed-water used, and its treatment, are important factors affecting the durability of the boiler.

A boiler attendant should be a man of intelligence who knows something of the construction of that which is placed in his charge. Careless, untrained, and underpaid attendants ruin good boilers in a variety of ways: for instance, by neglecting to feed the furnaces, thus allowing the fire-bars to get bare, a passage being thereby formed through which the cold air passes; by driving the furnaces too much and generating steam rapidly, afterwards opening the furnace doors in order to prevent blowing-off at the valves; by using means when being laid off to quickly cool the boiler instead of waiting till the brickwork is cool before running the water out, and allowing time for the boiler to cool gradually; by filling or partly filling it with cold water immediately after it is emptied; and on the boiler resuming work, by forcing the fires and raising steam more rapidly than should be attempted from cold water. Intense firing should specially be avoided in boilers having thick plates.

Care is required in hand-firing boilers, otherwise much smoke is emitted. The fuel should be introduced frequently and in small quantities which must be spread thinly and evenly on the front portion of the fire-grate, and at intervals the hot fuel pushed back. No portion of the grate should at any time be bare so as to form an air-passage through the fire.

Every boiler should be thoroughly emptied of its contents periodically, say once a week or once a fortnight according to the water used, and afterwards be filled by an entirely fresh supply.

All natural water contains more or less of foreign matters, and none, therefore, is ever pure. The impurities may be gaseous, mechanical, mineral, or organic; one class will predominate in one locality and another elsewhere. The purest water obtained is rain-water, but even that is charged with the atmospheric gases oxygen, nitrogen and carbonic acid gas. In 100 parts of water there are about 2 parts of air, but the amount varies slightly. Next to rain-water in point of purity is that which flows over or through soil which is not easily dissolved, or water which receives no other soluble matters than those washed down by the rains, such as that of mountain torrents, and lakes on rocky surfaces of slate or granite.

Certain gaseous impurities in waters result from the strata through which they flow; such are carbonic acid and hydrogen sulphide. Part of the rain which falls upon the surface of the earth is evaporated, another portion finds its way directly to the surface streams, whilst the remainder sinks into the earth's crust, to issue afterwards at the surface in the form of springs. Where the course taken by the water in following the natural fissures is tortuous and long, it may pass through rocks of varying character and composition. Water has great solvent power, and in flowing through certain strata becomes charged not only with mineral matter, but also often with gas; as for instance, the waters of Bath, Harrogate, Llandrindod, &c., which have dissolved hydrogen sulphide.

Mechanical impurities are those mixed with and suspended in the water arising from its passage through or over the earth. They consist of clay, chalk, sand, &c., which are insoluble in water. They affect the purity of the water, but can be removed by subsidence or filtration.

Dissolved mineral or chemical impurities in waters result from their passage through or over the earth, and are those most commonly found. They consist of carbonates and sulphates of lime, and carbonates of magnesia, potash, and soda, chloride of sodium, &c. Water impregnated with iron is said to be chalybeate. The presence of the salts of lime and magnesia renders water hard. Sulphuric acid, arising from pyrites, seriously corrodes the metal of pipes, rails, and pumps, and rots the leather of boots.

Organic impurities in water arise from animal and vegetable organisms, either living or dead.

Glasgow is partly supplied with water from Loch Katrine, which in point of purity compares favourably with that of any city in the world, while the water supplied to London is in great part obtained from the river Thames and contains nine times as many impurities as that from Loch Katrine. The water used for domestic purposes in the town of Pontypridd is for the most part derived from a mountain torrent, and shows on analysis about double the impurities of the Loch Katrine water.

The water available for boilers should be first submitted to careful analysis by a competent chemist, and if found to be of good quality, it may require no treatment. But if certain impurities exist, efforts must be directed to neutralise their effect, and these must be guided by the analysis.

The effect of supplying boilers with water containing only mechanically suspended impurities is to have a deposit of mud formed in the boiler, and although this is not so objectionable as an incrustation, the water should not be allowed to enter the boiler, unless some means are afterwards taken to remove it before the sediment forms a deposit. Where no provision is made to purify water of this description either before or after it enters the boiler, it necessitates frequent blowing off and cleaning. If no other impurities are present in it, the water may be greatly improved by being filtered before it enters the boiler, as water percolating through sand is divested of much of its mechanical impurities.

Waters which contain certain mineral impurities require chemical treatment in order as far as possible to prevent them from forming deposits within the boiler, and from wasting the plates by internal corrosion. The amount of cleaning necessary in the boiler will depend on the success of the chemical treatment.

An incrustation formed in the boiler interposes a bad conductor of heat between one side of the boiler plates and the water at a point where the other side of the plates is exposed to the fire. This leads to considerable waste of fuel, and to danger of water coming into contact with an over-heated plate owing to cracking or loosening of the scale.

Carbonate of soda may be added to water which contains sulphate of lime, which will then be converted into carbonate and form a soft scale which may be easily removed.

Some benefit is derived, where hard water is used, by frequently blowing off the water and so getting rid of matter which if allowed to remain would form into scale. This method is wasteful as compared with purifying previous to heating the water.

Carbonate and sulphate of lime and carbonate of magnesia form the larger part of ordinary scale found in steam boilers.

The salts of lime and magnesia in the form of carbonates are easily dissolved in water which contains carbonic acid gas. In water raised to the temperature of 212° F. carbonic acid gas is driven off, and the maintenance of the same or an increased temperature is followed by the salts being deposited on the interior of the vessel in which the experiment is conducted. In steam boilers the result of such a deposit is to form scale.

If a similar experiment be tried with water which contains sulphates of lime or magnesia, it will be found that the salts will not be deposited on the water reaching a temperature of 212° F., and it is not until the water has undergone much evaporation that it loses its power of holding the sulphates in solution, after which they are precipitated on to the surface of the vessel as were the carbonates at a lower temperature.

A number of anti-incrustation compounds are manufactured for the purpose of preventing the water impurities forming into a scale on the boiler whilst the compositions do not themselves injure the plates. The addition of a compound to act chemically after the water is in the boiler, no doubt in some instances renders the deposit formed in the boiler softer and therefore easier removed, but to be effectual the composition must contain elements, specially arranged to combine with the impurities in that water wherein it is placed. It is impossible therefore for any one composition to eliminate all kinds of impurities from water.

Elaborate and more costly methods of purifying water—such as the "Porter Clark" or the "Stanhope" water purifier, &c.—may be adopted for removing the salts of lime and magnesia from the water before it is passed into the boilers.

A simple and most efficient apparatus for the purification of water is that invented by Messrs. Archbutt and Deeley, and manufactured by Messrs. Mather and Platt, of Salford. Owing to the special method of manipulation adopted, by which the precipitate is caused to settle very rapidly, relatively large volumes of water can be softened and clarified in plain tanks of moderate size, and the necessity for filtering appliances is done away with. Two tanks, placed side by side, are generally used, the processes of filling, adding the chemicals, and allowing the precipitate to settle being carried on in one, whilst the softened and clarified water is being drawn off from the other, but for smaller quantities of water one softening tank and a small storage tank only are required. Fig. 148 shows an end view and a plan of one tank. A is a valve through which the tank is filled with hard water up to the dark dotted line. While the tank is filling, lime and alkali, in suitable proportions depending upon the character of the water, are boiled with water by a steam jet in the small chemical vessel on the platform. When the tank is full, and the valve A has been closed, steam from the boiler is admitted to the blower B, causing a current of water to circulate through the rose R, the three-way cock C, down the vertical pipe P, and back into the tank through the perforations in the *upper* row of horizontal pipes. On opening the small tap T, the prepared chemical solution is slowly drawn into the circulating current and thus diffused throughout the body of water in the tank. The cock D is next opened to admit air through the pipe at the top of the blower, and by reversing the three-way cock C, this air is forced through the perforations on the under side of the *lower* row of pipes. From these it rises in streams of bubbles, powerfully agitating the water, and stirring up the precipitate or mud from previous operations which lies on the bottom of the tank. Before being mixed with this mud, the new precipitate, which is very finely divided, will not coagulate, and takes a long time to settle; but, when the old precipitate is stirred up and mixed with it, the two together settle rapidly, and leave the water practically clear and ready for running off in from thirty minutes to one hour.

The chemicals, however, have been added to the *cold* water, and it is found that when such water has been softened and afterwards becomes raised in temperature in the hot pipes, injectors, &c., connected to boilers, a further slight precipitation occurs, which gradually forms a coating on the interior of the pipes, &c. To prevent this, the water is carbonated, which has the effect of rendering permanently soluble the trace of hardening matter retained by the water. The carbonic acid is supplied by a coke stove, containing enough fuel in the cone to last for several hours; and the gas from this stove is forced, by a small steam blower, into the upper end of the hinged pipe H, and is carried down with the water into the storage tank. Baffles fixed in the pipe cause a thorough mingling of the gas and water, and thus the carbonating is simply and easily effected. The upper end of the draw-off pipe H, where the gas enters, is bent over to form a syphon, and when the ball valve V closes, the bend fills with gas. It is not until the pressure of this gas is relieved by the opening of the ball valve that the water can pass into the pipe, and when the valve opens sufficiently to allow all the gas to pass, then the water falls down the pipe with it and is carbonated.

The precipitate is prevented from unduly accumulating in the tank by its partial removal at regular intervals; this is effected in various ways, to suit different circumstances. From the nature of the precipitate it is easily dealt with. In small plants, a discharge pipe, connected to a drain, is provided in one corner of the tank, and by lifting a plug daily, a sufficient quantity of mud can be run off. In larger plants, the mud is swept out, at longer intervals, through mud doors, into a trough which conveys it on to waste ground; or, it is raised out of the tank or trough by a steam lifter, and discharged into a cart lined with waste furnace ashes, through

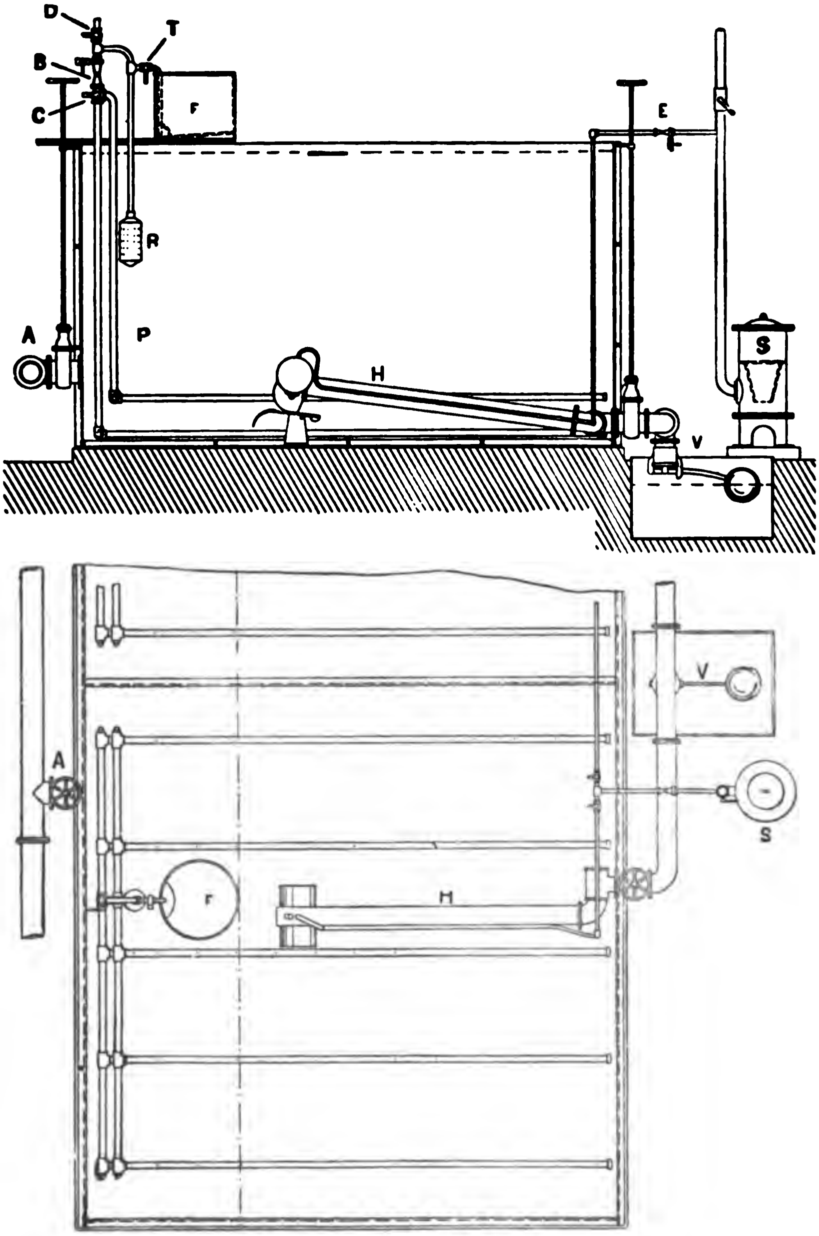

Fig. 148.—The Archbutt-Deeley Patent Water Purifier.

which the excess of water readily drains, leaving the nearly dry mud and ashes ready for tipping.

That steam boilers, and their feed apparatus and economiser tubes, can be kept quite free from scale by the above process of softening the water, has been amply proved at several works where it has been adopted. The largest quantity of water under treatment by it is at the locomotive works of the Midland Railway Company at Derby, where a plant softening 30,000 gallons per hour has been in successful operation since 1891. The hardest water at present (1895) being treated is at a mill in Nottingham; it has 35½ degrees of hardness, and contains both carbonate and sulphate of lime, as well as a large quantity of magnesium salts; by the treatment, the hardness of the water is reduced to an average of 3·2 degrees and the formation of scale is entirely prevented.

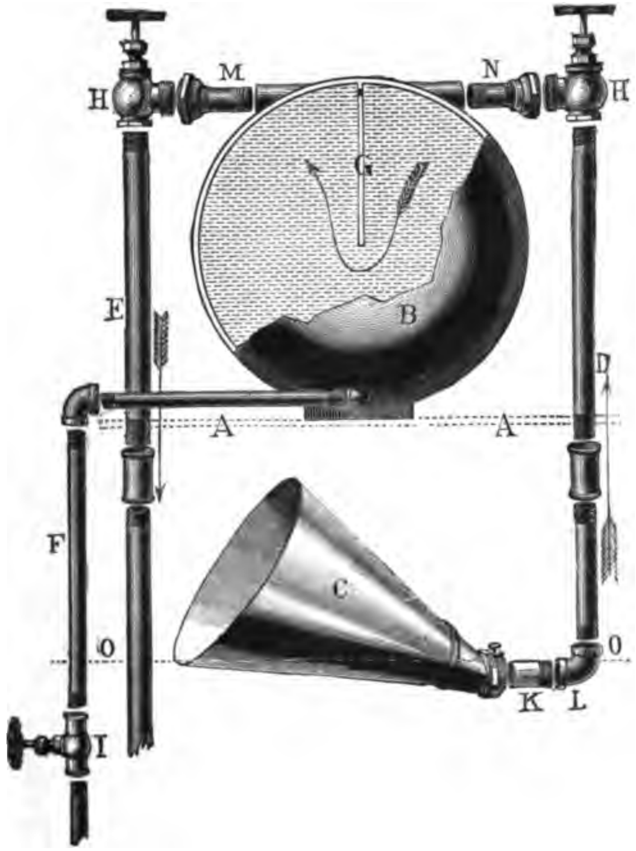

Fig. 149.—The Hotchkiss Boiler Cleaner.

Where the feed-water is of a muddy description the *Hotchkiss boiler cleaner*, Fig. 149, as described by Mr. Horatio Nelson, 90, Worship Street, E.C., is said to be very effectual in removing scum and the floating deposits in the water.

In Fig. 149 the double dotted line AA is the top of boiler shell, which usually is found to be the most convenient position to place the reservoir. The reservoir is connected with the funnel C by the up-flow pipe D, and to a lower part of the water in the boiler by the return pipe E. Through these pipes and the reservoir a circulation is established which flows in the direction shown by the arrows. The funnel C is set within the boiler on the low-water line, as indicated by the dotted line OO. G is a diaphragm in the reservoir to divert the flow of water therein. F is the blow-off pipe, for removing the deposits B from the reservoir. HH are two valves, used to shut off reservoir from boiler if required. Angle valves, as shown, are generally used to simplify the connections. I is a valve on blow-off pipe F. K is a socket nipple, secured by thumb-screw. M and N are nipples, each with half-union to complete the connection.

This cleaner is automatic and, from natural causes, certain in its action.

The manner in which the "cleaner" acts in removing sediments from and preventing scale formation in steam boilers, is as follows:

As soon as the water in a steam boiler becomes heated, currents are established; these currents are formed by the hotter, and therefore lighter, water flowing upward and away from the source of the greatest heat, while the colder, more dense water flows to the source of heat, to replace the other, and in its turn becomes heated.

In all boilers where fire is applied at one end, the currents established will be upward and from the fire on the surface, and downwards and towards the fire in the lower part of the boiler.

In a boiler with the cleaner attached, the funnel C is set near the surface of the water, but partly submerged, and in such position that its opening will intercept the currents of hot water flowing towards it.

By the syphon action of the apparatus the hot surface water containing any floating matter that enters the funnel is caused to pass through the up-flow pipe D into the reservoir, thereby displacing the cooler water therein, which flows back to the boiler through the return pipe E, which terminates at a lower level than the funnel.

By this means, so long as firing is kept up, a constant circulation is maintained and all the water in the boiler is caused to pass through the reservoir. The water while in the reservoir is comparatively quiet, and entirely free from the agitating currents within the boiler, and while in this quiescent state the contained sedimentary matter B is precipitated, and remains in the reservoir, from which it can be blown as often as necessary by the pipe F.

The action of this apparatus creates a current in the boiler, thus utilizing the usual cold strata of water lying under the flues, and which is useless for creating steam.

Every one familiar with steam boilers is aware that deposits and incrustations seek naturally the quietest part of a boiler. The office of the Hotchkiss mechanical boiler cleaner is, therefore, simply to provide a place for their accumulation, outside of the boiler itself, and removed from heat and its agitating effects; from whence they can be readily removed as fast as they accumulate, instead of shutting down the boiler to clean them out by hand, or by blowing down the boiler in the ordinary way, thus losing a large amount of water already heated to the steaming point, which is wasteful as to fuel, and also a great loss of time.

It is not claimed for the "Hotchkiss mechanical boiler cleaner" that it will remove scale bodily from the boilers when the scale is already formed, but it is claimed for it, and *guaranteed* that it will prevent the formation of new scale by removing all the floating deposits and mineral salts, which become scale if not removed from the water before they have had time to adhere to the heating surfaces.

By preventing the formation of new scale, the old, by expansion and contraction of the heating surfaces, soon becomes loose and readily detached.

A simple means of purifying the feed-water before it enters the boiler is by *Seale's patent water purifier,* shown in Figs. 150–153, as applied to a

Lancashire boiler. Fig. 150 is a front elevation, Fig. 151 a sectional elevation, Fig. 153 a sectional plan, and Fig. 152 a detail view, showing the mode of carrying the pipe.

The feed-water enters through the feed valve at the front end, and traverses a length of about 50 feet of 2-inch piping, arranged in the form of a horseshoe. This pipe is enclosed within a larger one of about 8 inches in diameter, suspended near the working level of the water, shown in Fig. 151. The 2-inch pipe terminates slightly before the 8-inch pipe and the feed water passes through the

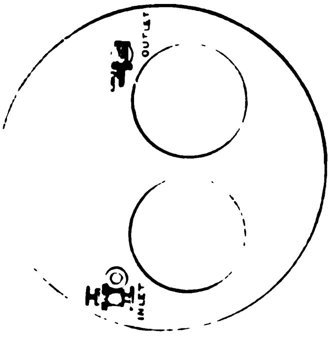

Fig. 150.

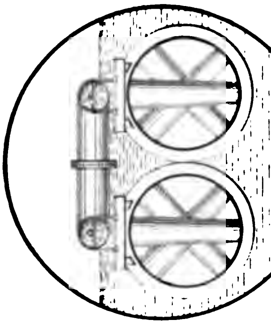

Fig. 151.

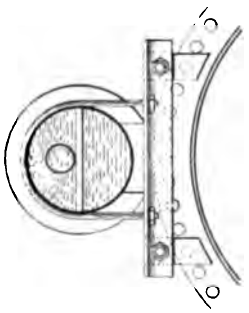

Fig. 152

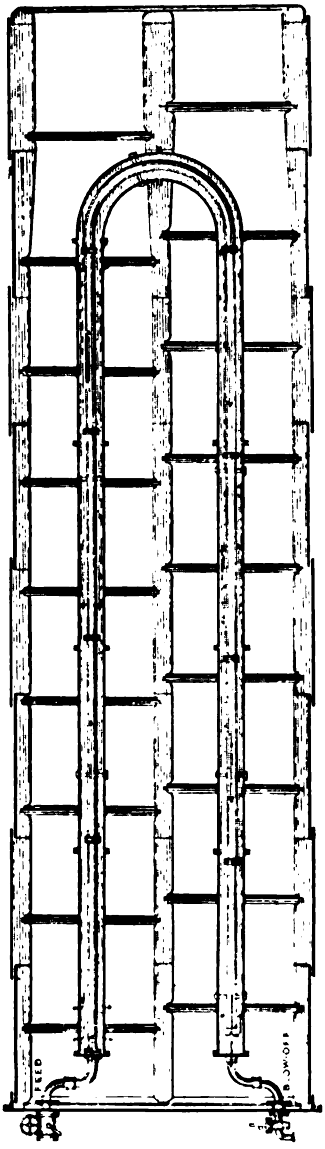

Fig. 153.

Seale's Patent Water-Purifier.

smaller pipe, and returns through the annular space between the two pipes along its former course to near its starting-point, where, as shown by the arrows, it is discharged through slots into the boiler. By this means the feed-water becomes heated practically to the same temperature as the water in the boiler before it mixes with the latter. In its course through the pipes, as it reaches 212° F., carbonic acid will be driven from the water, and the carbonates of lime and magnesia be liberated. When a temperature of 300° F. is attained, the sulphates will also be liberated. The water is reckoned to reach this temperature just after it emerges from the open end of the internal pipe; and as the outer pipe is larger the velocity of flow is correspondingly reduced, so that the annular space between the two pipes acts as a settling tank, in which the particles of lime and magnesia are deposited, along with any other impurity the water may contain, as the water slowly travels to its exit.

The removal of the accumulated deposit is effected by means of a blow-off tap in front of the boiler, as shown on the drawings. This is used at intervals, the interior of the apparatus scoured, and the deposit thus prevented turning to a hard scale. The apparatus can be applied to any existing boiler.

In *Sanderson's patent feed-water purifier and heater*, advantage is taken of the fact that when water is heated beyond a certain point, it becomes incapable of longer holding the sulphates or carbonates of lime in solution, and further, if the salts are separated in the body of the water, they settle to the bottom of the boiler in a fine, floury state, easy of removal by blowing off or washing out, instead of being deposited in hard adherent scales by a process of slow chemical accretion on the surface of the plates. This is accomplished by introducing the feed-water into the steam space by means of a diffuser made of gun-metal. It is fixed near the crown of the shell, the feed-pipe being taken in the first place through the length of the boiler in the water space. By this means the water is partially heated, and on reaching the diffuser falls from it through the steam space in the form of a fine spray, and enters the body of the water in the boiler at practically the same temperature.

Boilers, for their proper preservation, should be placed under the periodical inspection of experts. An experienced inspector, skilled in the detection of the defects to which boilers of different types are liable, may detect early signs of deterioration, before an ordinary engineer would note anything amiss. These defects may then easily be remedied before much harm is done, and at a trifling cost. Any alterations that may from time to time become necessary or advisable, from the advancing age of boilers, such as a diminished working steam pressure, or unfitness for further work, will be reported by the expert.

Boilers are subject to explosion from a variety of causes. Defective material is a frequent cause. There may not be the necessary ductility or tensile strength in the plates, or the plates may have blisters in them.

Boiler plates whether of iron or steel are liable to blisters which are formed during the process of manufacture, but they proceed from different causes.

Those in the iron plates arise from pieces of slag or cinder getting between the layers whilst the manufacture proceeds. The slag is hammered and rolled with the iron, each process it goes through enlarging the superficial area over which it extends. Blisters weaken the plate, the weakening effect being greatest where the blister is thickest. They have been observed as large as 2 feet in diameter, and may be any smaller size. Where the piece of slag is very small, its effect may only be to cause a lamination in the plate, but where of serious size the blister may attain a thickness of $\frac{1}{4}$ of an inch.

In manufacturing steel plates, all slag rises to the top of the ingot when the metal is in a molten state, the end afterwards being rough and uneven. The after treatment of the ingot consists in hammering and rolling, and results in intensifying the irregularity of the one end. Neglect to cut off this end and separate it from the rest, results in lamination in those plates made from the top end of the ingot. The lamination may not be suspected unless attention be called to it by cutting testing strips from it, or unless the plates require flanging in the boiler-shops. If the latter operation is resorted to the lamination is at once apparent, as there is then some difference in the position of the ends of the two pieces caused by the bending, as shown in Fig. 154, but the two portions of plates will always be in close contact with each other.

A defective safety-valve may cause an explosion; the valve may be overloaded or out of order, or may stick in its seat.

Surcharged steam is another. This occurs when the water gets below the level of the flues, or where the flame by any means heats the brickwork above the water line, causing a great addition to the temperature of the steam with little alteration of the indicated pressure.

Explosions have occurred where, while cleaning the boiler, wooden plugs have been driven into the branch, at the top of the boiler, common to both the stop-valve and the only safety-valve on the boiler. The plug has been driven in to prevent the entry of steam from other boilers of a range into the one requiring cleaning through a leaky stop-valve, and on re-starting the boiler, the plug has, through forgetfulness, not been removed. Consequently, as the steam is raised in the boiler and has no means of escape, on attaining sufficient pressure it bursts the boiler.

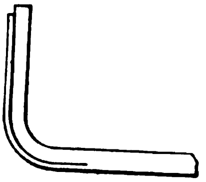
Fig. 154.—Laminated Steel Boiler Plate.

To guard against such occurrences, safety-valve mountings should invariably be connected with separate and independent branches, and to guard against one being out of order, two should be placed on a boiler.

Accumulation of deposit is another possible cause of explosion. The impurity of the feed-water produces this as already explained. The mineral salts in solution cannot pass away with the steam, and the water becomes so saturated that it deposits a portion as a crust on the inner surface. This deposit often takes the form of a hard scale, and becomes firmly attached to the boiler plates. Owing to unequal expansion this scale may get separated from the plates, and the water then rushing in between them generates a considerable charge of highly elastic steam, which may result in an explosion.

Continual wear and tear may cause an explosion where boilers, too small for their work, are subjected to excessive firing resulting in injury to the plates. In time the boiler may be so weakened as to explode at the ordinary working pressure.

Deficiency of water is another cause of explosions. This may be brought about by negligence, or by relying on a single water-gauge which has got out of order.

However correctly a boiler may be seated, deficiency of water in the boiler causes the plates at the top of the side-flues, in Cornish and Lancashire boilers, to be exposed to the direct action of the fire, resulting in damage to the plates from the overheating and in the generation of steam at such pressure as may burst the boiler.

Fracture of plates and angle-irons, produced by unequal expansion and contraction, causes explosions.

Corrosion, either internally from the impurity of the water supplied, or from the grease and oil sent into the boiler with feed-water which has been heated by [illegible] externally from leakages at the joints, may cause explosions, [illegible] corrosion may result from contact with damp brickwork at the [illegible] The effect of corrosion is a wasting away of the plates, [illegible] weaker and weaker as the corrosion continues. Internal corrosion [illegible] It may cause clear and well-defined pit-holes, or honey-[illegible] close in appearance, or a large extent of plate surface may [illegible] as to defy detection unless the plates be drilled. Instances [illegible] where corrosion has reduced the plates over the seating-walls to [illegible] thick, and even less; if boilers, on examination, are found with [illegible] it is easily seen how little force is necessary to rupture them.

[illegible] which has already been described, causes explosions.

[illegible] have doubtless exploded from old age.

[illegible] hand boilers, bought cheap, which may have lain about and [illegible] are erected—very soon to prove an expensive bargain.

[illegible] and ill-constructed boilers, even when made of good materials, [illegible]

[illegible] frequently result from faulty construction, and explosions may [illegible] improperly fixed water-gauges or cocks, their position being too high [illegible] on the boiler.

[illegible] which some boilers are subjected to when undergoing repairs may [illegible] explosion on being again set to work. A thoroughly well-designed [illegible] boiler may be placed, for instance, in careless or ignorant hands [illegible] in those of a practical working boiler-maker whose practice is [illegible] making tight joints, he being unable to calculate the sizes of material [illegible] to resist strains or pressures. Again, in removing Galloway tubes [illegible] boilers, blank flanges are substituted instead of making some [illegible] to replace the sustaining strength to the flue of which it was robbed, [illegible] a collapsed flue on re-starting. The various kinds of expansion-[illegible] furnace-tubes get removed for repairs, and are replaced by flat belts [illegible] allow expansion and contraction to proceed. The gusset stays get [illegible] from the flat ends of boilers so as to enable a leakage to be stopped. [illegible] to compensate for the diminished strength is provided. A leaky [illegible] shell may be removed, and one considerably thinner used in the [illegible] thereby to produce an explosion.

[illegible] to workmen engaged in cleaning boilers has often resulted from such [illegible] practices as the following. To prevent annoyance from leaky [illegible] plugs are sometimes driven into the [illegible]

[illegible]

An economical method of warming the surface-buildings about a colliery is by means of steam-pipes laid from the boilers through the buildings. The amount of steam passing may be regulated by a cock. The steam should not be used for any other purpose, such as driving an engine, for the passage of the steam through the buildings in the summer months would make the buildings unbearably hot. Where it can be applied it saves a considerable amount of trouble in attention to fires, grates, &c.

A few rules relating to boilers will now be given.

The material for boiler-plates, rivets or stays, should have a tensile strength of at least 20 tons per square inch for boilers intended to be worked at ordinary pressures up to 60 lbs., but should have more for higher pressures.

Rule to find the Load on a Safety-Valve.

$\frac{\text{Whole weight on valve}}{D^2 \times \cdot 7854}$ = pressure per square inch, and the whole weight = the weight of the ball × the leverage, or the ratio of the length between the fulcrum and the valve, to the whole length of the lever. Therefore to graduate a lever for any desired pressure we have $\frac{\text{area of valve} \times \text{pressure}}{\text{leverage}}$ = required weight of ball, and consequently for any lower pressure with the same ball the leverage will be $\frac{\text{area of valve} \times \text{pressure}}{\text{weight of ball.}}$. For strict accuracy the weight of the lever also requires to be taken into account, that is, its whole weight acting at half its length and reckoned as part of the ball.

Rule to fix the Area of the Valve.

Area of valve in inches = weight of water evaporated per hour × ·005. Two valves each having half the area given by the rule are preferable to one large one.

The weight of steam in lbs. discharged per square inch of opening per hour through the safety-valve is:—Absolute pressure × 50.

A Lancashire boiler evaporates, as a maximum, about 200 lbs. of water per hour per square foot of grate; and a Cornish boiler, in which the ratio of heating surface to grate surface is less, a maximum evaporation of 150 lbs. per square foot of grate per hour. A Cornish boiler with a 3 feet 6 inch internal flue and a fire-grate 6 feet long would have 21 square feet of grate; a Lancashire boiler with the same length of grate, but with two 2 feet 9 inch internal flues, would have 33 square feet of grate.

Molesworth gives the following rules for land boilers:—

For each nominal horse-power a boiler requires,—

1 cubic foot of water per hour.
1 square yard of heating surface.
1 square foot of fire-grate surface.
1 cube yard capacity.
28 square inches flue area; 18 inches over bridge.

For cylindrical double-flued boilers an approximate rule is, $\frac{\text{length} \times \text{diameter}}{6}$ = nominal horse-power.

Rules for Heating and Grate Surfaces.

G = Fire-grate surface in square feet.
H.P. = Number of nominal horse-power.
h = Heating surface in square yards.

$$\text{H.P.} = \sqrt{h\,G}.$$

$$h = \frac{\text{H.P.}^2}{G}$$

$$G = \frac{\text{H.P.}^2}{h}$$

The maximum safe working pressure for well-made boiler shells may be found by the following rules:—

Iron.

Single riveted
$$p = \frac{12{,}500 \times t}{d}$$
$$t = \frac{p \times d}{12{,}500}$$
$$d = \frac{12{,}500 \times t}{p}$$

Double riveted
$$p = \frac{15{,}600 \times t}{d}$$
$$t = \frac{p \times d}{15{,}600}$$
$$d = \frac{15{,}600 \times t}{p}$$

where p = the pressure of steam in the boiler, t the thickness of plate in inches, and d the diameter of the shell in inches. For exceptionally riveted joints it will be better to take 5 tons as the working strength of the plate, and then by reference to the table given, the proportion of strength of joint to the solid plate can be calculated.

A steel boiler may be worked up to $\frac{1}{5}$th higher pressure than an iron boiler, and the following rules will apply for—

Steel.

Single riveted
$$p = \frac{15{,}000 \times t}{d}$$
$$t = \frac{p \times d}{15{,}000}$$
$$d = \frac{15{,}000 \times t}{p}$$

Double riveted
$$p = \frac{18{,}750 \times t}{d}$$
$$t = \frac{p \times d}{18{,}750}$$
$$d = \frac{18{,}750 \times t}{p}$$

Frequently the tensile strength of good boiler plates of iron is taken at 21 tons per square inch of section, and the working pressure of a boiler ascertained by assuming a factor of safety of 6. By some it is considered that the factor of safety should not be much higher than 4; that 6 is too high and does not agree with the practice of the Insurance Companies, whilst the above rules do. The pressures must however be regarded as a maximum, not to be exceeded.

Suppose it is required to find the thickness of iron plate necessary for a Lancashire boiler 7 feet 6 inches diameter which is required to work at a pressure of 70 lbs. per square inch. The longitudinal seams are to be lap joints double riveted.

$$\text{Here } t = \frac{70 \times 90}{15{,}600} = {\cdot}41 \text{ of an inch.}$$

If the tensile strength be taken at 21 tons, 21 × 2,240 = 47,040 lbs., and a reference to the table shows the joint to have 69 per cent. of the solid plate if the holes have been punched ∴ the tensile strength of the joint is 47,040 lbs. × ·69 = 32,457·6 lbs. per square inch.

Taking the factor of safety of 6, the bursting pressure must be 70 × 6 = 420 lbs. per square inch.

$$\text{Then } t = \frac{p \times d}{2 \times 32{,}457{\cdot}6} = \frac{420 \times 90}{64{,}915{\cdot}2} = {\cdot}58 \text{ of an inch.}$$

A simple approximate rule for ascertaining the strength of a well-made boiler is to remember that for a 7 feet double riveted shell of wrought iron, each $\frac{1}{16}$ inch plate thickness, is equivalent to 10 lbs. pressure fully. Thus, for a $\frac{7}{16}$ inch plate we get 70 lbs. boiler. For other diameters and other materials make due allowances.

To determine the number of boilers required at a colliery, allow one 7 feet 6 inches diameter 28 feet long Lancashire boiler for every 200 horse-power exerted by the engines, and then add an extra boiler to the range to permit of one being laid off for repair.

A boiler of 20 horse-power is usually 15 feet long and 6 feet wide, therefore 90 feet of surface or $4\frac{1}{2}$ feet to 1 horse-power; a boiler of 14 horse-power 60 feet of surface or 4·3 feet to 1 horse-power, but engineers allow 5 feet of surface to 1 horse-power.

The steam pipes must be suitable in size to the cylinders of the engine they supply with steam. The proper way to determine their size is by considering the velocity of the steam through them. Mr. Tredgold says, the force of steam in the boiler multiplied by the velocity and the area of the passage must be equal to the elastic force on the piston multiplied by its area and velocity. That is

$$a.f.\,v. = A\,V\,p.$$

where a = area of the steam passages.
f = the force of steam in the boiler in inches of mercury.
v = the velocity of steam through the pipes.
p = the force of the piston in the same denomination as f.
A = the area of the piston.
V = the velocity of the piston.

$$\therefore\ v = \frac{A\,V\,p}{a\,.\,f}$$

$$\text{or } a = \frac{A\,V\,p.}{f\,.\,v}$$

The pressure of the steam in the boilers exceeds the pressure in the cylinders in a ratio which varies from $1\frac{1}{13}$ to double in different engines, and for general calculations is taken at 1½ times. Thus a boiler pressure of 60 lbs. would be equal to 40 lbs. pressure in the cylinders, the difference being employed to overcome the resistance of the steam pipes, valve ports, &c.

The size of the steam pipe should be such that the velocity of the steam flowing through it shall not exceed 100 feet per second.

Molesworth gives the following as the average evaporative power of fuel:—

1 lb. of coke evaporates 9 lbs. of water.*
1 „ „ coal „ 9 „ „ „
1 „ „ coal (slack) „ 4 „ „ „

Stationary expansive condensing engines use from 4 to 7 lbs. of coal per indicated horse-power per hour. Compound engines, 1¾ to 3 lbs.

As supplementary to the foregoing, a few questions and answers relating to boilers will now be given.

Question 26.—Describe fully separators, steam-traps, and Burnam's steam loop, and their use.

Separators are arrangements for draining the water out of long ranges of steam-pipes, which is formed there either by "priming," that is, carried from the boiler with the steam into the pipes, or by condensation. The under part of the separator is below the level of the steam-pipes, and it is formed so that the water and steam, in passing from the steam-pipes into the separator, are effectually separated there, hence the name. The water is precipitated to the bottom of the vessel called the "separator," whilst the steam is made to pass upward round a deflector and enters the steam pipe on the other side quite dry.

A steam trap is an appliance for automatically freeing from condensed water all steam passages and pipes in which such water may have accumulated. This water, when allowed to remain in, occasions considerable leakage and damage, as well as wear and oxidation arising from the water which remains in the passages while the engine is standing. It is sometimes applied direct to the lowest level of the steam pipes so that all water may flow to it; in other forms it is attached to the separator, which it automatically frees of the water accumulated there. There are many different steam traps before the public. The following sketch, Fig. 155, and description of MM. Geneste and Hercher's self-acting steam trap, are taken from the *Colliery Guardian* of April 16th, 1875:—

Two cylinders, C and D, of the same weight, but of different densities, are arranged at the two ends of a rod which is free to oscillate on a shaft at right angles to it. This shaft carries a toothed pinion which gears with a rack, at the end of which is fixed a slide valve, E, serving to open an escape orifice, B. If the apparatus is connected by means of the pipe, A, to the steam pipe which it is intended to keep clear of water, the water and the steam enter this first-mentioned pipe. As the two cylinders in equilibrium are of the same weight but of different densities, it is evident that their volumes differ in inverse proportion to their densities; the result is that the condensed water, which will be accumulated in the lower portion of the trap, will in time rise to one or both of the cylinders; the volumes of water displaced being different, the equilibrium will be disturbed, and the oscillation which will result will bring about the opening of the slide-valve, E. The water will thus be at liberty to flow away, and it is easy to conceive that the slide valve will be opened in proportion to the quantity of water to be ejected. It is claimed for this trap that it is applicable for removing water in passages for compressed air.

Tangye's self-acting steam trap and separator, which, correctly speaking, is a condensed-steam trap, is an automatic apparatus for separating and afterwards discharging the water resulting from the condensation of steam in pipes, steam chests, receivers, drying apparatus, &c. It is made by the well-known firm of engineers, Messrs. Tangyes, Limited, of Birmingham.

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* Feed-water supplied at 212° F.

As will be seen by Fig. 156, it is exceedingly simple in construction, having few working parts, and therefore not liable to get out of order, and can be easily applied by fixing in a line of pipes to which it is joined at the branches cast with the top vessel of separator.

The separator consists of a cast-iron cylinder with a perforated bottom, and divided vertically into two chambers by a partition, and provided with a cover at the top, in which is fixed a small valve, opening inwards, by which air is discharged when first turning on the steam, or to admit air when the steam is turned off. By means of the partition, the steam and water from the pipes are made to pass from one chamber down through the perforations when the water falling is trapped, while the lighter steam, rising through the holes into the opposite chamber, proceeds onwards freed from water.

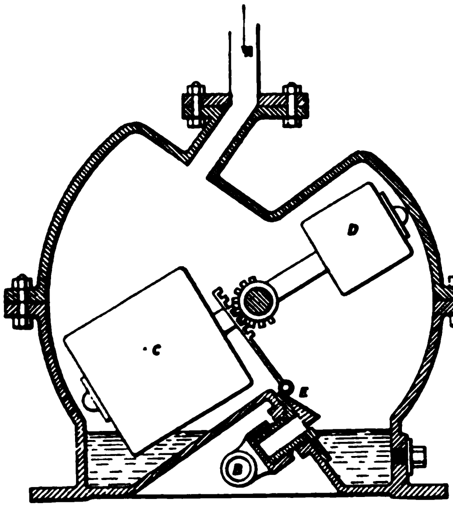

Fig. 155.—MM. Geneste and Hercher's Self-acting Steam-trap.

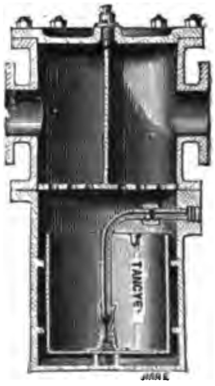

Fig. 156.—Tangye's Self-acting Steam-trap and Separator.

The bottom casing in which the water is trapped, and from which it is afterwards expelled, consists of a cast-iron cylinder, having a solid bottom, to the top of which is bolted the separator—a small iron cistern cast of such a thickness that it will float in water by displacement, and a bent pipe secured to an outlet at the side of casing, and extending therefrom over the side and down the centre, nearly to the bottom, of the cistern, and terminating with a brass nozzle, having a flat valve face at its extremity. The cistern is provided with a small brass disc valve, which is rounded on its underside, so that it shall readily adjust itself to the seat on the nozzle, and is kept in position by a guard.

The action is as follows:—Suppose the casing with cistern to be filled with water, the cistern with its valve will immediately sink to the bottom of casing by the force of gravity, and thus open communication with the atmosphere, through the bent pipe, where the steam pressing on the surface of the water will discharge the cistern of so much of its contents until the weight of the remainder, together with that of the cistern, are overbalanced by the surrounding water

The cistern will then rise, carrying with it the small valve, and thus close the passage through the bent pipe, and remain in that position until the accumulation of water from condensation again fills the cistern, when the operation will be repeated.

Mr. John J. Royle, of Manchester, has devised a steam trap by means of which the condensed water may be returned automatically to the boiler. It is shown in Fig. 157. A is a steam chest on the top of the receiver B, having a three-port slide valve C—of which the port D communicates with the receiver B—the port E communicates, through a pipe not shown, with the drip box, not shown, and the port H with the atmosphere. The slide valve C is actuated by pistons I and J, working in open-ended cylinders affixed at each end of the steam chest A, as illustrated. The closed ends of the cylinder communicate, the one through the bent pipe with the inside of the receiver B through a valve actuated by the float M, and the other through a pipe communicating with the drip box through a valve also actuated by a float.

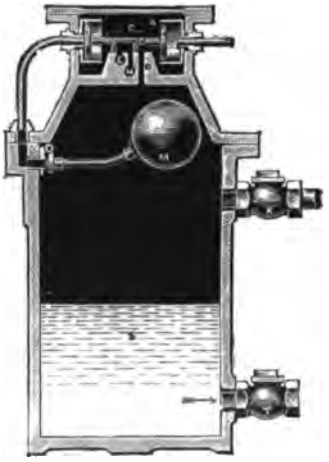

Fig. 157.—Royle's Automatic Return Steam-trap.

The fixing of the apparatus will be readily understood from Fig. 157, where the receiver B is shown fixed above, and arranged to feed back the condensed water to the boiler coming from a system of heating pipes and coils all draining into the drip box situated considerably below the boiler. Steam at the full boiler pressure is supplied through a pipe to the steam chest A, and entering through the open port D presses upon the water S shown in the receiver B, and establishes an equilibrium between the boiler and the receiver, allowing the water to gravitate through check valve T and pipe continued from it to the boiler. Meanwhile the condensed water from the heating pipes, &c., has been entering the drip box through a check valve, and as soon as the water in the drip box reaches the ball of float and lifts the valve connected with it, the equilibrium of the pistons I and J is destroyed and the slide-valve C caused to travel to the right, so reversing the position of the ports, admitting steam at boiler pressure through the pipe leading into the drip box, and allowing the steam contained in the receiver B to exhaust. The condensed water in the drip box is by this means forced up another pipe and through check valve X into the receiver B, until, as soon as it in turn reaches the float M, the slide-valve C is automatically moved to the left into its former position, so admitting full steam pressure on to the water S and feeding it to the boiler as before described.

Meanwhile the condensed water accumulates in the drip box, and as soon as it again lifts the ball of float there, the same action is repeated, and so on continuously.

The special features claimed for this steam trap are:—1. It will automatically elevate boiling water from any distance below the boiler, and feed it into the boiler against any pressure, without requiring any back pressure on the drip pipes—a feature unique, and possessed by no other return trap. 2. Heating pipes can therefore be worked at any steam pressure, and at any distance above or below the boiler. 3. As a boiler feeder this apparatus possesses special advantages for feeding boiling water lying either above or below the boiler level, and which would be impossible to feed back to the boiler by other means.

In the "Expandisc" steam trap shown in Figs. 158 and 159, 158 is an exterior view, while 159 is a sectional one. In it advantage is taken of the quality which certain alloys of metal possess of expansion and contraction under varying temperatures.

The steam trap consists of a cast-iron cylindrical casing—17½ inches high and 10 inches diameter at the base—in the centre of which is a brass valve spindle, carrying four circular valve discs, all cast in one piece from a specially expansive alloy. The casing is bored out conically and the discs are turned conically. They can be raised and lowered by means of a hand-wheel and screw working in a bush screwed inside and outside.

The air and condensed water enter the steam trap at the top. Their temperature being lower than that of steam causes the conical discs to contract. This contraction leaves an annular space between the circumference of the discs and the interior surface of the casing, sufficient for the outflow of the water. When there is no more air or water the steam comes in contact with the discs, expands them and thus closes the annular space, thereby preventing the steam from escaping. As soon as water again accumulates in the trap, the contraction of the discs takes place, the water escapes, and the incoming steam raises the temperature, expands the discs and closes the trap.

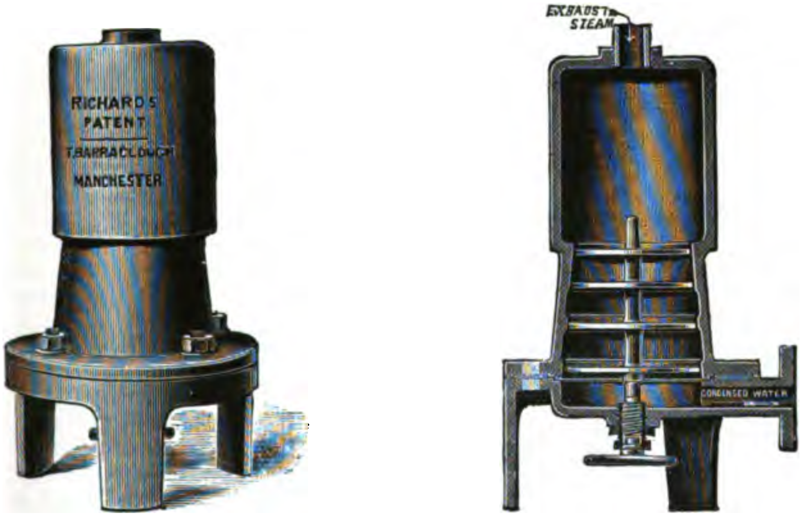

Fig. 158. Fig. 159.

THE EXPANDISC STEAM-TRAP.

These expansions and contractions alternate automatically, according as steam or water is present, and the efficiency of the steam trap is based on the power which the special alloy has of expanding and contracting.

The desired position of the discs is attained by screwing up the bottom bush by means of the key furnished with each trap. This position is easily regulated according to the outflow. The discs must be raised until only water flows, and when once regulated no further adjustment is required.

The valve spindle and discs are lowered by means of the screw and hand-wheel. The steam is allowed to escape for one or two minutes through the annular passage thus formed, and sweeps away any impurities which may have settled on the interior of the casing or on the discs. The valve spindle is then raised to its former adjusted position.

Extreme height......	17½ inches.	Inlet......	1¼ inches diameter.
Diameter at base ...	10 inches.	Outlet ...	¾ inch diameter.

Burnam's steam loop is an appliance by means of which condensed water can be raised from a lower level back to the boiler, without the use of a pump or injector. It is placed in a position similar to that occupied by a steam trap.

Fig. 160 shows the loop as applied to steam engines, &c. It commences at the point G and consists essentially of a pipe, partly vertical and partly horizontal, called the "riser"; another called the "horizontal"; and a third, vertical, called the "drop leg" which ends at the stop-valve, F, on the boiler. If this stop-valve, F, be closed and the whole system blown through and cleared of air and water by opening the drain-cock E for a few moments and afterwards closing it, then this end of the pipe at F becomes the point of lowest pressure. The principle on which the loop acts depends upon the natural law that steam or any other gas will flow to the point of lowest pressure. Supposing the case of a boiler with 100 lbs. steam pressure, and assume there is a loss of pressure at the point G of 5 lbs.; then by the time the steam has travelled along the loop to the top of the drop leg at A a further loss of 5 lbs. may be expected, making a total diminution of 10 lbs. If then there is 100 lbs. pressure in the boiler, and 90 at the point A, how is the difference overcome? If now the riser were filled with water up to about 25 feet above the level of the water line in the boiler, the pressure on either side of the valve F would be equal, and for every drop of water which comes on the top of the column an equal amount must go into the boiler if E be closed. A now becomes the point of lowest pressure, and steam will flow towards it, carrying with it, by its sweeping action in the riser, any water that may be mixed with it up into the horizontal, and draining from that into the drop leg and thence into the boiler. The horizontal acts as a condenser and increases the velocity of the flow. Every cubic inch of water condensed here requires about a cubic foot of steam to replace it, so that the velocity of flow in the small riser is very great. The

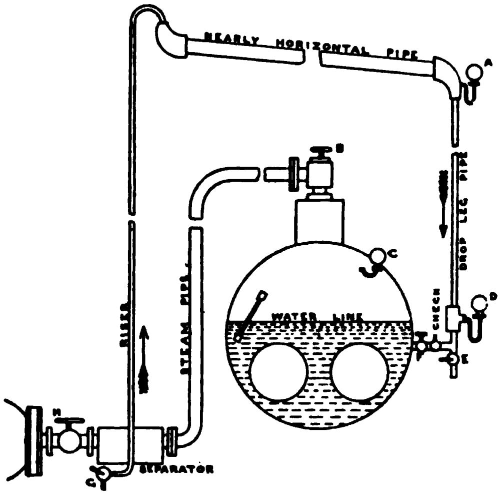

Fig. 160.—Burnam's Steam Loop applied to Steam Engines, &c.

radiation from the loop represents the total amount of heat used up in bringing the water back to the boiler, and if the riser and drop leg are well covered with a good non-conductor, the loss is chiefly in the horizontal, which must be enough to give sufficient velocity to the system. The action of the steam loop then depends upon the weight of the water in the drop leg plus the pressure of the steam being sufficient to force the water to return into the boiler. The operation is continued entirely by the condensation of steam; whether this is more economical than dealing with the condensed water in other ways is somewhat doubtful. To commence the operation a certain amount of steam must be consumed to expel the air and water standing in the pipes. After the cock is closed the condensed water is pushed through the pipes. The operation is continued by virtue of the steam condensing in the horizontal, but whether the consumption of steam necessary for this purpose is not greater than would be required to work a pump, is open to question.

Question 27.—What are feed-water heaters and economisers?

A feed-water heater is simply an appliance for heating the feed-water by means of exhaust steam, before passing into the boiler. There is manifestly an advantage in sending the feed-water to the boilers hot, and at collieries there is usually much exhaust steam that may be utilised for the purpose. In its simplest form the feed-water heater consists of an old boiler, into which the exhaust is conveyed below, and the feed-water is conveyed above. The exhaust steam is condensed on coming into contact with the cold water, and the temperature of the feed-water is thereby raised. This is then conveyed to the boilers by another pipe. This plan frequently requires the use of two pumps, one for the hot and one for the cold water, and whatever impurities come from the cylinders go into the boiler and from the boiler into the cylinder again. By the use of Berryman's feed-water heater, or others of that description, the evil of passing the grease and dirt from the cylinders into the boiler is avoided. The exhaust steam enters the heater and passes through a number of brass and copper tubes, which are surrounded by the cold feed which enters the heater at another point. The passage of the steam through the tubes raises the temperature of the feed-water, which is conveyed by a pipe from the heater for the purpose to the boilers. The exhaust steam does not mix with the feed-water, but passes out of the heater by a proper outlet. Only one pump is required for the cold water, or an injector may be used. The Berryman heater is also used as an interheater where compound engines are working. It may be fixed at any convenient distance from the cylinders in or outside the engine-room. The exhaust steam is passed through the inside of the tubes in the interheater and thence into the steam chest of the low-pressure cylinder. The objection to these heaters is their tubes and complications which are liable to get out of order, and probably the best feed-water heater is the exhaust injector, which has been fully described. In it there are no tubes to get out of order; it requires no pump, is simple in construction, and effectually heats the feed-water being conveyed to the boilers. An economiser is somewhat similar to a feed-water heater, but in it the water is heated by the hot gases which issue from the boiler flues. These gases pass through the tubes similarly to the exhaust steam in the feed-water heater.

Question 28.—Besides the ordinary lever safety valve, describe any other safety arrangements that have been devised for boilers.

Cowburn's dead-weighted safety valves are stated to be much more efficient than the ordinary lever valve. They are made singly or in groups. Each valve in a group is exactly one square inch area and the heap of weights is just the intended pressure in pounds. These weights keep the valve in its seat until the steam attains a certain pressure, when the steam lifts the valve and escapes through the opening at the top of the weights.

Johnston's patent self-acting alarm whistle guards against accidents from deficiency of water in the boiler. A hollow cast-iron float is made sufficiently heavy that on falling with the water in the boiler it opens an orifice through which the steam rushes, thereby causing the alarm whistle at the top to be sounded. The apparatus is free of all stuffing boxes, glands, cocks, or any complicated contrivances. As long as there is sufficient water in the boiler, the alarm valve is kept close against its seat by the float.

Smith's steam sentinel is an invention to prevent over-pressure in steam boilers and is a check on the safety valve, because it gives a distinct and unmistakable warning immediately the maximum pressure is exceeded. Its construction is simple. A conical valve stops a hole in the boiler, and is kept down by a spring carefully adjusted to resist the pressure of the steam up to a certain point. As soon as this pressure is exceeded, the valve is liberated by the compression of the spring, and a communication is opened for the steam into a whistle of the ordinary form, which gives a loud warning of approaching danger.

> *Question* 29.—In a lever safety valve, the whole length of the lever is 32 inches, the distance between the fulcrum and the valve 4 inches, the diameter of the valve 2½ inches, required what weight must be put on at the end of the lever so as to have a pressure of 50 lbs. per square inch upon the valve; also to divide the lever so as to have 40, 30, and 20 lbs. upon the valve with the same weight?

The area of the valve will be $2{\cdot}5^2 \times {\cdot}7854 = 4{\cdot}9$.

The leverage will be $\frac{32}{4} = 8$, so for a 50-lb. pressure we have $\frac{4{\cdot}9 \times 50}{8} =$ $30\frac{5}{8}$ lbs. as the weight to be put on the end of the lever to give 50 lbs. per square inch. And $\frac{4{\cdot}9 \times 40}{30\frac{5}{8}} = 6{\cdot}4$; and $6{\cdot}4 \times 4 = 25{\cdot}6$ inches, the distance from the fulcrum at which the weight must be put to have a pressure on the valve of 40 lbs. Similarly $\frac{4{\cdot}9 \times 30}{30\frac{5}{8}} = 4{\cdot}8$ and $4{\cdot}8 \times 4 = 19{\cdot}2$ inches, the distance from the fulcrum at which the weight must be put to have a pressure on the valve of 30 lbs. Again, $\frac{4{\cdot}9 \times 20}{30\frac{5}{8}} = 3{\cdot}2$ and $3{\cdot}2 \times 4 = 12{\cdot}8$ inches, the distance from the fulcrum the weight must be put to have a pressure on the valve of 20 lbs. The weight, it will be thus seen, must be moved towards the fulcrum $(32 - 25{\cdot}6)$ $6{\cdot}4$ inches for every 10 lbs. taken off the pressure on the valve.

> *Question* 30.—What is the pressure per square inch in a boiler, the whole length of the lever being 32 inches, the distance between the fulcrum and valve 4 inches, the diameter of the valve being 2½ inches, a weight of 30⅝ lbs. being placed at the end of the lever?

Here we have $\frac{32}{4} = 8$ for the leverage, therefore the whole weight on the valve is $30\frac{5}{8} \times 8 = 245$ lbs., and $\frac{245}{2{\cdot}5^2 \times {\cdot}7854} = 50$ lbs. as the pressure per square inch in the boiler.

> *Question* 31.—If the safety valve is 4¾ inches in diameter, and the lever is 38 inches long to the centre of the weight, and 4½ inches from the fulcrum

to the centre of the valve, and the weight is 85 lbs., what is the pressure per square inch?

The leverage is $\frac{38}{4\frac{1}{2}} = 8{\cdot}\dot{4}$, and $8{\cdot}\dot{4} \times 85 = 718$ lbs. nearly. The area of the valve would be $4{\cdot}75^2 \times {\cdot}7854 = 17{\cdot}72$ square inches. Therefore $\frac{718}{17{\cdot}72} = 40\frac{1}{2}$ lbs. per square inch.

Question 32.—A steam gauge shows 20 lbs. on its face; what does this mean?

As already explained, that the pressure of the steam inside the boiler is 20 lbs. above the pressure of the atmosphere.

Question 33.—What is the effect of sediment or incrustation forming in a boiler? Is there any danger likely to arise from it?

The deposits or incrustations formed internally on the boiler plates are salts, and bad conductors of heat. Consequently, the boiler plates will become red hot whilst the deposit remains; after becoming red hot, through unequal expansion, the incrustation separates from the iron, the consequence being that water rushes between, a sudden generation of highly elastic steam takes place, which sometimes the valve will not allow to pass, and an explosion follows.

Question 34.—Are boiler plates ever made less than $\frac{5}{16}$ of an inch thick? What is the proper size and strength of plates for high-pressure boilers?

Boiler plates are not often made less than $\frac{5}{16}$ of an inch thick. For high-pressure boilers, they are made from $\frac{7}{16}$ to $\frac{11}{16}$ of an inch thick, and are commonly 6 feet × 3 feet, and they must have a tensile strength of at least 20 tons to the square inch. Steel plates are now rolled large enough to admit of one plate forming an entire ring. Thus there is only one longitudinal seam in the boiler made of such plates.

Question 35.—What evaporating surface should boilers have?

They should have at least 5 square feet per horse power.

Question 36.—When the water in a boiler is dangerously low, what would you do?

I should open the fire-doors, close the damper, and stand at a safe distance until the boiler had got cool, and for this reason. If the water is allowed to get dangerously low, some of the plates will be subjected to great heat—possibly may be red hot—and any attempt to pass water into the boiler while in that state would be followed by a rapid generation of highly elastic steam, which the safety valve would not pass, and an explosion would result. After the boiler had cooled, it might safely be re-filled and started.

Question 37.—At what part of the boiler would you fix the water gauge?

The water gauge should be placed on the front and furnace end of the boiler, where it would be constantly under the eye of the fireman, and any deficiency or surplus of water in the boiler at once detected.

Question 38.—What size of steam pipe would you lay from the receiver to a 30-inch cylinder engine?

The maximum velocity of the steam in the main steam pipe should not exceed 100 feet per second. The piston speed of the engine is not given, but taking it at 300 feet per minute, the size of pipe for a single cylinder engine would be found thus:—A 30-inch diameter cylinder has an area of 706·86, therefore $\frac{300 \times 706{\cdot}86}{100 \times 60} = 35{\cdot}343$ area in inches, and $\sqrt{\frac{35{\cdot}343}{{\cdot}7854}} = 6{\cdot}7$ inches, or say $6\frac{3}{4}$ inches in diameter. With a double-cylindered engine it would be $\frac{300 \times 706{\cdot}86 \times 2}{100 \times 60} = 70{\cdot}686$ area in inches, and $\sqrt{\frac{70{\cdot}686}{{\cdot}7854}} = 9{\cdot}487$ inches, or say $9\frac{1}{2}$ inches in diameter. If the engine is a compound one, only the size of the high-pressure cylinder need be considered in estimating the size of the steam pipe.

Question 39.—In a colliery where engines were working, and having 1,800 horse-power, what boilers would you consider necessary?

$\frac{1800}{200} = 9 + 1 = 10$ Lancashire boilers 28 feet long and 7 feet 6 inches diameter.

Question 40.—What height and size of chimney would you construct for such range of boilers?

$10 \times 5 = 50$ feet area = say 8 feet diameter at the base, and to be well proportioned the chimney should be 25 times the diameter in height $= 8 \times 25 = 200$ feet high.

Question 41.—What is the nominal horse-power of a Lancashire boiler whose length is 30 feet and diameter 6 feet?

$\frac{30 \times 6}{6} = 30$ horse-power.

Question 42.—Are boilers better calculated to resist pressure lengthwise or crosswise, and in what proportion?

A common cylindrical boiler is twice as strong to resist pressures acting lengthwise than to resist pressures crosswise, and for this reason all the horizontal seams should be double riveted.

Question 43.—How much stronger are boilers having double-riveted plates than those having single-riveted plates?

Double riveting weakens the plates about one-fourth, single riveting about one-half, therefore double-riveted boilers are stronger in the proportion of about 3 to 2.

Question 44.—Find the bursting strength of an iron cylindrical boiler 6 feet in diameter, and made of $\frac{1}{2}$-inch plates, with doubled-riveted joints.

$\frac{{\cdot}5 \times 21 \times 2240 \times {\cdot}69}{6 \times 12} = 225{\cdot}4$ lbs. per square inch.

At most collieries railways are required, but the circumstances of each must decide what railways shall be made. Siding room for full and empty trucks sufficient for the day will be required, but in laying out, regard will of course be had to the intervals between the trains and the output of the colliery. It is not necessary to say more of this here, nor is it necessary to explain the different forms of wagon in use, as these vary much in different districts. Coke ovens and coal-washing machinery, briquette and brick-making machines are in use in some districts, and form part of the surface arrangements; but as these subjects are adjuncts to mining, and a knowledge of them not absolutely necessary for a student to pass an examination in mining, it is not necessary that they should be treated here.

At collieries which supply gunpowder or other explosives, it will be necessary to remember the provisions of the Explosives Act, and to erect the gunpowder magazine in accordance with legal requirements. The written approval of H.M. Inspector should always be obtained both as to plans and position before erection.

A ventilating-fan and a pumping-engine may also possibly form part of the surface erections, but more will be said of these under the heads of Ventilation and Drainage.

CHAPTER VII.

TIMBERING AND WALLING.

The kind of Timber used at Collieries—Storing it Underground—Method of fixing Props and Lids—Temporary Props and Lids—"Dog" for drawing Props—"Sets" of Timber and their fixing in Main Roadways—Timber for Collars—Sills under Props—Timbering for a bad Roof, where the Floor and Sides are good—Lagging—Timbering for a bad Roof and Side, the other Side and Floor being good—Timbering for a bad Roof and Sides, with a good Floor—Timbering for a bad Roof, Floor and Sides - Lagging of Trees and Brushwood—Sizes of Timbers and their distance apart—"Cogs" or "Chocks"—Methods of Timbering in France—Notching the Timber—Cast-Iron Props—Wrought-Iron and Steel Supports—Storing the Timber on the Surface—Creosoting as a means of preserving Timber from Decay—Customs as to Setting and Drawing the Timber—Walling the Main Roads from the Shaft—Material used in Walling—Semi-circular arched Roadway—Invert under Side Walls—"Horse-shoe" Arch—Elliptical Arch for Roadway—Process of building Arches—Necessity of removing all Timber, and tightly packing behind the Walls of Arches—Packing the Top and Sides with Sand.

TIMBERING is the cheapest way of securing roads, regard being had to first cost only; but if the roads are used a number of years, and the cost of maintenance is taken into account, walling may be a much better and cheaper plan.

The timber used at collieries to support the roof and sides is chiefly pine, fir, and oak. The sizes vary from 4 to 12 inches in diameter, the size and arrangement depending upon the material to be supported and the excavation itself. Where used to support the roof only, the timber requires very little preparation. It is cut into suitable lengths at the surface, sent into the pit, and, in accordance with the Mines Act, 1887, for the convenience of the workmen who have to timber the working places, a proper supply must be kept stored near at hand.

The usual manner of supporting the roof in the working places is by means of single props (called "posts," "trees," &c.), having short lids or caps (in Somersetshire called "traps") on the top. If these props are cut from the tops of old trees, they are spongy in texture, and less durable than when cut from the lower portions of young trees. The bark should always be left on them, as it helps to preserve the wood.

In fixing props, the workman with one hand holds the lid under the roof requiring support, whilst with the other he moves the top of the prop, the bottom of which rests on the floor, until it touches the lid, which is firmly held by the post, while the latter is driven well under the lid by means of a sledge-hammer. The post should be upright if the seam lies flat; if not, the prop will not be upright, but at right angles to the floor and roof, or, as the roof will sink a little notwithstanding the prop, the latter may be set in a direction which deviates slightly from the perpendicular between floor and roof towards the vertical. A single prop and lid is sometimes though not often fixed in the main roadways as well as the working places. Fig. 161 shows a single prop and lid. At times it is required to fix a single prop and lid for a temporary or passing purpose.

For instance, where a double row of "chocks" is used in Longwall workings, as a protection to a continually advancing face, the back row is often taken down and re-fixed in front of that which was previously the front row. Each of these chocks should have temporary single props and lids, placed round it to protect the workman while in the act of taking out the chocks. These temporary single props may afterwards be withdrawn, and to facilitate the after removal, when fixing them a short sill is placed on a few inches of rubbish, the prop and lid being placed over the sill (see Fig. 162). These props are afterwards safely removed and recovered by means of a "dog." Fig. 163 shows a "dog" for drawing props, as used at the Lundhill Colliery, Yorkshire. It is an iron bar 3 feet long, with one end shaped in the form of a hook, as shown in the sketch. Five inches from the hook end is a chain with two links and a hook about 6 inches long. The "dog" is used with a piece of half-inch long-linked chain 6 feet or more in length, with a hook at the end of it. The chain is passed round the prop to be drawn, and hooked into a link. The other end of the chain is put into the hook at the end of the "dog" chain. Another prop is used as a fulcrum, the point of the lever is pressed against this, and the prop is drawn out. The length of the chain must be such as to allow the timber-drawer to be safely clear of the falling roof consequent on the withdrawal of the post.

Scale: 6 Feet to 1 Inch

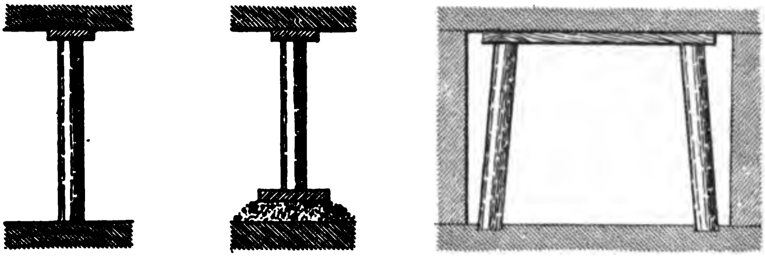

Fig. 161. Fig. 162. Fig. 164.

METHODS OF TIMBERING ROADWAYS.

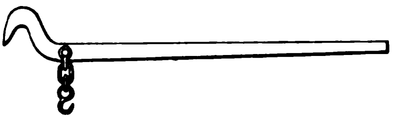

Fig. 163.—DOG FOR DRAWING PROPS.

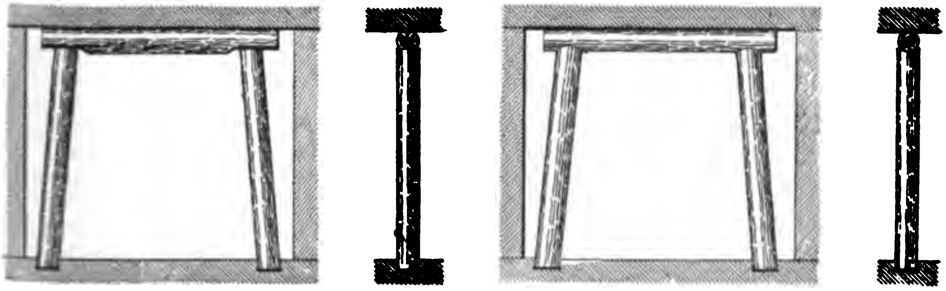

Fig. 167. Fig. 168. Fig. 169. Fig. 170.

METHODS OF TIMBERING ROADWAYS.

Usually the main roadways are secured by "pairs" or "sets" of timbers, as shown in Figs. 164, 167, 168, 169 and 170, Figs. 164, 167 and 169 being front elevations, and Figs. 168 and 170 side elevations of roadway. These consist of a head-piece or " collar," which is held up in position next the roof by two props, reaching from the floor to near each end of the " collar."

It requires two timbermen in large roadways to fix this timber, and if the roof is very bad it may require more.

The collar, having been placed where required, is held there temporarily by a prop from the floor to the middle of the collar. Holes or notches are prepared beforehand in the floor at the sides of the road. One end of a prop is now placed in one of these notches, and the other is brought under the collar, where it is driven sufficiently to be quite firm. The other prop is treated similarly, and the two props may afterwards be alternately tightened by being more firmly driven under the collar, after which the middle prop is removed. As seen on the drawings, these props instead of being vertical when in position are slightly inclined at the top towards the centre of the roadway. The object of this is to strengthen the support. The distance of these props from each other on the floor of the road is often regulated by the gauge of the way and the size of tub or tram having to pass between them; if made the same distance apart at the top and bottom, the weight of the incumbent strata would be more likely to break the collar in the centre than when the props are slightly inclined towards each other at the top, thereby reducing the length of collar between the supports.

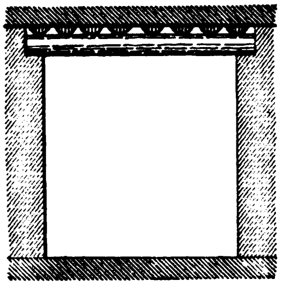

Fig. 165.—Timbering a Roadway.

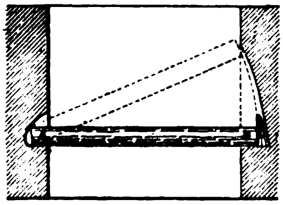

Fig. 166.—Timbering a Roadway.

Sometimes the "collars" consist of slabs of wood or "flats" sawn into suitable lengths, as shown in Fig. 164, but as this kind of collar will not bear much weight without breaking, more frequently trees with the bark left on exactly the same as the props are used. In other cases they are split down the middle, so as to form two half-round pieces, and where this is so the splitting should be done by cleavage, because sawing slightly weakens the piece by dividing the fibres. Where space is not important, however, it may be found more economical to select slightly larger timbers and saw them, the larger size compensating for the loss of strength resulting from sawing. Where these half-round pieces are used as "collars," the flat portion should be laid next the roof, in order to cover as large a surface as possible. Whatever the collar is, care must be taken to ensure its bearing against the roof.

If the latter has an even surface, there will not be much difficulty in this, but in other cases the irregularity of the roof presents obstacles to true contact between it and the collar.

In fixing the props which support the collars, they are placed sometimes with the smaller ends downwards, and sometimes with the smaller ends upwards, but if the floor is strong the smaller ends should be downwards.

If the floor is very soft, the pressure from above causes the props to sink, and the larger ends should be downwards. In some cases this will not be sufficient to prevent the sinking, and advantage may be derived from placing a sill under the foot of each prop. The size of sill will be determined by experience.

Roadways having a strong floor and sides, but a bad roof, do not require the " sets" of timber just described. In these no props will be required, as the sides of the roadways are notched sufficiently for the insertion of the collars, which thus rest or have a bearing on the sides. Fig. 165 shows the front elevation of a roadway timbered in this way, and Fig. 166 shows a plan of the same. A hole or notch is prepared in the side A, near the roof, and is cut to a depth sufficient to give the bearing it is requisite the collar should have. On the opposite side of the roadway, and at the same height, is cut a similar hole B, but the fore side of the hole is shorn away, as shown in the figure, to allow for the fixing of the collar. One end of the collar is now placed at A, the collar being held in the position shown by the dotted lines in the figure, whilst doing so. As the end B approaches its permanent position, a wedge is placed at C, having its thin end outwards, as shown in the figure, and against this wedge the collar is driven home by blows from a heavy hammer. If the road is steep, the shearing back for the hole B should be on the higher side, as the collar would be thus less liable to work out after fixing.

If the collars are not sufficient to prevent falls from the roof, lagging is placed in between the collars and the roof at right angles to the collars. The lagging may be of slabs, or round or half-round timber. If half-round be used, the flat side should be placed next the roof, as shown in Fig. 165. The object of either kind of lagging is to distribute the support of the collars over the roof. If after placing the lagging over the collars, any spaces exist, either between the collars and lagging or between the lagging and roof, these should be closed by having wedges firmly driven into them.

If a roadway has a bad roof and side, the other side and floor being good, it may be "half timbered," as shown in Fig. 171. A hole is cut in the sound side at A to receive one end of the collar and the floor is notched at C, in which is fixed a prop, slightly inclined. A collar is then placed with one end in the notch A and the other resting on the prop. The prop and collar are now driven by blows till the former is quite upright or slightly inclined towards the centre of the road. If the two pieces are not held firmly together, owing to the yielding of the floor during the driving, wedges are driven either above or below the prop.

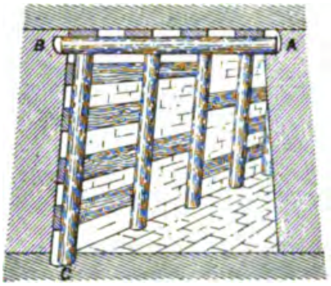

Fig. 171.—Timbering a Roadway.

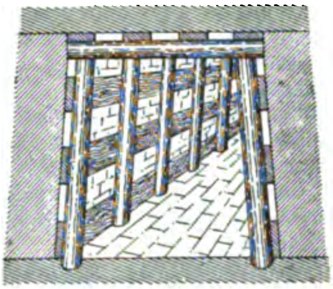

Fig. 172.—Timbering a Roadway.

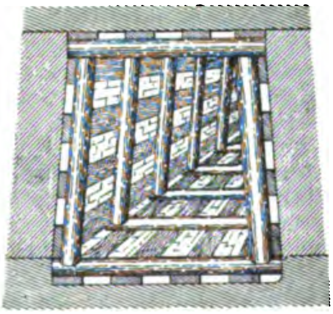

Fig. 173.—Timbering a Roadway.

Lagging is placed next the roof and also next the side from which the pressure is threatened. The lagging between the props and the side is firmly wedged so as to retain its proper position.

Fig. 172 shows the timbering in a roadway having a bad roof and sides, but a good floor. Here "sets" of timbers are placed as before described, and lagging placed next the roof and against both sides. Flat, half round, or round lagging may be used, according to the pressure it is likely to have to resist, vertically, or laterally.

It often happens that the roof, floor and sides are all weak and require support. Where this is the case, if the roadway is to be maintained for any length of time, walling will be the most effectual and economical way of preserving the road. But if only required for a short time, it may be timbered as shown in Fig. 173. In this case, the "sets" of timber have placed under them pieces similar to the collars, and these may be of half-round pieces if the pressure is not great. Where the half-round pieces are not strong enough, whole pieces must be used; and again, where these do not effectually resist the pressure, lagging must be placed under them, similar to the lagging at the sides and roof as shown in Fig. 173.

In the southern portion of the Somerset coalfield, and also in some of the Pembrokeshire mines, where the roads are very difficult to keep open, branches of trees and brushwood are frequently used as lagging. These form a network against the sides and roof and distribute the weight more evenly over the supports than ordinary lagging.

The distance between the sets of timber or the collars must depend upon the state of the strata. In some cases they may be placed only a few inches apart, the road being quite lined with them, or in others at intervals of 3 feet or upwards.

Where the timbers are not of a uniform size along a roadway, the larger and smaller should be made to alternate, so that a weak pair may come between two strong ones, but this method is not desirable.

The diameters of the timbers should increase with their lengths, so that those cut for a high or wide road must be thicker than those used in roadways of smaller dimensions.

In timbering Longwall workings, or the pillar workings of Post and Stall, besides the props cut the height of the seam with lids placed over them of about 15 inches long, "cogs" or "chocks" are used. No drawing is here given of these, as many examples are shown in the next chapter. They are pieces of timber about 2 feet long and from 6 to 12 inches square. If it is intended to recover them, a little rubbish is laid on the floor, and two of these timbers are laid on this parallel to each other and about 18 inches apart. Two similar pieces are then placed on these crosswise, also 18 inches apart, parallel to each other, and at right angles to the two first laid. Two more similar pieces are placed over the last in a line with the first two laid, and so on till the roof is reached, where they are wedged with pieces of chip.

Chocks which are to be taken out and used over and over again are generally made of hard wood, such as oak, elm, or ash, but in cases where no attempt is made to recover them ordinary soft round pit timber is used with the bark on, the latter being often 4 or 6-foot timbers. If intended to be left in, the space between them is filled with rubbish as they are built up.

Figs. 174 and 175 show a form of timbering employed in France, as described by André in his *Treatise on Coal Mining*, the chief features of which are the struts supporting the props and the collar at the point where they have a tendency to give way, and also the use of longitudinal pieces to bind the different sets together.

Fig. 174 shows the arrangement for a road having a single, and Fig. 175 that for a road having a double line of rails. The operation of fixing the bracing pieces inside the other usual set of timber is as follows :—

The longitudinal immediately under the collar is placed and temporarily held there by pieces of wire. The length of the piece is 9 or 10 feet. Next the two side longitudinals are placed and temporarily held in position similarly to the one immediately under the collar. Two or three upright struts are then placed under the side longitudinals, and afterwards some of the upper struts are inserted obliquely and driven towards their permanent position. The wires are now removed, and the remaining upright and oblique struts inserted and driven firmly into position by blows from a wooden mallet.

Scale. 6 Feet to 1 Inch.

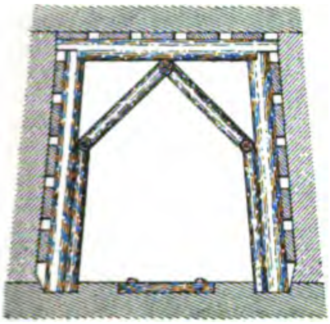
Fig. 174.

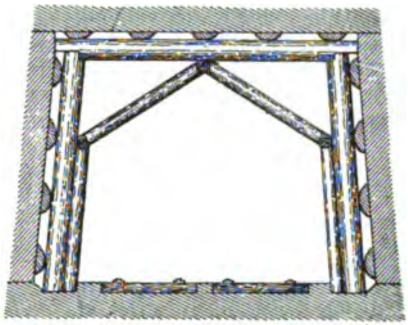
Fig. 175.

Methods of Timbering Roadways.

If every set of timber is thus braced it is capable of resisting enormous pressure. The struts and longitudinals are 3 inches in diameter for a single line roadway, and 4 inches for a double line roadway. Figs. 176 and 177 show a modification of the same system of bracing the main timbers.

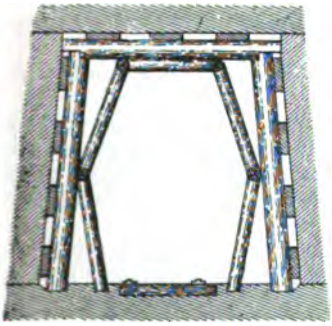
Fig. 176.

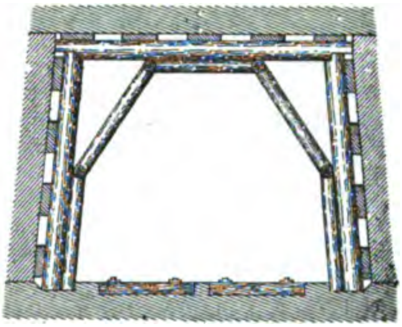
Fig. 177.

Methods of Timbering Roadways.

In Fig. 176 the upright struts, instead of being in contact with the props throughout their length, are set out at the foot, and the oblique struts, instead of supporting one longitudinal in the centre of the collar, support two, which again support a short horizontal strut under the central portion of the collar.

Fig. 177 is much the same as Fig. 176, but is meant for a double line roadway; and in this case the upright struts are in contact with the props throughout their length.

Where the props in any "sets" of timber have lateral pressure thrown on them it is usual to bind the " set " together by notching the timbers. The notches in the floor prevent the props from being displaced at the bottom, and the notch formed in the collar and props is designed to prevent the lateral pressure from disturbing the position of the props at the head.

There is no doubt that notching weakens the timber, as a portion of the wood is cut away. The notches formed on the collars cut away from $\frac{1}{3}$ to $\frac{1}{2}$ of their thickness at that point. But no great objection arises from this weakening, because the collar, after being notched, placed in position, and supported immediately under the notch by a prop, is still much stronger at that point than in the centre. If any roadway be carefully examined it will be found that the collars nearly always break in the centre or at a distance from the notch.

Figs. 167 and 168 show a method of slightly notching, in which the prop is left cut off square, and the collar is cut away on either side with an axe sufficiently to give a flat bearing surface.

Scale. 6 Feet to 1 Inch.

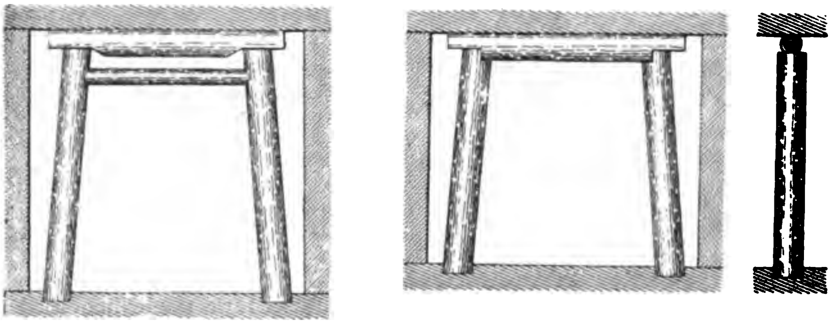

Fig. 178. Fig. 179. Fig. 180.

Methods of Timbering Roadways.

Figs. 169 and 170 show another method, in which the upper end of the prop is hollowed out with an axe to receive the rounded ends of the collar, which is not cut. There is much less protection from lateral pressure in timbers cut thus, but it may sometimes be a convenient arrangement for fixing sets of timber where there is little or no side pressure.

Fig. 178 shows the same arrangement of notching as Figs. 167 and 168, with a stay placed across between the two props, the stay being hollowed out to receive the props.

If the lateral pressure is greater than the pressure from the roof, the props will be knocked out by it unless a better form of notch be adopted, such as shown in Figs. 179 and 180, which is cut in both collar and the props. The lateral pressure sometimes causes the collar to split; to prevent this a stay may be inserted similar to that in Fig. 178.

In Figs. 181 and 182 the Welsh system of notching timber is shown. This notch is formed in placing the sets of timber at the most important collieries in South Wales, in many of which the side pressure to be resisted is enormous. No stay is required with this notch, and the collar may be sawn to the length shown on the figures, or it may be rather longer. The shape of the roadway determines this. The prop ordinarily is in contact with the collar for 3 inches on the inside, or nearest the centre of the road, and for 4 inches on the outside.

Frequently a loose stone at the side of a roadway requires support. It may be thick and partly protected by the roof, or partly by the side, but having a weak place it must be secured, if it cannot be taken down. In circumstances which admit of it, a prop and lid are placed under it; in others, the prop has to be placed in an oblique position, either being set on the floor or the other side of the roadway. In such cases a good plan is to slightly scoop out in a curved form the top of the post, and then to have a wedge-shaped piece having a rounded bottom to fit the prop, and a flat top surface. It may then be driven in over the prop by blows from a hammer till tension is obtained.

Cast-iron props have occasionally been used, but their inelasticity or brittleness is a great objection, and this, probably, more than the cost, has prevented their coming into general favour. Where the pressure is great a metal prop breaks suddenly without warning, and has none of the bending property of timber, which, as a rule, shows signs of giving way, thus allowing time for repairs or renewal.

Wrought-iron and steel are also used to some extent for securing underground roadways. In main roads which are likely to be used for years, and which, if timbered, would necessitate the renewal of the timber every two or three years, it would probably be cheaper in the long run to adopt iron or steel. Although the first cost of these materials is relatively high as compared with timber, there is no comparison in their serviceableness; beside which the iron or steel, if recovered, even if bent or broken, is still worth a considerable part of the first cost. There is, however, this great convenience in the use of timber, which is wanting in iron or steel, that it may be cut to suit any size of roadway.

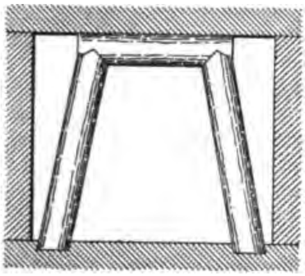
Fig. 181.

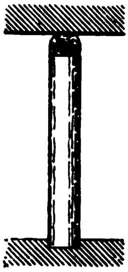
Fig. 182.

TIMBERING A ROADWAY.

Timber is usually bought in large quantities. To prevent its deterioration care should be taken in the storing, which is best done in a covered building placed, for convenience, near the shafts. The pieces should be placed horizontally, and crosswise, to allow air to circulate, and means should be taken to ventilate the building. The small timber should be placed on end. When using from the store-room, the oldest stock should be used first.

Some roofs fall for a few feet upwards to a harder rock, and advantage is taken of this fact by allowing the fall to go on for a time, and then, after it has ceased, the permanent road is made with little or no timber under the harder rock. Other roofs will fall to an indefinite height, necessitating walling in the main roads. It is much better to let the goaf * settle before putting in

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* *Goaf* or *gob* is the name given to the space from which coal or other mineral has been entirely worked. There is usually more or less of this space *open* at and near the edges of an area from which the mineral has been entirely extracted, the closing of the interior depending upon (1) the nature of the roof and floor, and their capability of resisting the incumbent pressure, and (2) the amount of packing built and the débris thrown back into the goaf in the process of working. In general, the interiors of large goaves are practically closed.

walling, if even the road has to be timbered for a year or two before the walling is built. Sometimes, in working thick seams, a bad roof over the coal prevents roads being carried immediately under it. In this case the top bed of coal is often allowed to remain, thus affording a better roof than that over the coal.

Wherever timber is placed in damp situations it is affected injuriously by the watery vapour in the atmosphere. Some kinds of timber decay more rapidly than others where so exposed. English larch stands a considerable time in damp places, but any timber used in wet roadways may be rendered twice as durable by "creosoting." 100 parts of coal tar contain, when distilled, 65 parts of pitch, 20 of essential oil (creosote), 10 of naphtha, and 5 of ammonia. The oil produced from this distillation is used for creosoting timber. It prevents the absorption of moisture in any form, under any temperature. It is noxious to animal and vegetable life, repelling the attacks of insects, and preventing the propagation of fungi. The oil is injected at a temperature of 120° F. under a pressure of 150 lbs. per square inch, so that ordinary fir timber absorbs about 8 or 10 lbs. weight of creosote per cubic foot.

In return airways which are damp and warm ordinary timber soon rots, and if its use cannot be dispensed with in these situations, it should be creosoted before being placed there.

In many districts props and sprags are used at the working faces to prevent the coal from falling on the workmen whilst engaged at their work. As the system of propping the coal is inseparably connected with the mode of working, which again is regulated by considerations respecting the seam, we reserve our remarks on these subjects until the next chapter, when the systems of propping and spragging the coal will be dealt with, and many examples shown of what is being actually done.

In Northumberland and Durham skilled workmen called deputies are employed. The duties of a deputy for the most part consist in setting and drawing timber in any district over which he has charge. Besides the timbering, he lays the rails where required, and takes up any from recently abandoned roads, and also attends to the bratticing, ventilating doors, &c. Usually he has about a dozen men to attend upon in his district.

In the counties mentioned the workmen at the face are relieved of the responsibility of propping the roof, which responsibility devolves on the deputies, who are officials acting on behalf of the owners, and in these districts, where the roofs are fairly good, this plan seems to work well.

In Derbyshire and Staffordshire a kind of Butty system is in vogue, in which stallmen have charge of the Longwall faces, receiving a tonnage or contract price on the coal sent out, and these stallmen employ workmen to hole and take down the coal, whilst they or others employed by them put in the props and build the pack-walls. A stallman has from 8 to 16 men under him.

The usual arrangement is for the miners or persons working at the face to set all the timber required there for their own protection, and a great deal may be said in favour of the arrangement, but it should always be under supervision from the officials, and subject to some kind of regulation as to the extreme distance allowed between props, &c., according to the requirements of the case.

The timbering on the main roads is done by timbermen appointed for the purpose.

WALLING.—The roadways forming the main underground arteries are frequently walled. Sometimes the walling is continuous over long distances, at others where the road has passed here and there through weak beds the walling is in short pieces. Air-crossings may require walling, and the spaces between doorposts and the road sides where important doors are fixed. Walling may be of stone or brickwork. If stone is used the hard sandstones of the coal measures are found very suitable and are much harder than ordinary bricks. Bricks, however, are very convenient to handle, and may be moulded to suit the curve of any archway, whereas the hard sandstones require a considerable amount of dressing if used for arching. Sometimes, where arches have to be turned the side walls are of stone and the arch bricked.

Scale. 4 Feet to 1 Inch.

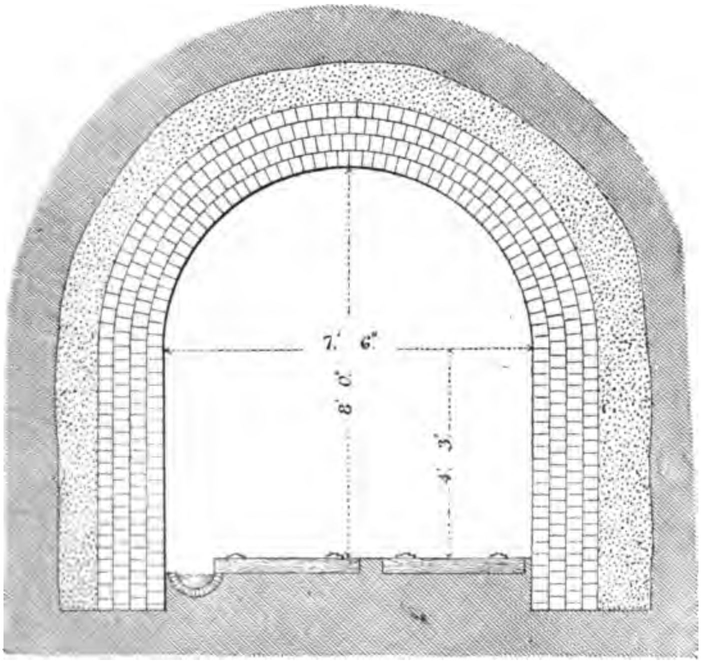


Fig. 183.—Method of Securing a Roadway having a Good Floor but a Bad Roof and Sides, by means of Side Walls and Semi-circular Arch.

Where the roof is good and only the sides bad, walls of stone or bricks may be built from a few inches below the floor line to the roof on each side. The thickness of masonry will depend upon the nature of the rock, but should not be less than 9 inches. Very little mortar should be used between the stones or the bricks in building. If the floor and sides are strong and the roof bad, an arch may be turned resting on the side rock. If one side and the roof are bad a wall is built next the bad side and the arch turned resting on this side wall and the firm rock on the other side. When both sides and the roof are bad two side walls may be built and an arch turned resting on them. An arch which is semi-circular in shape, is shown in Fig. 183. The size on the drawing is for a double line of rails, and a gutter for carrying the water is shown at the side. Where the floor is bad as well as the sides and roof, the side walls may be built on an invert as shown in Fig. 184. Where a considerable amount of lateral pressure is expected, and the upright side walls are not calculated to resist it, the roadway may be walled as shown in Fig. 185. This form of arching is usually adopted for railway tunnels, the sides and roof, forming part of an ellipse, resting on an invert and sometimes called a "horse-shoe" arch. Fig. 186 shows an elliptically arched roadway, which is the strongest form that is suitable for underground roadways (the circular not being practicable), and will resist pressure from any direction better than any of the other forms given. The masonry is built in such lengths as the strength of the rock will permit, and if found necessary, temporary timber must be set.

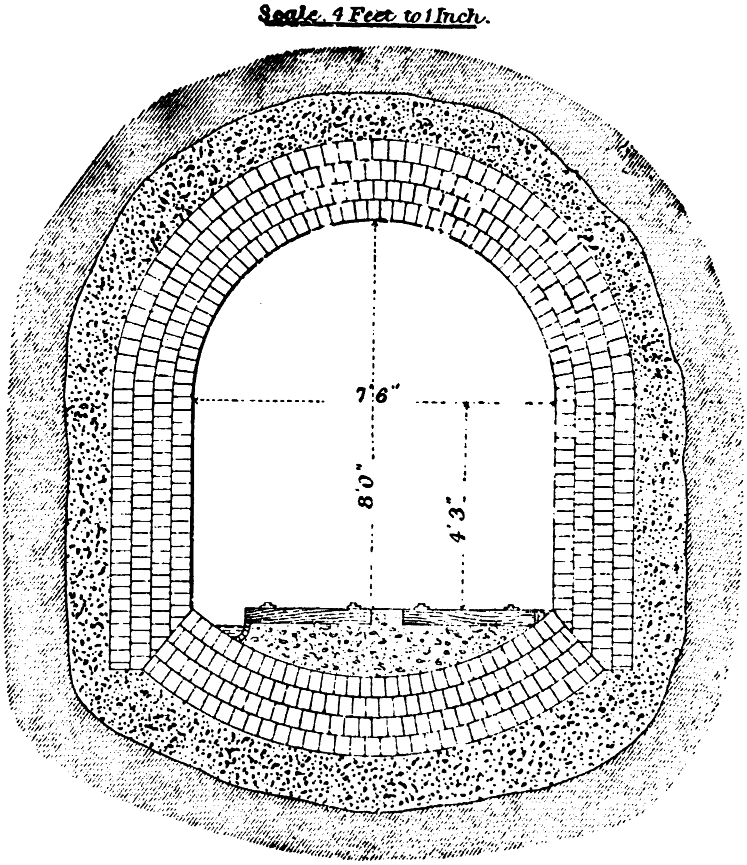


Fig. 184.—Method of securing a Roadway by Walling with Upright Sides, over which is turned a Semi-circular Arch and underneath an "Invert" or Flat Segment Arch.

Where timber is used, the length of masonry put in must be a short one, as the timber has to be taken out gradually. Very little should be removed at a time, and when removed the walling should be built rapidly so as to support the roof where previously timbered, without loss of time.

In the case of an arch having an invert, the invert is first built, the necessary rock having been taken out below the floor line and at the road sides. The invert is kept in advance of the side walls and the arching, and centering is used in building it.

The side walls are built on the invert and are kept in advance of the arching, so that the masonry may be said to proceed in three sections. A staging is erected from which the masons build the arch over the side walls. If no invert is required, the masonry proceeds in two sections, the side walls being kept in advance of the arching.

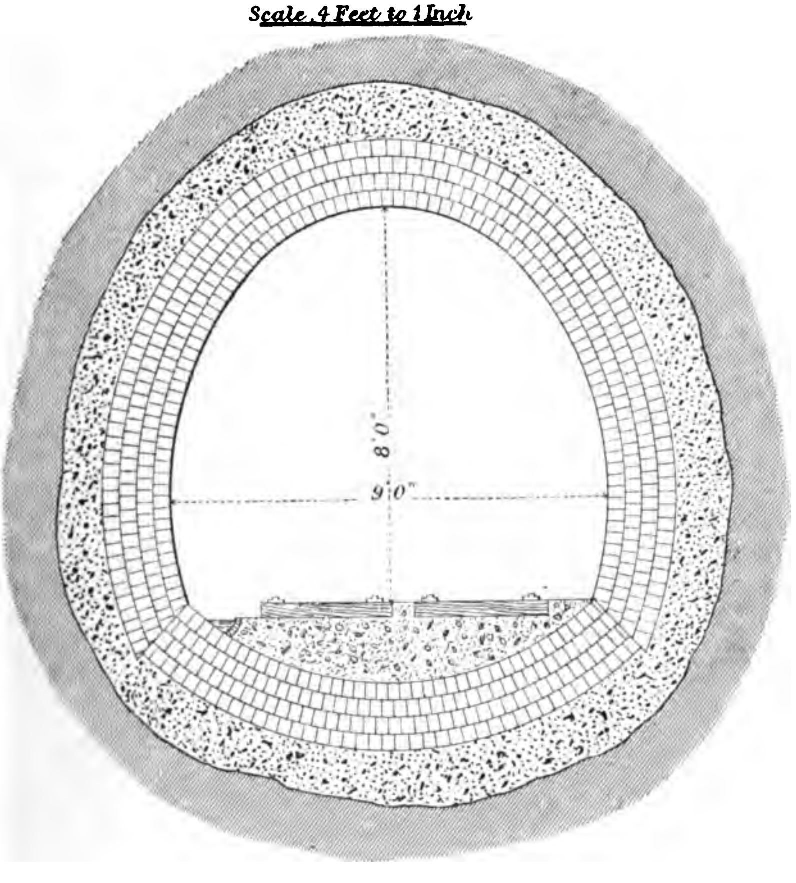


Fig. 185.—Arching for Underground Roadway where the Sides and Roof form part of an Ellipse, and the Floor an "Invert" or Flat Segment Arch.

Iron centres instead of the usual wooden ones are used for turning the invert and arch. Blocks are fixed on the floor to take those for the invert. The centres for the arching may rest on blocks slightly projecting from the side walls after they are built.

All old timber should be taken out if possible, and the space behind the walls and over the arch should be tightly packed with rough concrete, or any suitable material. Timber left behind the masonry would rot in time and leave spaces between the masonry and the rock. If the masonry is proceeding in a road to which accumulations of fire-damp may possibly extend, care must be taken to prevent open lights being taken above the arch, even if these are permitted in the roadway.

Roads requiring to be walled, as shown in Fig. 183, may have the necessary

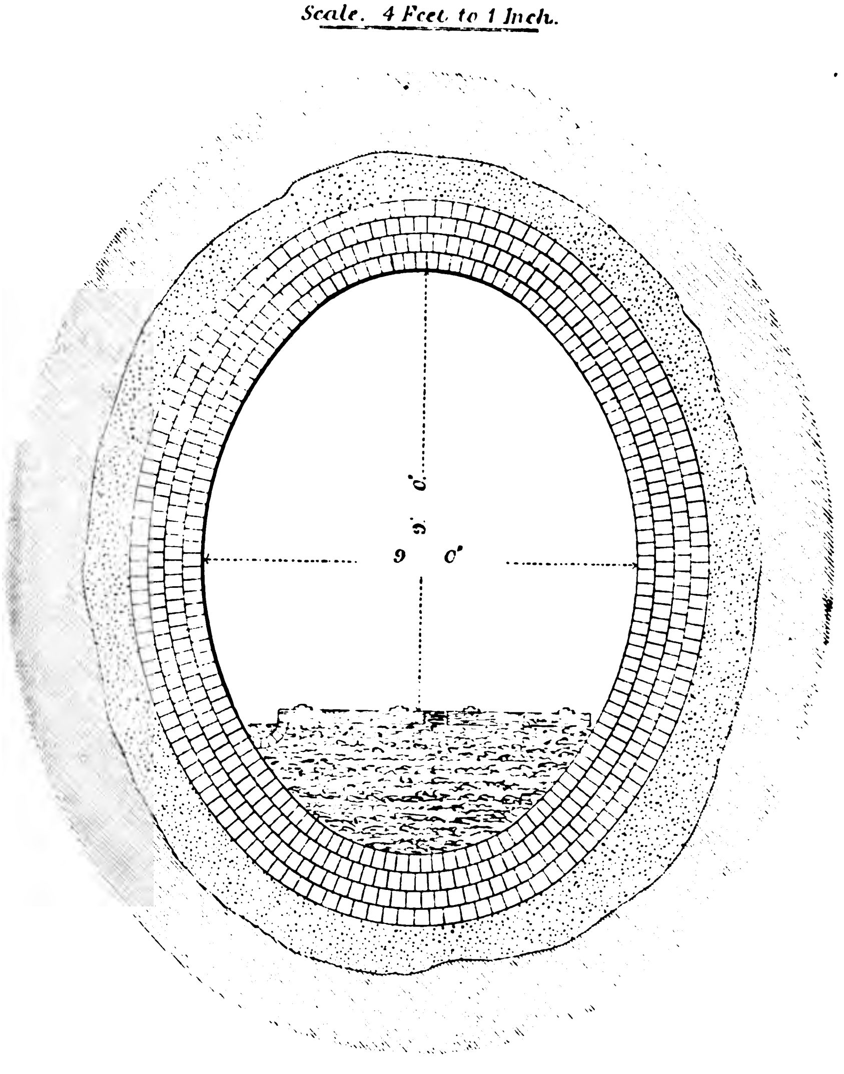


Fig. 186.—Section of Elliptically Arched Roadway, having One Foot of Sand interposed between the Arching and Stratification, and showing a Double Line of Rails laid, and Gutter formed in it.

material taken from the sides and roof, to admit of the masonry being put in during the day-time, without interfering with the usual traffic (if that be limited); so also, by using the iron centerings which do not obstruct the roadway so much as the ordinary wooden ones do, the masons may put in the walling whilst the usual work of the colliery proceeds. But if the quantity of coal passing along the road is large, it will be better to arrange that the work be done at night. Where an invert to the arch is required, the work must be carried on during the night unless the road is free from traffic.

It is better in all these operations to use hydraulic mortar, because it sets so quickly: some hydraulic mortars become solid in a quarter of an hour either in the air or under water.

Where the depth from the surface, and consequently the crush, is great, arches however strongly built are often destroyed by its force. It has been found that by packing the top and sides with sand, to a thickness of not less than one foot, the weight is distributed over the whole surface of the arch, and the walling has remained intact. The thickness of masonry required to resist a given pressure is less if packed firmly behind with sand than would be necessary if no packing be used. In the drawings Figs. 183 to 186, the different forms of arch are shown with the packing of sand filled in behind.

CHAPTER VIII.

NARROW WORK AND METHODS OF WORKING.

Shaft Pillars—Water-Levels—Cross-measure Drifts from Shafts sunk through inclined strata—Stone Drifts through faults—Longwall Method of Working—Post and Stall System—Different Arrangements of Single Road Stall Working—Double Road Stall Method and its Modifications—Method of Working and Timbering adopted at the following Collieries:—Celynen, Risca, and the Ocean—Wicket System of North Wales—The Bank System of South Yorkshire—Method of Working and Timbering adopted at the following Collieries:—Lundhill, Kiveton Park, High Park, Wearmouth, Silksworth, Florence, Great Fenton, Cannock and Rugely, Pemberton, Clifton Hall, Pendlebury, Sovereign, Radstock, Kingswood, Allanshaw, Cowdenbeath—Working thin seams in Northern France and Belgium—Square-work Working of the Staffordshire thick coal seam—Working the thick coal seams of Poland, Upper Silesia, and Bohemia—Dealing with excessively thick coal seams by Longwall and Post and Stall—Questions and Answers bearing on the subjects of the Chapter.

After the shafts have been sunk, drivings will be necessary to win the coal, and one of the first things to consider is the size of pillar or pillars to be left for the support of the shaft. If no pillar were left, but a longwall face opened at once from the shaft on either side of it, the subsidence of the roof, except in very thin seams, consequent on such proceeding would disturb the strata near the pit, and might cause injury to the shaft-walling, displace the shaft-fittings, and entail a considerable after-expense in restoring the shaft to a working condition. The size of the shaft pillar or pillars should be such that, when the coal is worked away beyond a sufficient area round, the shaft will be unaffected by the "draw"—the lateral disturbance of the strata beyond the point actually worked. The depth from the surface, the nature of the strata above and below the coal seam, as well as that of the coal itself, and the amount of dip all influence this. For any depth to 100 yards, it may be sufficient to leave a pillar 40 yards square. Adopting this size as a minimum we may fix any size of pillar for greater depths by increasing the pillar 5 yards for every 20 yards in depth, so that for a shaft 150 yards deep, we should require a pillar 52½ yards square, for a shaft 200 yards deep, 65 yards square, for a shaft 300 yards deep, 90 yards square, for a shaft 400 yards deep, 115 yards square, and so on.

The shaft should always be in the centre of the pillar or pillars left for its support, to ensure the same amount of protection on each side.

If water is likely to be met with, water-levels will be required, and these should be started some feet below the seam at the pit bottom. Roadways in the seam to be "water-level" should rise slightly, about $\frac{3}{16}$ths of an inch per yard, to allow the water to flow out to the shaft. For the purpose of ventilation, the two shafts, which are necessary to every colliery, are also connected as soon as possible by driving in the seam. Levels are usually driven on both sides of the shaft, and there may be either two or three on each side, driven parallel to one another, and about 20 or 25 yards apart. They are generally driven from 7 to 10 feet wide; if the roof be very bad, it may be desirable to make them as narrow as 5 feet. With regard to height, if the seam is thicker than 7 or 8 feet, the level is usually carried that height, and the upper portion of coal left as the roof. If the seam is less than 5 feet high, the roof is ripped down or the bottom cut to make height on the upper level, and when finished the road should be 6 or 7 feet high. The levels may be timbered or walled if the top is bad. For the purpose of ventilation they are connected every 30 or 40 yards by cross-holings driven at right angles. As soon as a fresh one is cut through, the previous one is closed by a brick or stone stopping built in it so as to keep the air well up to the face.

If the seams of coal to be won lie at a high angle of inclination, it is usual to sink the shafts below the seams and drive a cross-measure or stone drift, water level, from the shaft across the barren ground until it intercepts the seam or seams of coal. Fig. 187 is an example of this, where a cross-measure drift is shown extending from the shaft to two seams of coal which lie at an angle of 30 degrees. At the point where the drift intersects the seam of coal, water-levels are turned opposite each other from either side of the drift. For the purpose of ventilation, if a pair of drifts are not proceeding, air boxes or brattice are used as a temporary expedient, until a return air-way is driven in the seam of coal, back to the upcast shaft.

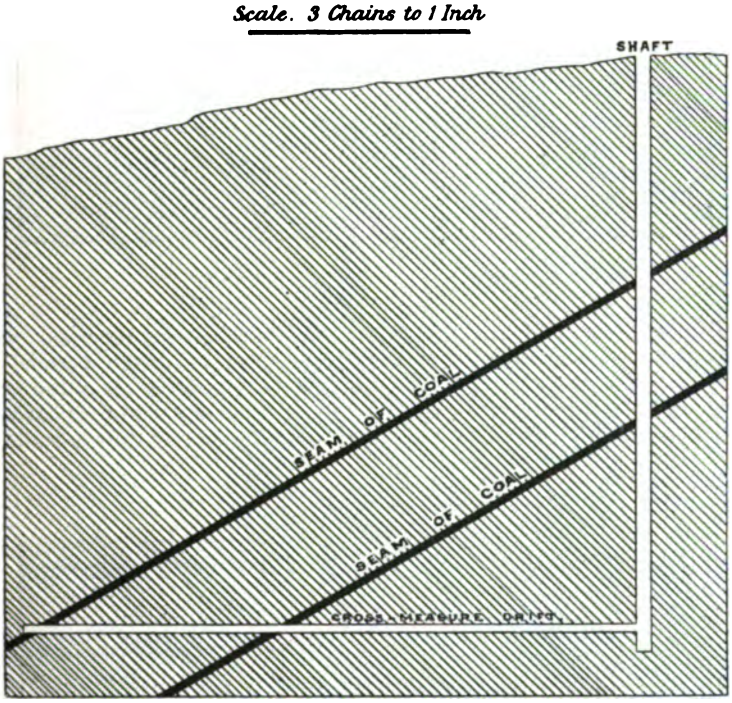


Fig. 187.—Section showing Cross-measure Drift driven from the Shaft to two Coal-seams lying at an angle of 30°.

At times it is desirable to make stone drifts from one seam to another, or through a fault to intercept the same seam of coal on its other side. Fig. 188 is intended to show an example of this kind. The workings after proceeding to the rise on No. 1 and No. 2 seams have stopped at a downthrow fault. If stone drifts be now carried at water level as shown in the two positions on the section from No. 2 seam, the upper cross-measure drift will cut the No. 1 seam on the other side of the fault, while the lower and longer one will prove the No. 2 seam on the far side of the fault, and these seams being connected by an air pit, working may be resumed on them inside the fault.

Instead of driving the two drifts as shown, the upper one might be continued water level past the point of cutting the No. 1, till it reaches the No. 2 seam, after which, to form a return air-way, the air-shaft between the seams must be sunk inside the fault, and another stone drift must be driven parallel to the first, and extending from No. 1 to No. 2 seams on opposite sides of the fault.

Scale. 3 Chains to 1 Inch.

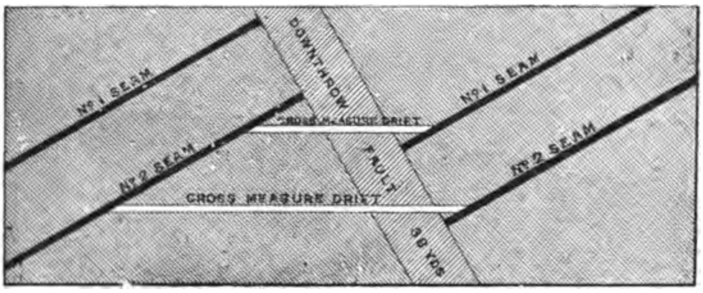
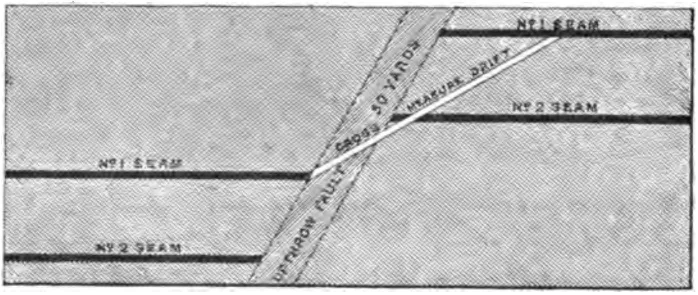


Fig. 188.—Driving through a Fault.

In Fig. 189, the workings of two seams lying level are shown as having proceeded against an upthrow fault, and here the cross-measure drift is made to rise rapidly in order to cut the seams thrown up. After the cross-measure drift has proved the two seams as shown on the section, it will be necessary that another be driven parallel to the first, either all the way, or parallel to the first through the part of its course from No. 1 to No. 2 seams on opposite sides of the fault, when the working of No. 2 seam where just proved through the fault may be continued, and an air-pit sunk to connect the two seams at the higher side of the fault.

Scale. 3 Chains to 1 Inch.



ig. 189.—Driving through a Fault.

The rock between seams may be removed upwards or downwards from the lower to the upper seam or *vice versâ*, for the purpose of making an air-shaft.

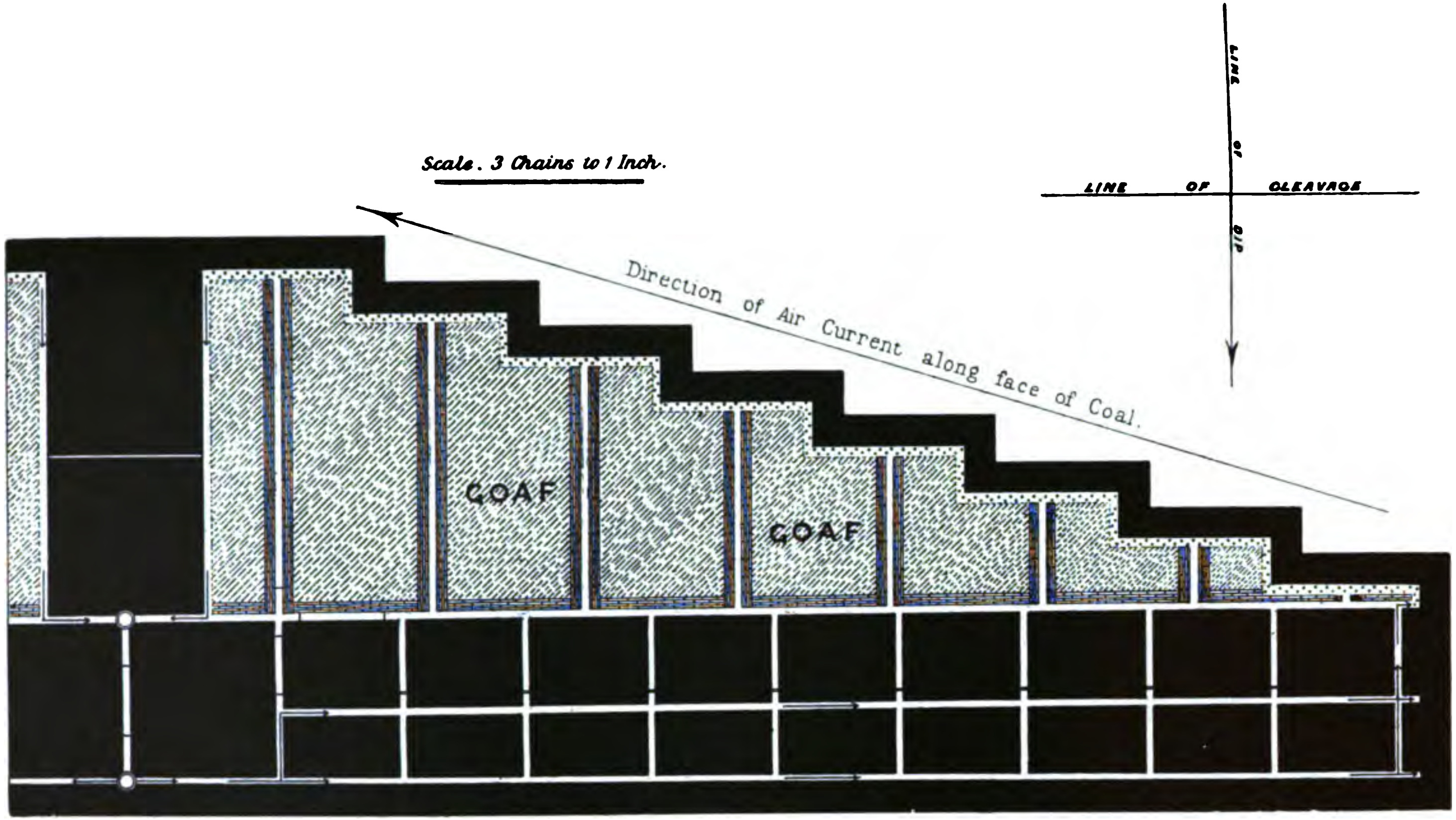

Scale. 3 Chains to 1 Inch.
LINE OF DIP
LINE OF CLEAVAGE
Direction of Air Current along face of Coal.
GOAF
GOAF

Fig. 190.—Plan showing Longwall Workings with Gob Roads advancing to the Rise and across the Cleavage.

These double communications are necessary for an intake and return air-way, and when made, the working of the No. 1 seam may be resumed inside the fault.

Although only these examples are here given, the student will readily imagine others in which the seams are cut off or interfered with by faults of different magnitudes.

The usual systems of working are what are termed "Longwall" and "Post and Stall," sometimes called "Bord and Pillar," and in Scotland "Stoop and Room." Some systems of working are practised which are modifications of these, and seams of exceptional thickness are often worked by a special method applicable only to the particular circumstances prevailing. The circumstances most favourable to Longwall are, a seam, not too thick, of rather hard coal, capable of bearing pressure, and which parts freely from the roof; a seam of coal having stone bands in it or ironstone over it to be worked with the coal, yielding material for packing. The circumstances most favourable to Post and Stall are, seams situated near the surface, the working of which on the Longwall would probably injure the buildings, but which may be worked on the Post and Stall in the whole mine, leaving the pillars, more or less robbed, to support the surface; this system is preferable also where the coal is tender under a heavy roof.

The advantages of working the Longwall where it is applicable are, a better yield of large coal, less injury to upper seams as the intermediate strata settle gradually, simplicity of working, ease of ventilating, and greater economy, for the superincumbent weight reduces the labour of "holing." These advantages are so manifest as to indicate the desirability of working all seams of usual thickness situated 100 fathoms or more below the surface on the Longwall system. There are many modifications of Longwall, and this is one of its merits; it is capable of being varied more readily than the Post and Stall to suit local circumstances. In all cases it consists of extracting all the coal at one operation, the roof settling down behind as the "face" advances. In practice it is generally found better to take out all the coal with the exception of the shaft pillars, but sometimes pillars are left between, and on each side of, the main roads. There is an advantage in letting the face advance across the cleavage of the coal, but some coals have no defined cleavage, and sometimes, even where there is a cleavage, the dip is not suitable for the face to advance across it. In Fig. 190 the gob roads are carried to the rise and across the cleavage of the coal. The distance of these gob roads apart varies, being seldom under 14 yards or over 50 yards. The roads are carried in the middle of the stall, for convenience in working the coal and in bringing it from both sides as shown on the plan, Fig. 190. The pack-walls are built as shown, and if the seam be thin, height is made by ripping either top or bottom. The gob must be packed with the rubbish yielded by the seam in being worked and by the rippings, and the closer it is packed the better will be the result. This is especially necessary in fiery seams, for it must be remembered that any portions of the waste not closely gobbed afterwards become receptacles for fire-damp, unless ventilated. At times, when the barometer is low, a portion of this gas finds its way into the roads, and is at all times a source of dread and anxiety. If the roof is bad a double row, and sometimes a treble row, of props, with lids placed over them, is kept next the face, the back ones being taken out, where this can be done with comparative safety, and re-set in front as the face advances. In some cases a double row of chocks is used instead of props. Often the water levels from the shaft, instead of running in a line with the cleavage, as shown in Fig. 190, will cross it at some angle; if this be a right angle and it is desirable to carry the "face" across the cleavage, Fig. 191 shows the method usually adopted.

The face thus advances against the cleavage, and if the dip and rise be rather great there is an advantage in keeping the gob road at or near one end of the

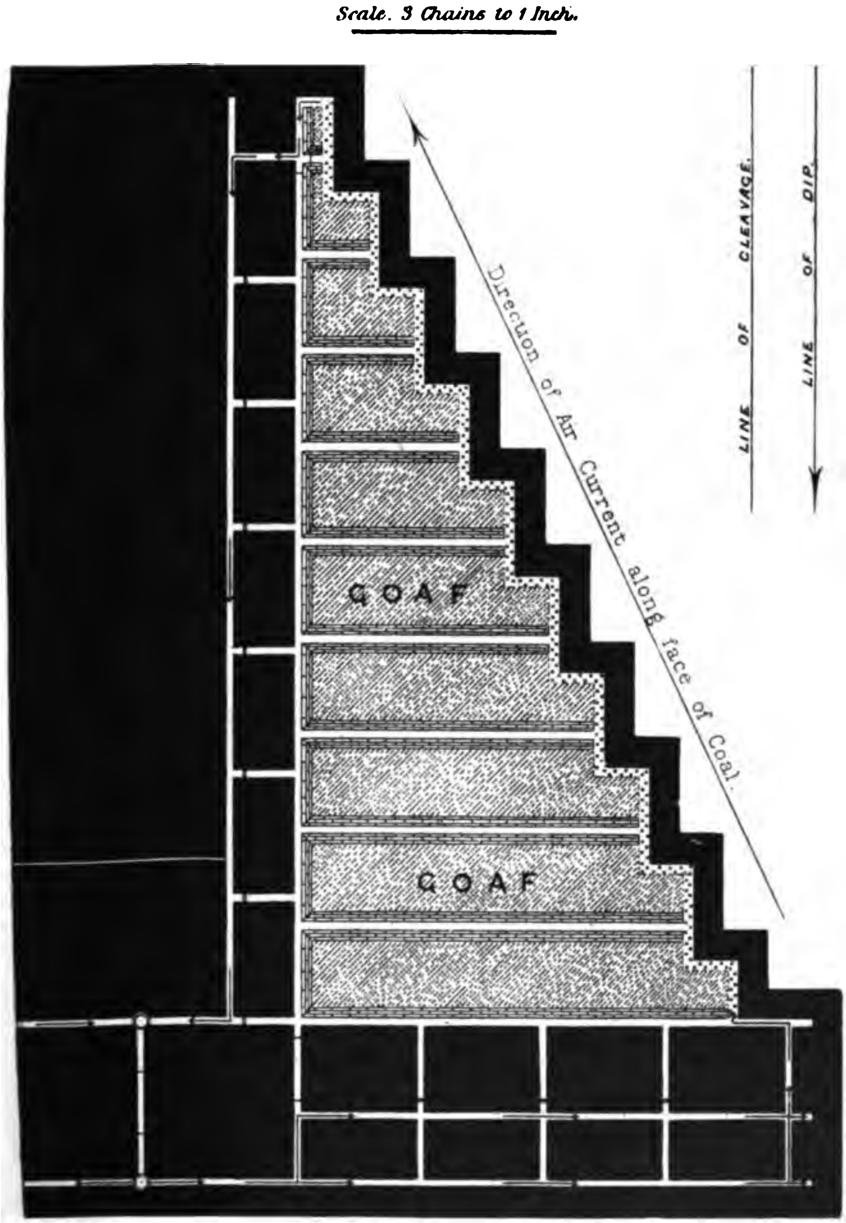

Scale. 3 Chains to 1 Inch.
Direction of Air Current along face of Coal.
LINE OF CLEAVAGE.
LINE OF DIP.
GOAF
GOAF

Fig. 191.—Plan showing Longwall Workings with Gob Roads advancing Level Course and across the Cleavage.

working face instead of in the middle, so that the coal may be brought "down hill" to the road. In very tender seams, there is an advantage in working the face *with* the cleavage, instead of *against* it, and frequently, where there is no cleavage in the coal, the face, instead of being marked out in steps, is connected from one gob road to another in curved lines. If ventilation alone be considered, it is of the utmost importance to have the whole face in a straight or gradually curved line, as this arrangement offers less obstruction to the passage of the air in its course along the face. Whilst fully admitting the importance of ventilation, other points have to be considered and kept in view, and it is not always practicable to keep a face of this shape advancing.

Where there is plenty of capital at command, and the area to be worked over is not great, the levels may, with economy, be driven to the boundary of the royalty and the coal worked back towards the shaft, goaf, which is the most dangerous part of fiery mines, being thus left behind.

The Post and Stall system of working is largely practised in the Northern collieries. A set of excavations are driven through the coal parallel with one another, and at certain regular intervals, leaving a rib of coal between them. At right angles to these another set of excavations is driven through the coal, also parallel with one another, and at regular intervals. The effect of these excavations is to leave rectangular blocks of coal which support the roof. Subsequently these blocks of coal are removed. The excavations in the coal alluded to are called *bords* and *headways*. The bords are driven as wide as the strength of the roof admits of, usually 4 or 5 yards wide, and these are carried against or at right angles to the cleavages of the coal. The headways are driven 2 to 3 yards wide in the direction of the planes of cleavage. The blocks of coal between these cross drivings are called pillars, and their size varies in different collieries, and according to the depth from the surface, the usual range being from 20 to 50 yards long, by from 10 to 40 yards wide. The chief consideration in determining the size of pillar is the thickness of cover over it. Both *thrust* and *creep* are caused by insufficient pillars being left. When the roof and floor are of hard, unyielding material, and the pillar of coal left in the first working is too small to support the pressure thrown upon it, the pillar cracks and breaks up, large pieces falling away from it, and finally it is crushed into small coal. This lets down the roof and chokes up the workings, and the action of this pressure is called *thrust*. Again, if the material composing the floor, or both floor and roof, is weak and soft, and the pillars left are too small, the downward pressure upon the pillars causes the floor to rise, and the roof, if of a yielding nature, sinks, the roof and floor thus approaching each other. This is known as *creep*, and is shown in Fig. 192. Indeed, in the majority of mines over 150 fathoms in depth, whatever size the pillars are, if the floor is of fireclay, or any material rather soft, there is trouble in keeping roads and airways open. But precisely the same thing occurs in Longwall working.

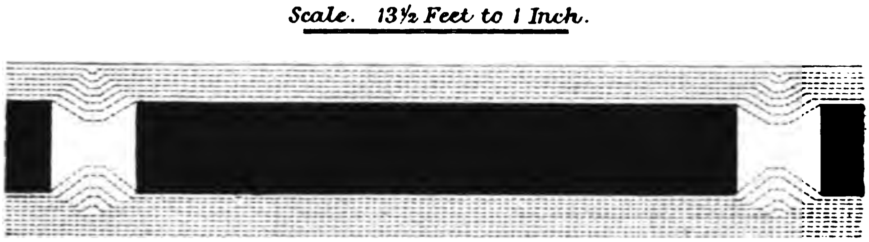


Fig. 192.—Effect of "Creep" on the Roadways of a Mine.

The process of removing the pillars is called *working the broken*, and it was formerly the custom not to begin working the broken until the workings in the whole mine had been carried to their destination, but the leaving of pillars unworked for long periods increases the danger of thrust and creep, and it is now usual to carry on the two operations simultaneously, the work in the broken following closely upon the work in the whole coal.

There are many ways of working away the pillars. Where these are small they may be removed at one operation, the whole of the coal being taken out and the road formed in the middle of the pillar as the "lift" advances. Circumstances must be favourable to admit of this plan. Again, a lift of half the pillar may be taken off throughout its length, the other half being the subject of another lift. If the seam lies flat, the pillar may be divided into four lifts, proceeding from the four corners of the pillar.

In other cases a narrow place is driven across, splitting the pillar in two, and then the two portions of coal left at the sides are brought simultaneously back as "lifts."

It will generally be found that a lift of six yards wide is quite sufficient to take at one time, and in working the coal in the whole mine, pillars should be left of a size suitable to resist the effects of thrust and creep, and yet easily arranged for their ultimate removal. The roof must be good that allows of a lift of 6 yards wide being taken 20 or 25 yards in distance. In the North of England the practice is to timber the waste formed in working any lift, the road being carried on the side next the coal. When the lift is finished, the rails are taken out; and, beginning at the inside end, the deputy takes out the timber, or "draws the jud," as it is called.

In other parts, a pack-wall is built on the waste side of the road which is carried next the coal, as in the last-mentioned case, but here the waste is not kept open until the lift is finished. Timber is kept next the face for its security, the pack-wall and the solid coal protect the road, and the other portion of the waste is allowed to fall as the lift proceeds.

Figs. 193 and 194 show lifts of 6 yards wide being taken out where they have 25 yards to proceed before finishing. The line of face is kept at a good angle, following the whole mine workings, so that the pressure at the goaf edge assists the working (by a steady uniform breaking down of the top) without injuring the coal, and the line of goaf is continuously maintained. It is better that these lifts should proceed only from one side of the pillar, as shown on the plans; and the ventilation should be arranged so that no air, after passing through "broken" workings, will reach workmen in other parts, but be conveyed to the returns direct.

The two important points to be kept in view, in any arrangement for working the broken, are: (1) Working away all the coal in a good marketable condition without leaving stumps to interfere with the uniformity of the top breaking down along the goaf edge; and (2) To carry on this working with the greatest margin of safety to the workmen engaged in the operation. Clearly, any arrangement by which the lifts are taken off at such angles that any one of them may have goaf on either side will result in the workman on that lift getting daily into a more dangerous position. In Figs. 193 and 194 it will be observed that the lifts have goaf only on one side, and the solid on the other affords protection to the workmen.

Further reference will be made to pillar working later on, when the practice at some collieries will be given in detail.

It was at one time thought that by laying out the workings into *panels* of moderate area, a certain amount of protection from thrust and creep was afforded.

By what is called *panel* working, one district is separated from another by a rib of coal which may be 40, 50, or 60 yards wide. The advocates of the system further claimed for it protection in times of explosion of inflammable gas, their contention being that the damage resulting would be confined to the district in which the explosion occurred.

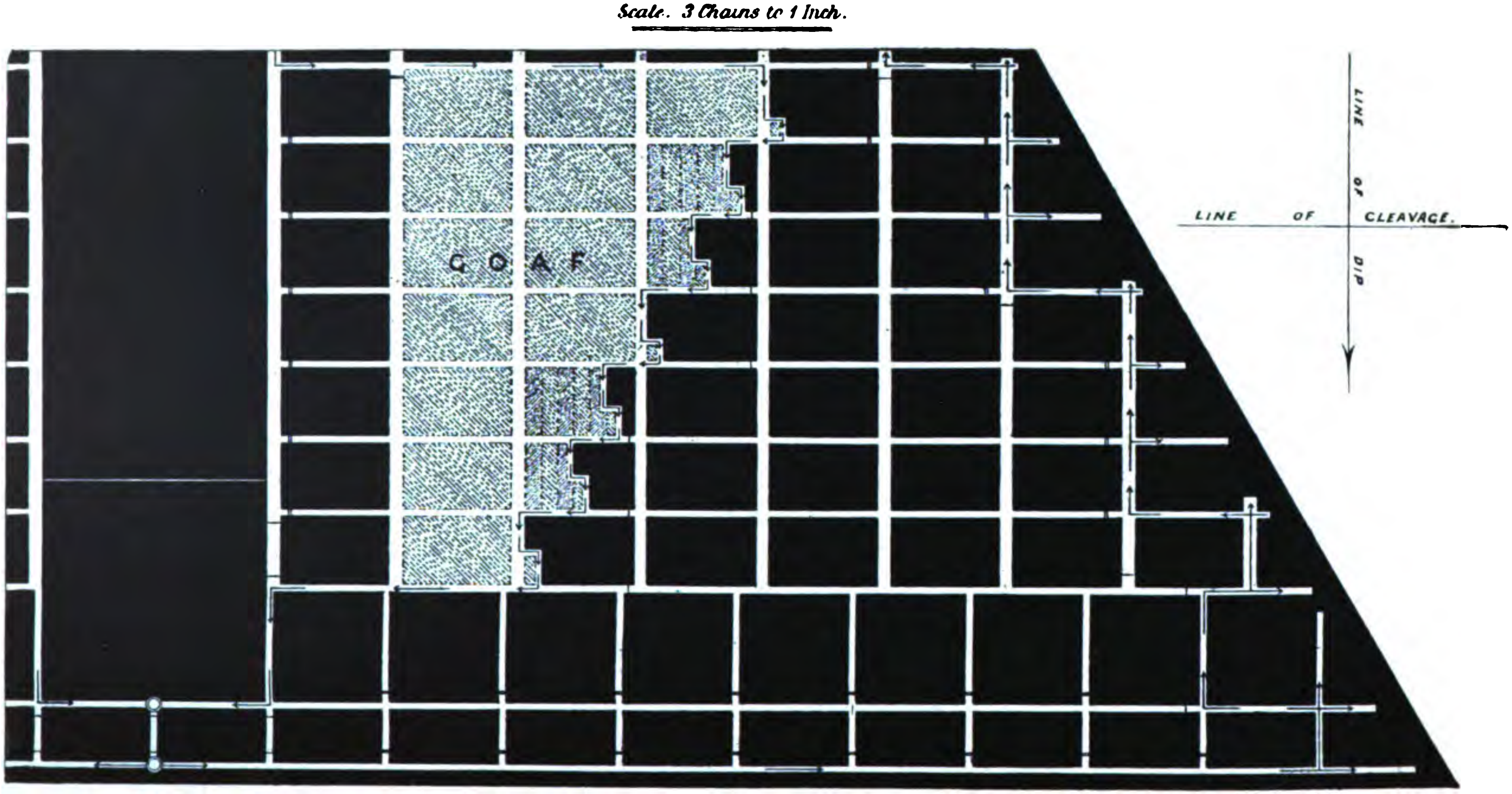


Fig. 193.—Post and Stall system of Working where the Levels are proceeding in the direction of the Cleavage.

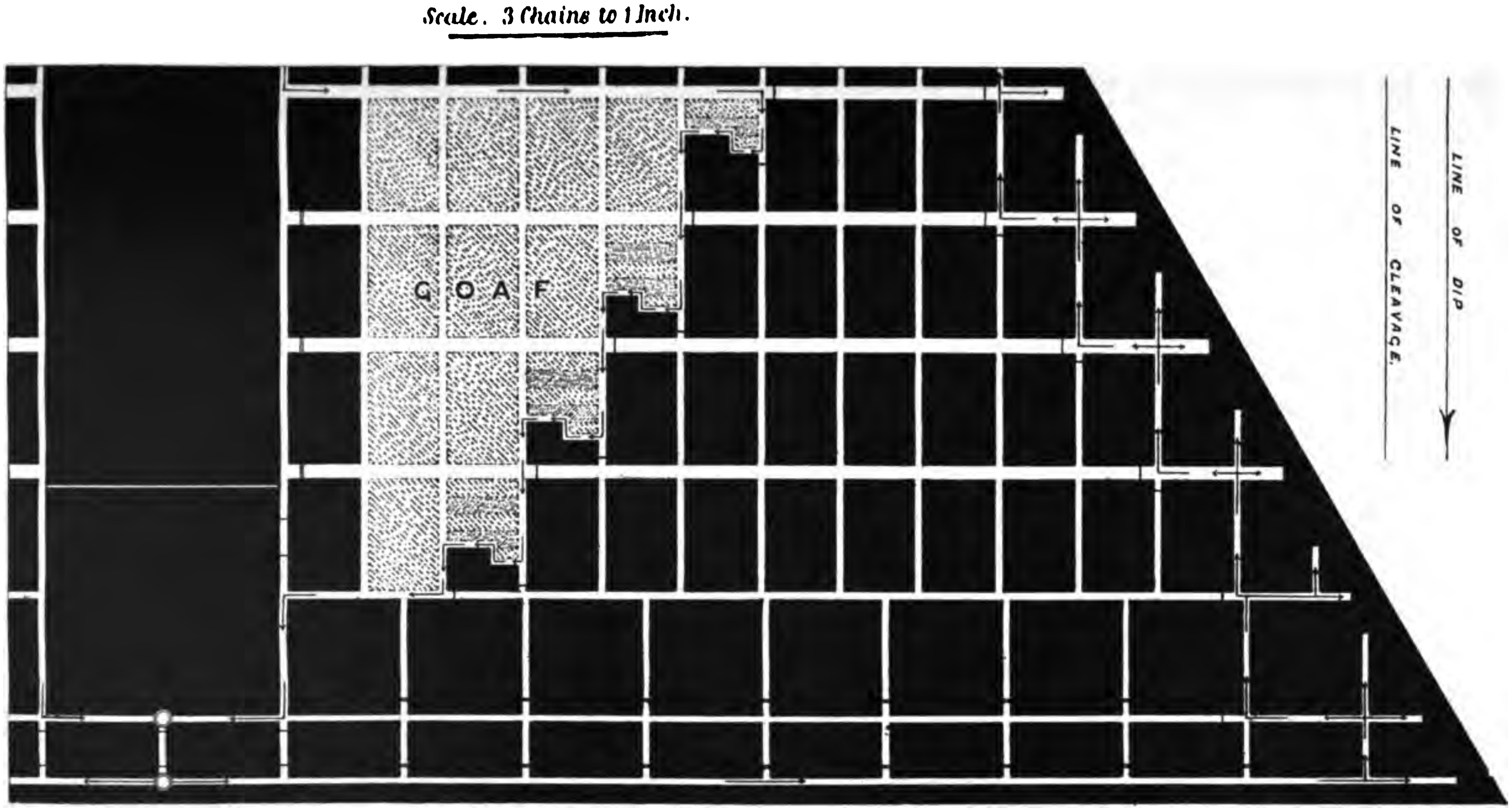


Fig. 194.—Post and Stall System of Working where the Levels are proceeding across the Planes of Cleavage.

A rib 40, 50 or 60 yards will have very little effect in preventing thrust and creep, if the pillars be left too small, and any good resulting in that direction must be very limited. As before stated, the only effectual means of guarding against thrust and creep, is by leaving pillars of a suitable size.

Then again, the rib of coal has very little effect on the damage caused by explosions of fire-damp, for in all cases it is proved, by the blown-out stoppings, disarranged timber, and other indications, that an explosion of this kind, wherever it occurs in the workings, usually takes a course against the intake air-current, and as long as there is sufficient gas to mix explosively with the air, so long will the ignited mass continue its course, often ending only at the downcast shaft. A judicious splitting of the air-currents, then, affords more protection from the effects of an explosion than any system of panel working.

Fig. 193 explains the post and stall system of working where the levels have followed the direction of the cleavage and where the working of the broken follows closely upon the working in the whole mine.

If the levels have crossed the planes of cleavage at right angles, the workings would appear as shown in Fig. 194.

In seams having a high inclination it may be required to set away a pair of winning places, or drivings, in a direction neither headways course nor at right angles to it, but midway between the two, and these are called "cross-cuts." They are usually driven by bearings, to keep them in true alignment, and the "marks" may consist of three or more wooden plugs with iron crooks in them, driven into the roof in the required direction. The marks are generally put up by the aid of a surveying dial, and the driving is kept straight by hanging three plumb bobs from the crooks in the plugs and looking in their direction to a light in the face.

In arranging the ventilation of a colliery, air-crossings, doors, brattice cloth, stoppings, and regulators are required. Generally speaking, the aim should be to have as few of these as possible, and to keep doors off main or important roads. In any plans of workings shown, the arrows indicate the direction of the air-current; a **X** indicates an air-crossing; a single line across the road a door, either of brattice cloth or wood; a double line across the road a permanent stopping.

Two methods of working peculiar to South Wales are the "Single Road" and the "Double Road Stall" systems.

Both systems are capable of modification to suit circumstances. In the Single Stall system shown in the drawing Fig. 195, the usual conditions prevail. Instead of driving a pair of narrow levels from the shaft, which from the difficulty of doing so are saddled with a heavy yardage or driving price, a more economical level is formed. A face sufficiently wide to hold the rubbish yielded in working the seam, and usually about 8 yards, is pushed bodily forward on either side of the shaft. The road is made next the coal on the rise side, and an air-course next the coal on the low side, the space between the road and air-course being closely packed with any rubbish yielded in the driving and the ripping of the roadway. A facing of the larger and stronger stones from the ripping of the top or the bottom, as the case may be, is used to form the road side, and when needed the road is timbered. When these levels have proceeded far enough from the shaft to form the shaft pillars, headings are turned, one on either side of the shaft, which are driven to the full rise of the seam. These headings, like the level, are driven 8 yards wide, and have the road formed on one side and the air-course on the other of the advancing face. The heading road is formed on the side farthest from the shaft, and out of it at intervals of 24 yards stalls are turned, as shown on the drawing, and driven level course or parallel with the main level. These stalls are turned narrow from the heading, being 6 feet wide, and are continued of this width for a few yards in distance, after which they are gradually opened to their full width of 12 yards, as shown on the plan, Fig. 195, but the width of these single road stalls varies according to the amount of rubbish yielded by the seam, and the size of pillar intended to be left; it also varies according to depth from the surface and the nature of the roof and floor.

By this time the main level has advanced sufficiently (dealing with one side of the shaft, as the operations on the two sides are similar) to allow of a narrow driving being made from it to the point in the low side of the stall where it is first opened to its full width. This forms an air-way by which the air passes from the main level into the stall. As the stall is now driven on, the road is made next the coal on the rise side, and an air-course formed next the coal on the lower side. The road thus forms a straight line to the heading, and the air-course is really a continuation at right angles of the air-course connecting the stall with the main level. The pillar of coal left next the main level and the heading is about 20 yards long, and remains to protect them until they are finished working.

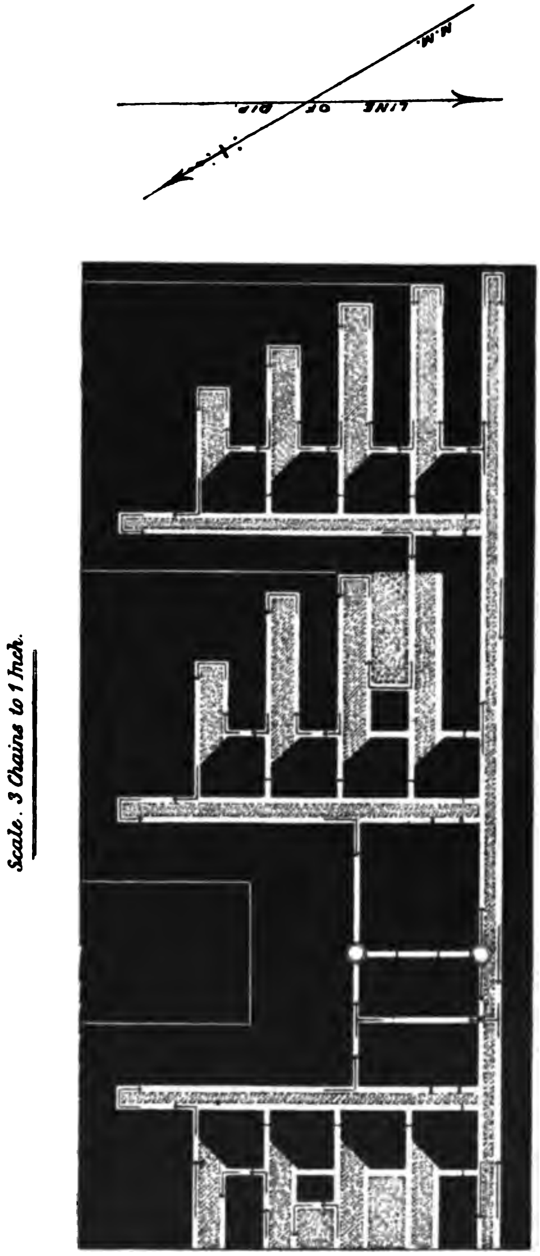


Fig. 195.—Plan showing usual Arrangement of Single Road Stall System of Working Coal in South Wales.

The stall is continued till it has proceeded about 75 yards from the heading, the space between the air-course and road being packed with rubbish. The coal is brought in boxes along the 12 yards of face to the tram at the end of the road, whence it is conveyed by horses.

When the main level has advanced sufficiently, another heading similar to the first is turned. There are about 90 yards of coal between the two headings. The stall which has reached a point within 15 yards of the second heading is now continued narrow through to this second heading and becomes an air-course, the object being to shorten the distance the air travels and leave a 15-yard pillar for the protection of the second heading on that side.

The first heading having advanced and other stalls having been turned out of it, the pillar belonging to this first-driven stall is now worked back towards the heading. This pillar consists of the 12 yards of coal next the stall road on the rise side and lying between it and the air-course above it. The coal is brought in iron boxes down-hill along the face to the tram, which is brought along the same line of rails as used in driving the stall.

In Fig. 195 this pillar is shown as being partly worked back. When the pillar coal is all taken out to the air-course (which it is necessary to drive between *all* the stalls) it is finished, the stump next the heading being left until the heading is finished, so as to protect it whilst working, but afterwards all the stumps in that particular heading, commencing at the top or innermost one, may be worked off.

Unless left a sufficient size, the stumps are too much crushed to be worth working. Not much timber is used in well-packed stalls where the roof is strong, but for the most part what is used there and in the pillars is lost. An effort ought to be made to recover at least the timber in the pillar working.

This process is continued, headings being turned at regular distances out of the main level, and these single road stalls turned out of the headings. A rib of coal is left next the main level for its protection during completion, after which it may be worked back from the far end.

Fig. 196 is a drawing of another method of the Single Road Stall system. Here the main level is shown as a narrow driving, but it may be driven precisely as in the last case, shown in Fig. 195. Where the roof is very good, the headings may be driven at greater distances apart from the main level, and instead of having the stalls turned from one side of the headings, they may be turned on both sides, as shown in Fig. 196.

In this case there is a distance of 200 yards between the two headings, and as stalls are to be turned on either side, the best arrangement is to make the heading about 24 yards wide, and to form a road next the coal on either side.

This would allow of two men and a lad, or two men and two lads, working at the face, instead of one man and a lad when only eight yards are taken out. The two roads forming the heading are turned narrow off the main level for a short distance and then opened out towards each other; when connected, the face is carried on. This leaves a stump next the main level. The method of turning and carrying the stalls is the same as that described for Fig. 195, but when they have advanced to a point midway between the headings they meet if the stalls opposite each other are started from the headings at the same time and the rate of progress in each is the same. The pillars are then worked back as in the last case. The chief point of difference consists in the greater distance the stalls are carried, owing to the excellent roof in this instance. Other modifications may suggest themselves as the occasion requires. Sometimes, instead of opening out the stall to its full width on the low side of the road and bringing back the pillar along the rise side, the reverse of this is done. The stall, after being turned narrow, is gradually opened its full width on the rise side of the road, the pillar being brought back along the low side. Again, where the inclination is great, instead of evenly dividing the coal to be taken out between the stall and the

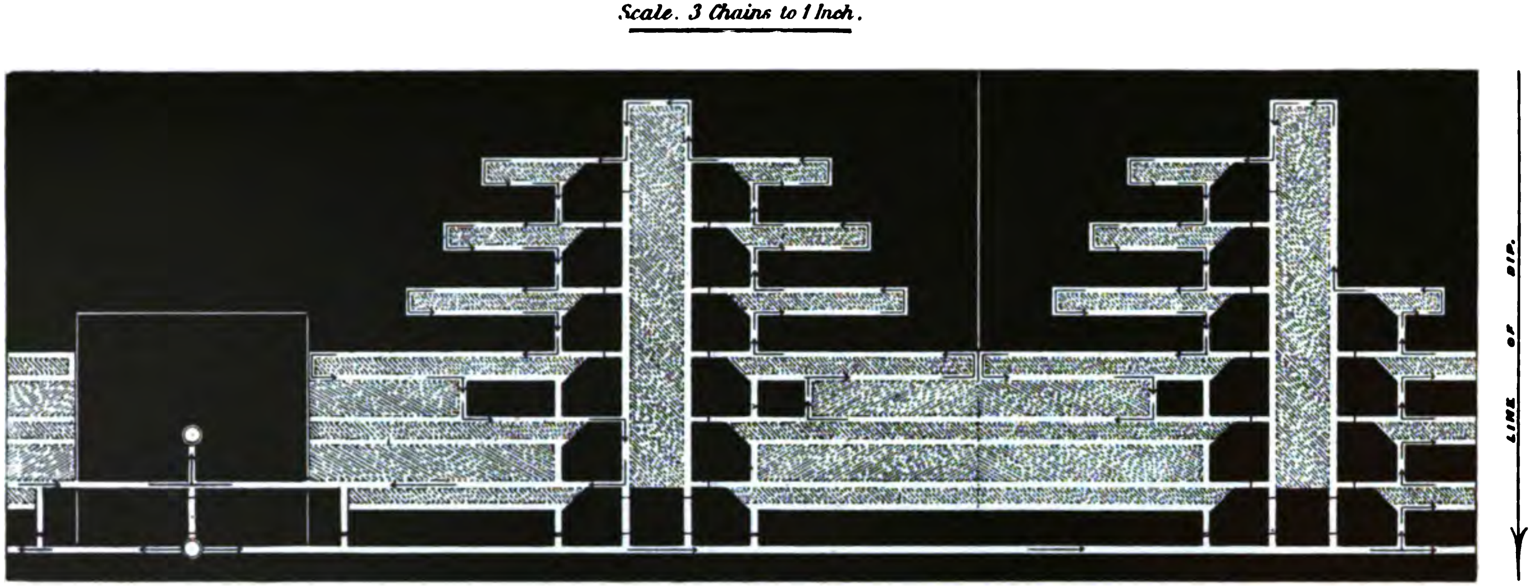


Fig. 196.—Arrangement of the Single Road Stall System of Working Coal in South Wales, where the Roof is very Good.

pillar, some advantage may result from taking, say, 14 or 15 yards from the rise side of the road and 8 or 9 yards from the low side.

One man and a boy usually work together in a single stall road, the man receiving a tonnage price which includes timbering at the face, tramming the coal, and making the road. He pays the boy a daily wage, chiefly to fill the trams for him.

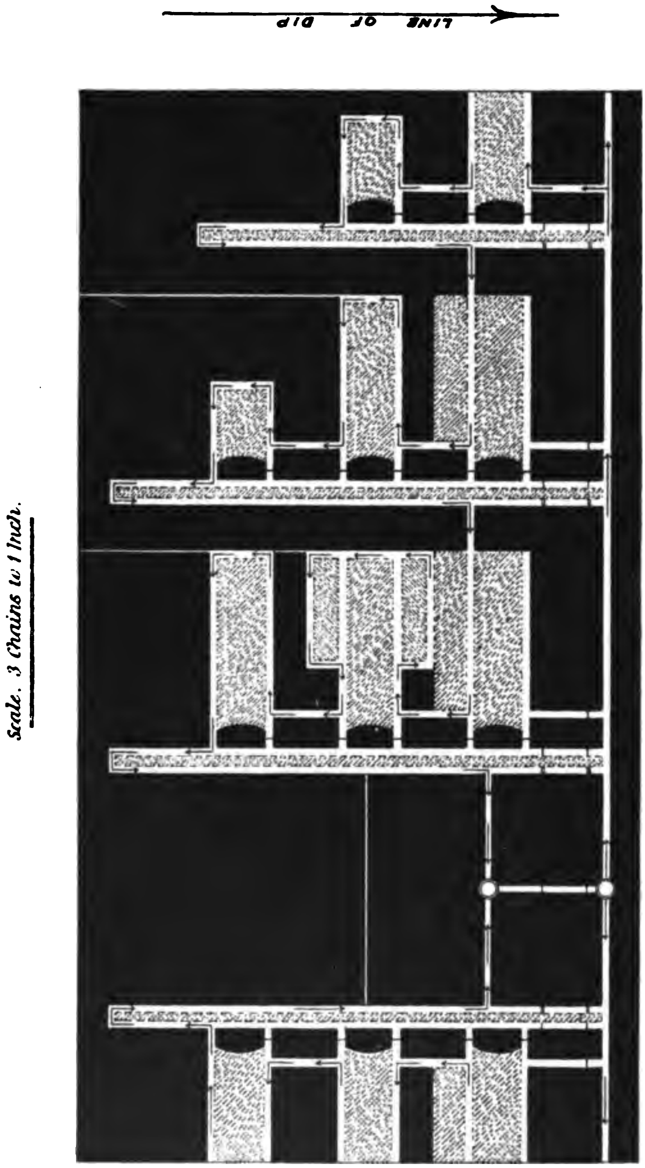


Fig. 197.—Plan showing the Double Stall System of Working Coal in South Wales.

A reference to Fig. 197, and a comparison between it and Figs. 195 and

196, show at once the chief points of difference between the Single Road Stall and the Double Road Stall systems of working. The latter, as shown in Fig. 197, has a main level from the shafts carried as a narrow driving, but it may be carried as shown in Fig. 195. The headings are the same as shown there, and are the same distances apart. Instead, however, of turning a single road out of the headings, two roads, separated by 18 yards of coal at the commencement, are turned, and after proceeding a short distance they are opened out towards each other, and when connected the face advances, being now 22 yards wide. The roads are carried next the solid coal one on each side of the excavated coal, and after proceeding to a point within 15 yards of the next heading, one of the roads (usually the lower one) is continued to the heading as a narrow driving. This afterwards forms the air-way, until another is made above it. Two men and two boys work in these stalls, and directly the stall has reached the far end, they divide, a man and a boy taking out half the pillar (5½ yards) below them, the other man and boy taking half the pillar or a similar distance on the high side. Both parties work their portion of the pillar backwards and finish about the same time. The stumps are left to protect the headings until the last, as are also the ribs of coal next the main level.

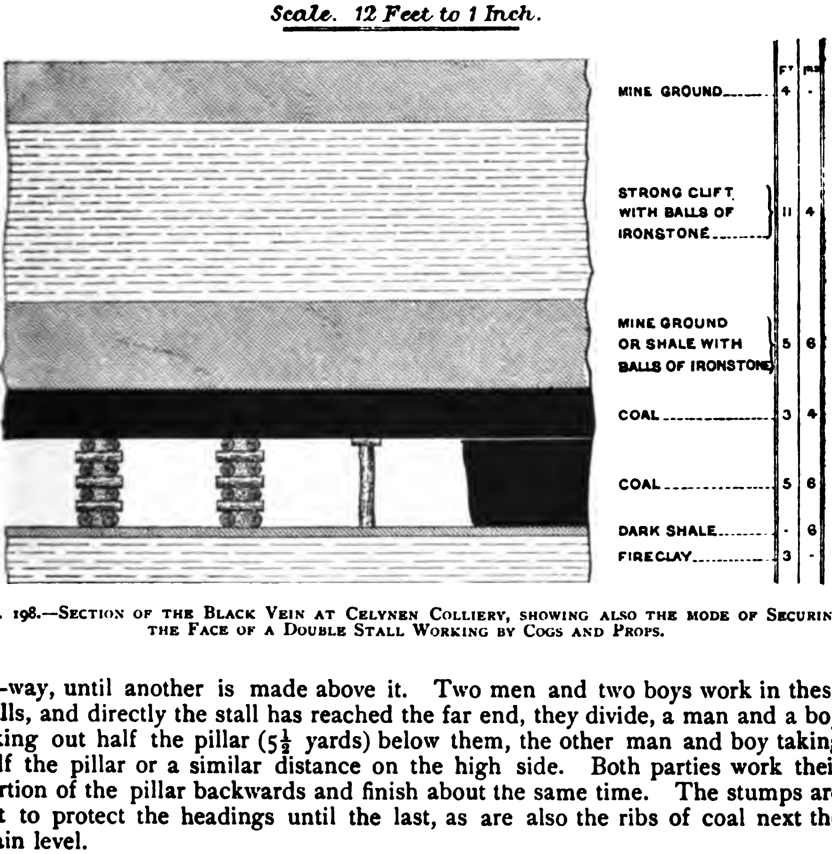

Scale. 12 Feet to 1 Inch.
FT
INS
MINE GROUND 4 -
STRONG CLIFT WITH BALLS OF IRONSTONE 11 4
MINE GROUND OR SHALE WITH BALLS OF IRONSTONE 5 6
COAL 3 4
COAL 5 6
DARK SHALE - 6
FIRECLAY 3 -

Fig. 198.—Section of the Black Vein at Celynen Colliery, showing also the mode of Securing the Face of a Double Stall Working by Cogs and Props.

The Double Stall is open to modification, as is also the Single Stall, and if the roof is good an arrangement of Double Stalls similar to that for Single Stalls shown in Fig. 196 may be adopted.

The great objection to these methods of working is in the formation of detached pieces of goaf, which afterwards cause injury to the stumps of coal left between them. With detached goaves there cannot be uniformity in the breaking of the top, a point which should be always aimed at in pillar

proposed in 1881 to sink pits from this thin seam to the goaves and so drain off the gas.

The stalls in the next pair of headings are driven in exactly the same way as the last, so that when the headings are finished and stowed up, there is a rib of coal 5 yards in width separating the goaf formed in one pair of headings from that formed in another. This rib is lost. Pairs of levels, parallel to the shaft levels, and on the rise side of them, are driven at intervals of 315 yards, so as to leave a barrier of coal 40 yards thick between the upper end of the headings and the next level above. The same system of headings and stalls is driven from the other levels above. After the levels have reached the boundary and the coal is worked out from all the headings, the pillars between them are worked back.

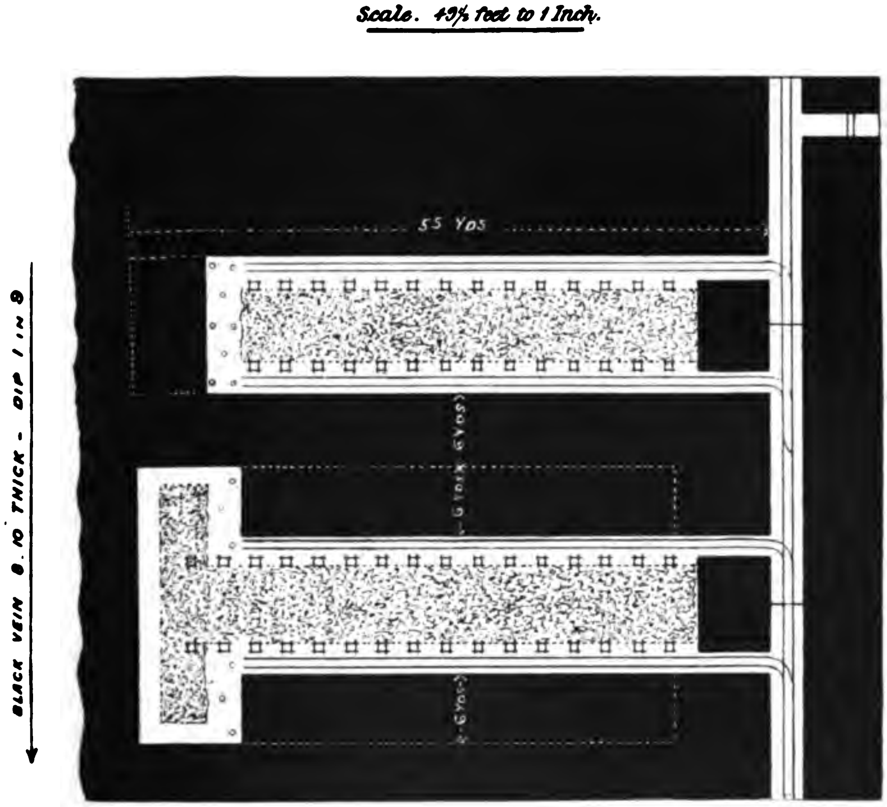

Scale. 49⅔ feet to 1 Inch.
55 Yds
6 Yds
6 Yds
6 Yds
BLACK VEIN 8.10' THICK - DIP 1 IN 9

Fig. 200.—Arrangement of the Broken in the Double Stall Workings on the Black Vein at Celynen Colliery.

Fig. 200 is an enlarged plan showing two stalls, as worked at Celynen, and is designed to indicate more clearly the details of working the stalls forward and the pillars backward, with the timber, &c., used in the operation.

The props and collars used are notched in the Welsh style (see Figs. 181 and 182), and these sets of timber, not shown on the drawing (for clearness), are placed at intervals of from 3 to 6 feet along the road from the heading, and the tram rails run between each pair of props. On the waste side of the road cogs are placed at intervals of 6 feet, and these are shown on the plan Fig. 200. Between the cogs old props are stretched horizontally, and are laid so as to protect the road from falls of small coal and rubbish from the waste. The cogs are built of old props and are from 2 to 3 feet square, being filled up with small coal in the centre. Any bad roof between the cogs is secured by props having lids. The roof is supported along the face by a line of props 6 feet apart, another row having first been placed behind and three feet from the other row.

The props are 10 inches in diameter, are set with their thick end upward, and the lids over them consist of old props. Wherever the roof or coal falls close to the face small stumps of coal are left to support the roof. The small coal made in the working is gobbed, the roof gradually settling down over it. No attempt is made to recover the props supporting the roof along the face, when driving the stall forward. In bringing the pillar back on either side of the stall, however, the props at the face, and the sets of timber and cogs in the road are taken out, and if, as is often the case, a bad roof necessitates the top coal being left for a roof in driving the stall forward, a considerable portion of this coal is got in working the pillars backward. The props, sets of timber, and chocks in the stalls are all set by the collier, and he also draws all that are afterwards recovered, receiving 1*d.* a prop for doing so. The custom is for the colliers to pack the stall-roads behind them when bringing back the pillars. Four men here work in a stall in each shift, and they get about 3 tons of coal each in the shift. In bringing the pillars back they divide into two companies, each two men having 6 yards of face in returning. In 1881, the colliers were paid 1*s.* 6*d.* for large and 5¾*d.* per ton for small, and these prices included propping, cogging, and stowing. For that portion of the roads driven narrow from the heading, they were paid 1*s.* 11*d.* per yard, but no yardage was paid after the two roads were joined and the place opened. The colliers do not require to hole under the coal; the stalls are shorn on both sides and the coal comes off in slabs.

The 5 feet 6 inches of coal under the top coal is not so good as that above it. Where the roof is sound enough to admit of it, the top coal is worked and 2 feet of bottom coal left on, except in the roads. When this is done, 4 inches of inferior roof coal is left on and there are lids put on the head-pieces, not extending from one head-piece to another, but projecting or overhanging equally on either side of the head-piece which supports it. In all other respects the working is the same as when the top coal is left on.

There is no regulation in force as to the distances between props and sets of timber on the main roads, but there is an understanding that these are not to be greater than 4 feet, and they are placed closer together where required.

The sets have the Welsh notch, the props and collars are 8 in. or 9 in. thick. The height of the roads to the inside of the collar is 7 feet, the width at the top is 6 feet and at the bottom 10½ feet. The object of this form of timbering is to resist the side pressure, which at the depth of the workings is very great.

A considerable amount of cover lies over the coal where worked, for although the shafts are only 354 yards deep the workings go immediately under the mountains and have as much as 700 yards of cover.

A special class of men timber the main roads, and these work during the day-time. Two men work together, and in 1881 were paid 1*s.* 8*d.* a set of 9-foot timber, and in addition they received 1*s.* 6*d.* per ton for the coal taken down from the roof. The cost for timbering at this Colliery was 9*d.* a ton on large coal and 7*d.* per ton if reckoned on the gross output. The price paid for timber at the Colliery was 23*s.* 6*d.* per ton.

At Risca Colliery, near Newport, also in Monmouthshire, the same seam, viz., the Black Vein, is worked by Longwall.

The Colliery began working in the same year as Celynen, and in 1880 an explosion occurred here. An upcast and a downcast shaft, each 17½ feet in diameter, are sunk to a depth of 284 yards, at which depth the Black Vein is reached. The downcast is used as a winding shaft, having single-decked cages. The cages carry two trams, one in front of the other, each holding about 18 cwt. of coal. In 1881, the average daily output was 1,200 tons, the day consisting of 16 hours. The shaft is fitted with six wire rope conductors, two of which pass through guides on the outside of each cage, and two in the middle, between the cages. The last two prevent the cages from touching each other. As they pass at meetings there is only the thickness of the rope between the cages. A pumping-engine lifts from 600 to 700 gallons of water per minute in this shaft, the water coming from old workings to the rise. The other pit is used only for ventilation, which is caused by a Guibal fan 40 feet in diameter. In 1881 a ventilating current of 180,000 cubic feet passed through the workings with a 3·5-inch water-gauge. The velocity of the current in the main air-way was 20 feet per second.

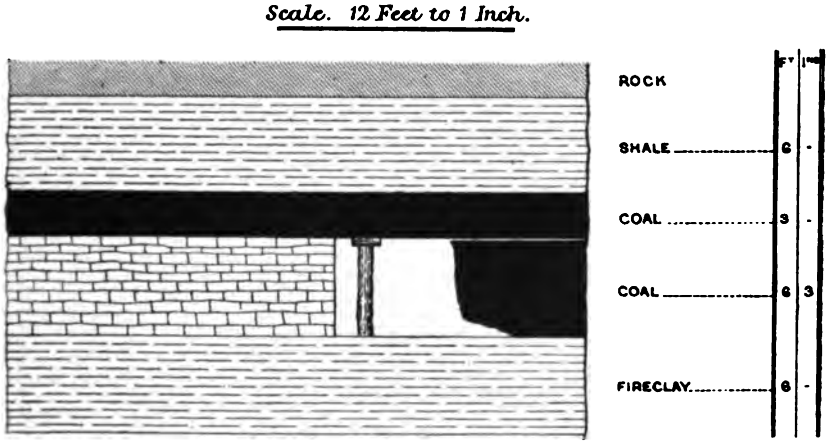


Fig. 201.—Section of the Black Vein at Risca Colliery.

The Black Vein here has not the same dip as at the Celynen Colliery. There it is 1 in 9; at Risca the seam is generally flat, but in some parts it dips 1 in 14.

The pits are sunk in a valley, but the workings run under mountains and thus have a cover of about 500 yards.

Two shifts of men are employed, one working from 6 A.M. and ceasing at 2 P.M., the other beginning at 2 P.M. and stopping at 10 P.M. The 8-hour period between 10 P.M. and 6 A.M. is used as a repairing shift.

Fig. 201 shows a section of the Black Vein at the Risca Colliery. The shale given in the section as 6 feet varies in thickness through the pit. Sometimes there is none and the rock forms the roof over the coal, at others the shale is 20 yards thick. Below the 6 feet of soft fireclay shown as forming the thill or floor in the section, is a harder bed of fireclay. Whichever bed is used for a floor the pressure causes it to rise in the road. The top coal of 3 ft. is left on for a roof.

Fig. 202 shows the method of working. The stall-roads are only 9 yards apart, and are cut off by cross roads at about every 50 yards. The roadsides are faced with stone brought down from the surface at a cost of 1*s.* per ton. The goaf spaces between the walls thus made are filled with small coal, and stones and rubbish, obtained from the repairing of roads. The collier puts in props where required, there being no rule to fix the distances between them.

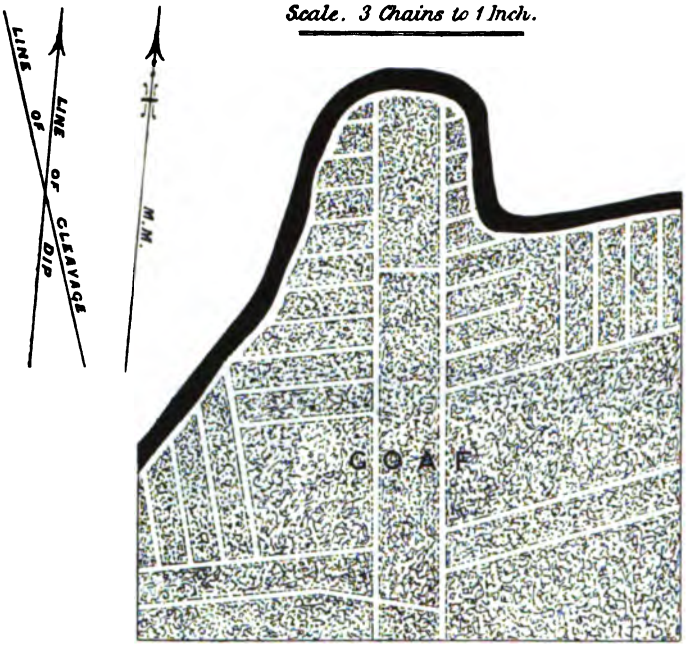


Fig. 202.—Risca Colliery, near Newport, Monmouthshire. Plan showing Longwall Method of Working the Black Vein.

None of the props along the face are recovered, because the seam is fiery. If the props were taken out, holes would be left in the roof in which the gas would collect. In the roads 10-inch props with lids are put in about a yard apart, but those for securing the faces are from 5 to 6 inches in diameter. Two men and a boy work together in each stall, and can send out from 9 to 10 tons of coal in their 8-hour shift. In 1881 they were paid 1*s.* 6*d.* a ton for large hand-filled coal; they take charge of the stall and the roadway for 40 yards back. The line of cleavage is shown on the plan, Fig. 202, the cleavage planes being very distinctly marked. This renders the coal somewhat easy to work, as the coal comes off in large pieces, the collier merely lifting these with his pick. No holing is required under the coal unless in very exceptional circumstances.

The top coal left on makes a good roof near the face, but at a distance of 150 yards back, the full subsidence has taken place, and it becomes necessary to take down this top coal in the roads to make height. This ripping of the top is followed by sets of timber, which are notched in the Welsh method, as they are fixed in the road. The pressure on the sides of the road is very great. Larch and French timber are used for the sets, 10 inches in diameter, and placed about every 3 feet—though this distance is subject to alteration to meet the requirements of the case. At the crossings of roads square timbers or frames are used. These consist of a usual set of timber being first fixed across both sides of the openings. Collars are then stretched between their collars and neatly notched into them, so as to form a frame. As at Celynen, a special class of men do the timbering, but here they work during the night from 10 to 6, the wages being in 1881, 4*s*. 6*d*. and 5*s*. a shift. Very few chocks are used, an occasional one being put in by the colliers at the corners of buildings, and where this is done, they are paid 1*s*. for each chock.

The timber is drawn out of the roads when they are abandoned, the roads being stowed up tight as the timber is drawn out in going backwards. The material for stowing these roads is got from the falls which are frequently taking place in the different roads.

The side pressure is so great that ordinary brick arching will not stand it. A weak point in the main road is secured by having strong stone walls built against the sides, and having strong oak beams stretched across between them. The cost for timber is 5*d*. per ton on the output, the price paid for pit timber at the colliery being 21*s*. per ton. The roads are dry and dusty, and are watered. The firemen look after the ventilation and visit each place twice during the 8-hour shift.

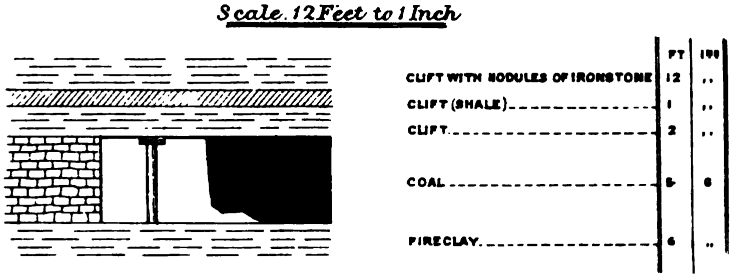


FIG. 203.—CWMPARK PIT, OCEAN COLLIERIES, TREORKY, SOUTH WALES. SECTION OF FOUR FEET STEAM COAL.

The method of working the steam coals in the Rhondda Valley, South Wales, is by Longwall. The Ocean Collieries comprise five coal-drawing and five upcast shafts, all working the smokeless steam coal.

The Cwmpark Pits, situated at Treorky, are 200 yards deep to the 4 feet steam coal, one being a downcast, the other an upcast. The upcast is used only as a ventilating shaft, and has a furnace at the bottom of it. The downcast is the winding shaft. The cages are single-decked and carry one tram each. The tram holds 2 tons of coal, and is made of iron, the curved sides being of sheet-iron, and having two cross-bars at each end. The body of the tram is placed within the wheels, which run loose on the axles, and the axles loose in the guides or carriages under the tram. The gauge of way is 3 feet. The coal is large, and is piled up 3 feet above the sides of the trams. In the underground engine plane the rails are flat-bottomed, and weigh 40 lbs. to the yard; those in the roads to the faces, 22 lbs.

Fig. 203 shows a section of the seam worked, which lies very flat. The 2 feet of clift next the coal is ripped down in the roads at the faces, and built into the waste. The collier does this ripping, and keeps the face of it within a foot or two of the face of the coal.

In the cross roads the second bed of clift above the coal is taken down, and also in the stall roads, when they get too low, and this is also built into the waste. The material from the ripping together with the small coal are sufficient to nearly fill the waste.

Fig. 204 shows the method of working adopted. The stall roads are 12 yards apart, and these are cut off by cross roads or headings every 50 yards.

Props and lids are used at the face, the lids being made of split props. A row is placed 3 feet back from the face and parallel to it, the props being set 6 feet apart.

Three-foot chocks are placed at the corners of buildings in both the stalls and the headings. The stall roads are secured by props and lids—6-inch props being used. Frequently they require no timbering. No rule is in force as to the distance between props; the collier puts these where he considers they are required. The roof is good, being subject only to occasional slips.

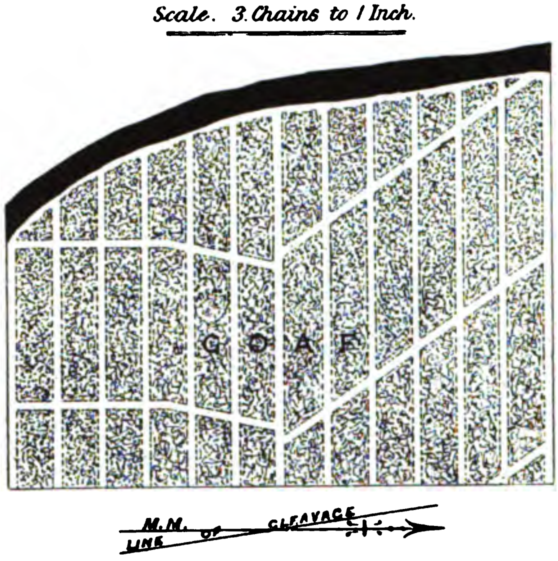


Fig. 204.—Ocean Colliery, Treorky, South Wales. Plan showing Longwall Method of Working the Steam Coal Seams.

Chocks or cogs are built along each side of the headings, at the corners of the buildings forming the stall roads, and also between them; the chocks on one side of the road are placed not opposite those on the other, but so as to break band with them. The chocks are 3 feet long, and old round timber is used for them. They are filled inside with rubbish.

In the headings, all the timber, props, and chocks are put in by the collier, who is paid according to a recognized scale for doing so. In 1881, the tonnage price of screened coal paid the collier for hewing, filling, and setting props was 1*s.* 7*d.*

Two men worked in each stall, getting about 6 tons of coal a day and putting it in large pieces in the tram at the stall road end. For any heading chocks put in by the collier he received 1*s.* 11*d.* each; for drawing any timber 1*d.* per prop; for ripping 1*d.* per yard per inch in height, the width being not less than 5 feet in the top of ripping, the road being 10 feet wide at the bottom. For the first ripping of 2 feet thick he received 2*s.* per yard for the necessary width. In the main roads sets of timber are used, but are not placed at any stated distance apart.

In the headings these sets are put at intervals of 4 feet. The timber used in the sets is 11 inches in diameter and notched in the Welsh fashion. The "set" timbering is done along with the second ripping by the repairers, who work at night. The price paid these men is 2*s.* for each set of 9-foot timber, and they are paid extra for ripping the headings. One fireman is placed in each district of about 40 or 50 places. His duty is to attend to the ventilation in his district, to put up wooden or canvas doors where required, or to place air tubes for ventilation. Timber at the colliery costs about 24*s.* per ton, or about 11*d.* a cubic foot. As an example of the amount of subsidence of the roof, it may be mentioned that a road, driven through stowing which had been standing for a year and a half and 150 yards back from the face, was found to be only 3 feet 4 inches high, so that the subsidence from the face to that point 150 yards distant was from 5½ feet to 3 feet 4 inches, nearly 40 per cent. of the original height.

In North Wales a system of working called the Wicket system is applied to seams of from 6 to 10 feet thick where the inclination is slight and the quantity of firedamp yielded is not great. Fig. 205 shows this method.

A pair of drivings proceed to the full rise and out of them are turned at right angles pairs of levels. A distance of about 130 yards separates one pair of levels from another and they are carried forward about 400 yards. The roads are driven 9 feet wide, and the two forming a pair are separated by 13 yards of coal. They are connected at intervals of 27 yards by cross holings. The coal formed between any two pairs of levels is worked out by juds or sections of the whole block proceeding to the rise from the lower pair of levels. These juds are carried much like the South Wales double stall roads, a road being formed on each side of the jud and the space between packed as closely as the available material will allow. Where no suitable material is yielded in working the seam and only the small coal is available for filling the waste, the ventilating current cannot properly be kept round the face owing to the imperfectly packed waste. The 7-yard rib of coal between the juds is not worked, so that much loss arises from working in this way.

In 1857, the Bank System of working prevailed in South Yorkshire, where the Barnsley Seam yields various qualities of coal from its several beds.* Very little of any other seam was at that time worked in South Yorkshire. The seam varies in thickness. Near its outcrop and 5 miles from the town of Barnsley the seam is divided into many thin beds, the divisions being formed by fireclay partings of several inches thick. Proceeding southwards, many of these fireclay partings disappear, and at one mile to the north of Barnsley, the seam presents the following section:

	ft.	in.	ft.	in.
Coal, called "Day Bed"			1	0
Parting	0	2		
Coal, called "Middle Bed"			1	1
Coal, called "Low Bed"			1	3
Parting	0	8		
Coal and Pyrites, called "Clay seam"			0	7
Coal, called "Hards"			2	8
Coal, do. "Slottings"			2	2
Total thickness of Coal			8	9

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* See Lectures delivered at the Bristol Mining School, 1857.

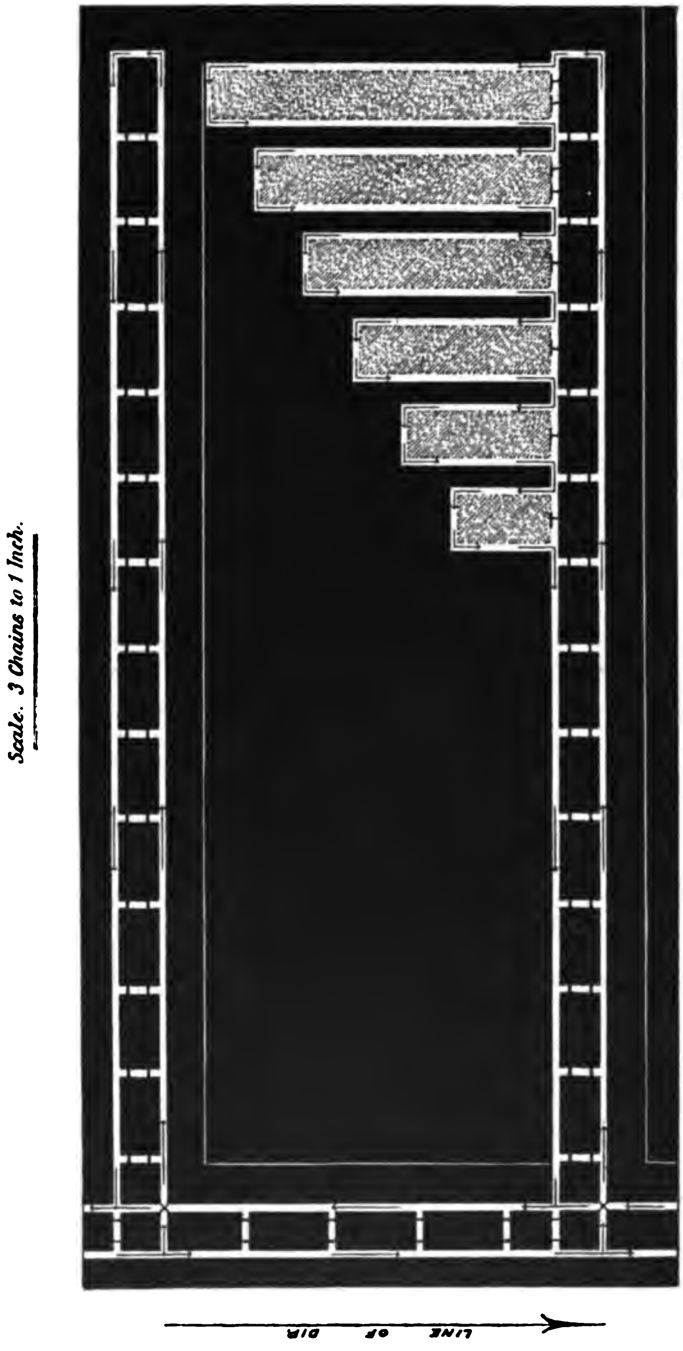


Fig. 205.—Plan showing the Wicket System of Working Coal in North Wales.

This is a highly prized section of the seam, and as in this state it lay within 150 yards of the surface, it was extensively worked in earlier years. Proceeding southwards to a mile south of Barnsley, the top coal, called the "Day Bed," is absent from the seam, and its other portions are so much thinner that the total thickness of coal in the whole seam is only 6 feet 6 inches. In the South Yorkshire Collieries, there is little or no parting in the seam, the average section of which is—

	ft.	in.
Soft Coal (Middle Bed)	1	5
Soft Coal (Low Bed)	1	6
Coal and Pyrites (Clay seam)	0	6
Hard Coal (Hards)	2	5
Soft Coal (Slottings)	2	4
Total thickness .	8	2

The "Day" and "Low" beds at the top and the "Slottings" at the bottom are filled and sold together as gas and household coal; the "Hards" (being the best part of the seam) is to some extent made into coke, but is mostly used as a steam coal, for which it is well adapted. The "Clay" seam is inferior, owing to the iron pyrites mixed with the coal, and is sold for lime and brick burning, or stowed in the waste.

The Barnsley seam contains much inflammable gas. When the working places are driven across the cleavage, the gas is emitted freely from the planes of cleavage, but where places are driven in line with the cleavage no gas is given off beyond that which comes from the coal actually obtained in such working places. In both cases the gas comes from the coal, only in the first case it is drained from the solid on either side of the working place. This remarkable peculiarity arises from the compact nature of the coal and the regularity of the cleavage planes which run without communication with each other.

In designing the Bank System of working, in which pairs of bordgates or winning places are driven to the full rise of the seam and across the cleavage, in advance of the main workings, it was thought that the gas would be drained or drawn off from the blocks of coal extending from one pair of bordgates to another, and experience has proved it to be so.

The Barnsley Seam is very free from water. Under it is a bed of fireclay from 3 feet 6 inches to 5 feet thick, and this fireclay contains large quantities of inflammable gas in bags or cavities. When the workings have reached the edge of these cavities, the pressure of the gas has been observed first to heave the floor up at different points along the face for many yards, this lifting of the floor being succeeded by large fractures in it (the line of fracture being more or less clear and parallel to the face of the place) through which the gas has issued with great force, accompanied by a sound similar to that made by high-pressure steam escaping from boilers.

A sudden yield of inflammable gas in this way is called an *outburst*, and has probably caused many of the unfortunate explosions which have occurred in South Yorkshire.

The Lundhill Colliery, near Barnsley, has belonged to the same Company since 1854, and in 1857, when the explosion occurred there, the Barnsley seam was worked on the Bank System, as shown on the plan, Fig. 206.

The lowest is the main or horse level, being driven on both sides from the bottom of the downcast, which is the winding shaft. The level to the rise is called the middle level, and the upper the bank level. All three levels are driven abreast of each other, and are separated by 20 or 25 yards of coal. Openings connect these levels at intervals of 30 to 40 yards. At a suitable distance from the downcast shaft a bordgate, 6 feet wide, is driven towards the full rise, to the bank level. Here another bordgate is started as a companion to the first, being separated by 8 yards of coal. This pair of bordgates advance abreast of each other, and are connected by openings through the coal at every 20 yards. When the three levels have advanced 110 yards, another pair of bordgates is driven out of them similar to the first pair. In the centre of the block of coal left between the two pairs of bordgates a leading bank, 18 yards wide, is started from the bank level and is worked towards the rise. In the meantime the bordgates have advanced sufficiently, viz. 40 yards from bank level, to allow of slits being driven level course out of the bordgates towards the approaching leading bank. When the slits are holed into the bank (one on each side) they become the roads through which the coals are conveyed from the face. "Following-up-banks" are now commenced at the bank level, and consist of a face of 6 yards from the sides of the leading bank. These "following-up-banks," one on either side of the leading bank, are carried up to the slit above, and, as the leading bank has been progressing, it has now holed into the second pair of slits. Afterwards the "following-up-banks" are continued forward and second ones started from the bank level, taking another portion of coal 6 yards wide from that left. This operation is repeated as each leading bank reaches the slits, until the coal between the goaf and bordgates has been reduced as much as may be deemed safe or advisable.

In each of the 110-yard blocks along the level, the bordgates and banks are repeated in the same order as before described, and are carried forward to any required distance. In order to maintain a passage for the air and travelling road down each side of the goaf formed by the leading bank, packs 5 feet wide and 6 feet from the coal rib are built with the blue metal which is allowed to fall behind the face of the bank. Another pack of similar size is carried up the centre of the leading bank, and these three packs are built and carried on as the face advances. Each following-up-bank builds one pack as it advances, 6 feet from the ribside, so as to maintain a roadway. All the packs in course of time are overthrown by falls in the goaf, except those at the side, which must be kept to maintain the air-way. The pillars left in driving the bordgates, that is, those formed in each pair, and any coal it may have been deemed prudent to leave next the bordgates, are worked at the last when the banks have been worked out. This pillar coal is first worked at the highest and inside point and worked backwards.

In working the coal the holing was done in the "slottings," the hard coal above it being then wedged down. The clay seam was carefully separated from the rest, and the low bed of soft coal taken down. The timbering was placed under the top bed of softs; as the face advanced, the subsidence of the roof caused the top soft coal to fall at the goaf side of the props and was filled into the trams.

The air was taken from the downcast to the extremity of the level, doors being placed in all the working bordgates. From the face of the levels it was carried up the inside pair of bordgates, around the face of the new leading bank, up the next bordgate to the face, returning to the slit leading into the next bank. Here the air divided as it entered the bank, a small portion descending the air-way maintained by the pack-wall to the following-up-bank on that side and returned by the middle level to the other side of the bank, ascended the air-way corresponding to the one it descended on the other side, and joined the current, which had traversed the leading bank face, at the highest holed slit. The united volume passed along the slit to the next pair of bordgates, where it ascended the one, returning along the other to the highest slit, dividing, as before, at the first bankgate. Here a division took place, one portion descending to air the following-up-banks on that side and passing into the middle level. The main current, after passing the leading bank and descending to the highest slit, again divided, giving

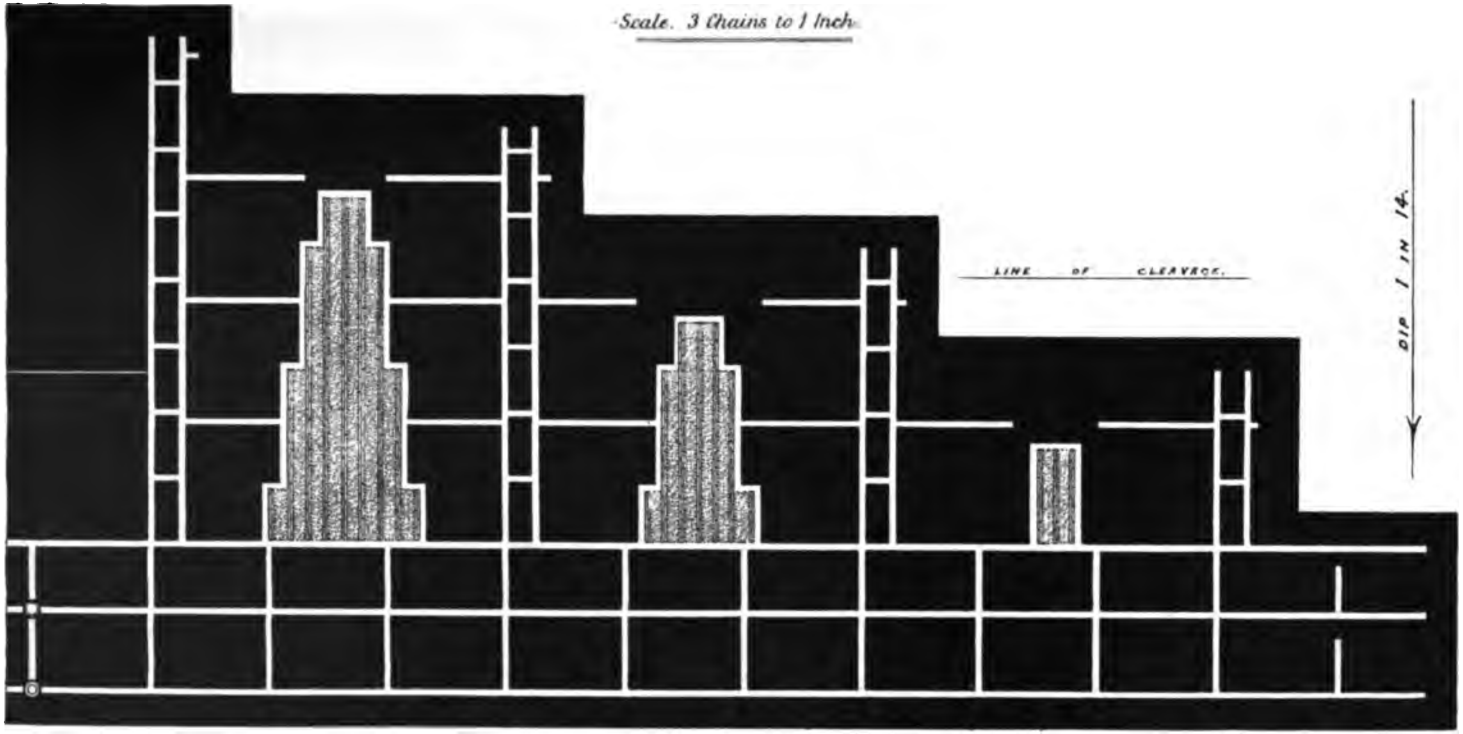


Fig. 206.—Plan showing the Bank System of Working the Barnsley Seam in Operation at the Lundhill Colliery, Yorkshire, in 1857.

off a split to ventilate the following-up-banks below. This current joined that which had aired the following-up-banks on the other side in the middle level, and the two currents passed on up the next bordgate to join the main current at the highest holed slit; these re-combined currents passed into the first pair of bordgates from the shaft, by one of which it descended to the upcast shaft.

It will thus be seen that whatever advantage was obtained from draining the gas into the bordgates was quite neutralised by these ventilating arrangements. The air, after traversing the bordgates, would become charged with the gas emitted there, after which it was passed to the workmen at the face of the banks.

The Lundhill pits are 210 yards deep to the Barnsley Seam, and prove the following seams:—

Melton Field .	4 feet thick at	40 yards from the surface.
Abdy	3 „ „ „	70 „ „ „
Kents Thin . .	2 „ „ „	106 „ „ „
Kents Thick .	3 „ „ „	144 „ „ „
Barnsley Seam .	8 „ „ „	210 „ „ „

Under the town of Barnsley the next coal seam of importance below the Barnsley Bed is the Swallow-Wood Coal, 60 yards below; the next is the Flockton, 140 yards lower. The Parkgate is 80 yards below the Flockton, after which comes the Thorncliffe Thin, 35 yards still lower. Next in descending order is the celebrated Silkstone Seam, 52 yards below the Thorncliffe Thin, but none of these seams approach the thickness of the Barnsley Bed.

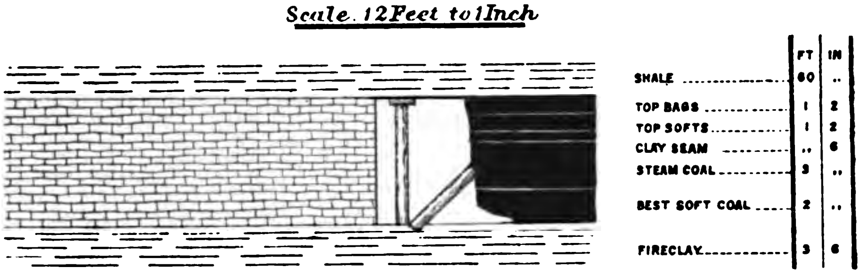


Fig. 207.—Lundhill Colliery, near Barnsley. Mode of Spragging and Propping in the Pillar Workings of the Barnsley Seam.

The Bank System of working has now given place to the Bord and Pillar. In 1881 the operations at Lundhill were confined to three shafts, one being used solely as a ventilation pit. All are circular shafts, the two downcasts being 11 feet 6 inches and 12 feet in diameter respectively, and the upcast 14 feet 6 inches. A pair of horizontal high-pressure winding engines, 25-inch diameter cylinders, 5-foot stroke, and 14 feet 6 inch drum, winds coal at one of the downcasts, and a similar engine at the other. Together they raise 750 tons in one shift of 9½ hours. A furnace, having a grate surface of 144 square feet, placed at the bottom of the upcast shaft, produces the ventilation. The quantity of air passing up the upcast is 300,000 cubic feet per minute. This air is not passed over the furnace, but enters the upcast shaft by means of a dumb drift 35 yards from the bottom. The total volume of 300,000 cubic feet includes 50,000 cubic feet of air used to ventilate the stables and feed the furnaces.

Fig. 207 shows a section of the Barnsley seam taken in 1881, from which it is seen that the coal has a total thickness of 7 feet 10 inches.

The method of working now adopted is shown on the plan, Fig. 208, and is an arrangement of Bord and Pillar. The pillars are 40 yards square, and the roads

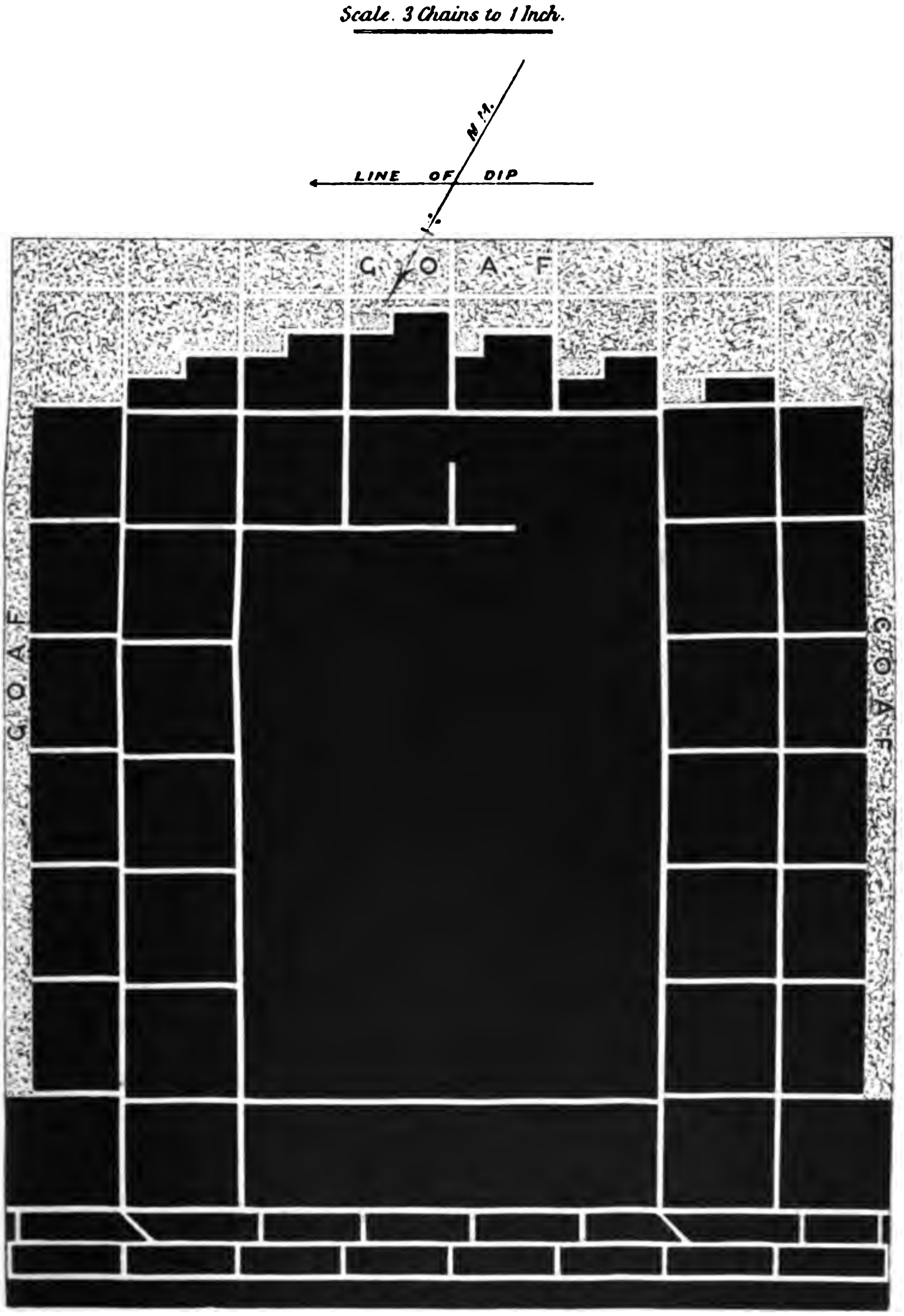


Fig. 208.—Lundhill Colliery, near Barnsley. Plan showing Bord and Pillar System in Operation for Working the Barnsley Seam.

about them, both "bords" and "endings," are of a uniform width of 6 feet. Pairs of levels are driven from the main-dip roads. A block of coal 150 yards wide separates the different pairs of levels. After these levels have reached a given distance, this 150-yard block is broken up into the 40-yard square pillars, and worked back. There is this peculiarity in it, however, that only one line of pillars is formed in advance of the line of broken. The "endings" are not quite opposite, but one is 3 feet inside the other, to enable the rails to be turned into the working face on either side. The workings are dry and the roads slightly dusty, and are therefore regularly watered. No timbering is required usually in the roads, as the top coal is left on and forms a good roof. Below the fireclay shown on the section are 7 yards of sandstone. The "Top Bags" are left on to support the roof, but are afterwards taken down in the pillar working.

Except for carrying the brattice, no props are used as a rule in the whole workings. The brattice is made of wood, in lengths of 10 feet, and 4 feet broad, overlapping in the centre and nailed to slight posts. The practice is to divide the road into an intake on the dip side, 4 feet wide, and a return 2 feet wide on the rise side. Two men work together in a place, advancing in the solid—a holer and a filler.

Scale. 12 Feet to 1 Inch

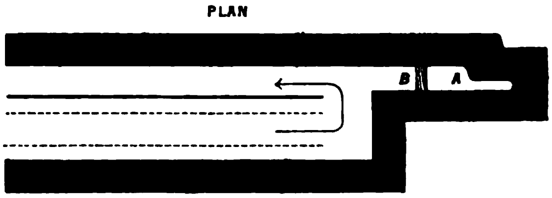


Fig. 209.

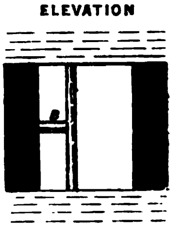


Fig. 210.

LUNDHILL COLLIERY. WORKING IN THE SOLID PLACES OF THE BARNSLEY SEAM.

The holer cuts a shearing, A (see sketches, Figs. 209 and 210), 6 inches wide, about 18 inches from the rise side of the place. When this shearing is in 3 feet or other suitable distance, he wedges off the 18 inches of coal, B (see sketches), on the rise side, and is thus enabled to extend his shearing further in, whilst the filler holes and works off the coal in the 4 feet width on the low side of the road. The holer places a sprag where shown on the sketches to protect both himself while shearing and the filler whilst holing. The shearing is kept 9 or 10 feet in advance, that portion of it which is 2 feet wide being 3 feet behind the face of the shearing.

The men driving one of these solid places received 5*s*. per yard, and 11*s*. 11*d*. for 10 tons of coal sent out.

Fig. 211 shows the method of working the broken, and is an enlarged plan of one of the pillars shown in Fig. 208. The juds are 8 yards wide, and are usually carried up to the rise the whole length of the pillar. Occasionally, if more coal is wanted, it is obtained by men commencing on the other side of the pillar, and working downhill to meet the jud coming up.

Fig. 211 shows one jud of a pillar as being worked off, and another in the act of being driven. The roadway is formed along the coal to one side of the lift, and is 6 feet wide. On the waste side of the road a pack-wall, 6 feet wide at the bottom and 5 feet at the top, is built with the 6 inches of clay-seam and débris from the waste. The face is protected by two rows of posts besides the pack-wall, the rows being 3 feet apart, the rear row breaking band with the front row. The props are set 6 feet apart in the rows and are 6 inches in diameter, the lids over them being of hard wood 8 inches square, by 2 inches thick. The pack-wall is added to every 3 feet, when the rear row of props is drawn and advanced to the front, where they are again set. After the withdrawal of the rear row of props, the top coal falls, and as much of it as can be removed is taken. Two colliers work in a lift. They hew and fill the coal, keeping separate the various qualities, and fix the props, receiving 11*s.* 11*d.* per score of 10 tons for so doing.

They hole 3 feet in under the coal, a foot high in front and put sprags in every 6 feet. (See Fig. 207.)

On finishing the holing and knocking out the sprags, the coal comes down. No powder is used in the solid or broken mine. Hand riddles having a ¾-inch mesh are used, the coals passing through the riddles together with the small is thrown into the waste. The colliers at the face do not build the packs; a special class of day-men called packers do this, and also draw the props. These men are paid 4*s.* 4*d.* per day. In drawing props a "dog" (see Fig. 163) is used. This "dog" and the method of using it have been described in the chapter devoted to Timbering and Walling. When the props have been drawn the roof falls quickly, but the pack-wall maintains the roadway without any kind of timber being placed under the roof of top coal. Occasionally the men at the face work the coal and build the pack-wall, and where this is done, the men receive 3½*d.* a ton beyond the usual score price. The top coal everywhere makes an excellent roof, both in the roads and faces, and the shale over the coal is not exposed anywhere except in the waste.

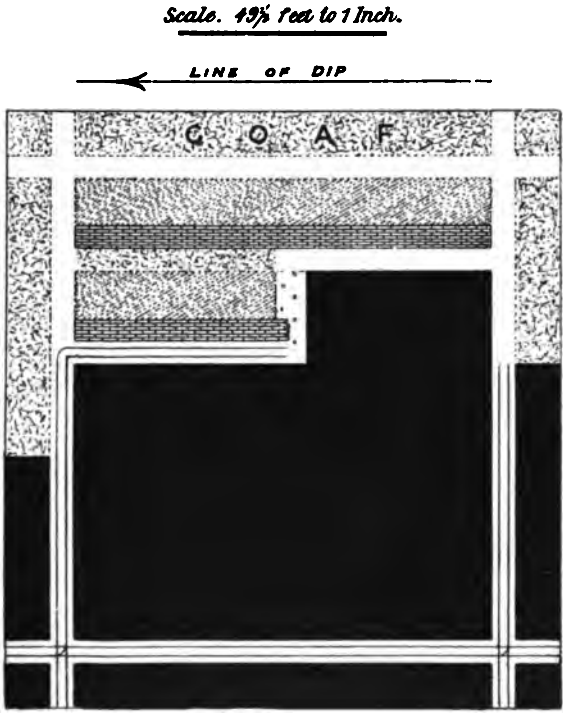


Fig. 211.—Lundhill Colliery, near Barnsley. Mode of Working off Pillars in the Barnsley Seam.

The advantage of this system over the Bank system previously adopted is manifest. The broken does not follow closely behind the exploring places, but the latter are carried to their destination, leaving large blocks between them, and when these blocks are worked, it is from the inside backwards, leaving the goaf behind.

It is true, that no amount of ventilation can dilute and render harmless such volumes of gas as are given off at the outbursts above mentioned, and it is probable they will occur, in every mode of working: still, the method adopted should give the greatest possible margin of safety. The practice of getting the coal without the use of powder at Lundhill increases the margin of safety.

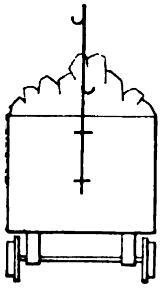


Fig. 212.—Kiveton Park Colliery. Double Fork for Endless Rope Haulage.

At the Kiveton Park Colliery, near Sheffield, the Barnsley Seam is worked by the Longwall system. The colliery has been working coal since about 1867, the furthest workings being about 1½ miles from the shaft.

The colliery consists of two shafts sunk 400 yards to the Barnsley seam. One of the shafts is a downcast and the other an upcast, each being 13 feet in diameter.

Only the downcast is used for winding, and for that purpose it is fitted with wire-rope guides and double-decked cages.

About 1,100 tons are landed in one shift of 10 hours, by means of a pair of horizontal high-pressure engines, 36-inch cylinders, with 6-foot stroke. A furnace produces the ventilation in the other shaft, which is fitted with rope guides, so that in cases of emergency, the workmen may be raised in it, but no coals are wound. An endless rope for driving the underground haulage, is taken down this shaft and passed round one of Fowler's clip pulleys at the shaft bottom. This pulley is on an upright shaft, as also are two other pulleys of the same description worked by clutches, driving endless ropes to different districts.

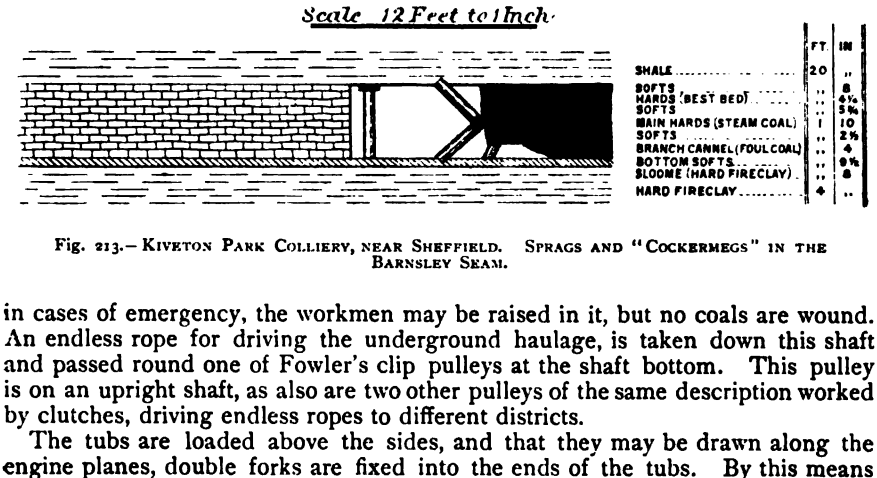


Fig. 213.—Kiveton Park Colliery, near Sheffield. Sprags and "Cockermegs" in the Barnsley Seam.

The tubs are loaded above the sides, and that they may be drawn along the engine planes, double forks are fixed into the ends of the tubs. By this means when the tub is loaded high, the rope rests in the higher catch of the fork, when empty, in the lower catch. See Fig. 212.

Fig. 213 shows a section of the seam, which is here 5 feet 4 inches thick, the coal being a little harder than at Lundhill, and used chiefly as a steam coal. Above the coal is 20 feet of strong bind, 2 feet of which is ripped down in the roads. Above this is a bed of black rock 6 feet thick. The thill under the coal consists of hard "clunch" or fireclay, and underneath this is "bind" (shale).

The seam dips to the east, the inclination being 1 in 22. Fig. 214 shows the method of working, which is by Longwall with the roads advancing to the rise.

One of the roads, most suitable for the purpose, is made a "jinney," or self-acting incline, and levels from it cut off the roads going to the rise every 200 yards. The distance between the stall roads is 60 yards. At each road-head iron plates are placed, and rails for the tubs laid along the face. The tubs are "spragged" in being taken down the road leading to the incline. In a 60-yard stall two stall-men work. They are small contractors, and receive a tonnage price on all coal sent from the stall and delivered at the top of the "jinney."

They employ other men in the stall. The contractors do the packing, the timbering, and the ripping at the stall roads.

The system of building pack-walls next the roads and in the waste is similar to that at High Park, shown in Fig. 217, and described later on.

The ripping in the roads yields material for building the packs at the sides of the road, which is 10 feet wide. These packs are made 6 feet wide, and others parallel to them are built in the waste of the material yielded in the holing and fallen stones. The face is protected by two rows of 6-inch timber, carried along the face behind the rails. The rows are 6 feet apart and a similar distance separates any two props in the rows. As the pack-walls are built, the rear row of props is taken down, and re-set in advance of the other row. As the props are drawn, the roof in the waste falls. Although the stall-men have entire charge of the place, they must build the pack-walls and place the props as directed by the deputy. Commencing at the road-head, these men hole along both sides to the extremities of their stall. They hole 5 feet under the coal, the height of the holing in front being 18 inches. Sprags are put in under the coal 6 feet apart as the holing proceeds. See Fig. 213. Where the coal is tender, besides the sprags, "cockermegs" are put in. These consist of a sloping prop reaching from the floor, and another reaching from the roof which hold a third prop placed horizontally along the face. See Fig. 213. When the holing all along the coal is completed, and held in position by the sprags, a cut or shearing is made in it at the road-head, after which one or two sprags are taken out, thus allowing a portion of the coal to fall. As this coal is removed in the tubs, other sprags are taken out, and more coal taken down. This process is continued along the stall on both sides of the road until the extremities are reached, the rails being laid forward as the coal is taken down in front. After a holer has worked to the end of his "bank," he returns to the road-head, and there begins another holing at the point from which the coal has just been removed.

The stall-men have two fillers, and one holer, so that 5 men work in a 60-yard stall, from which they send out daily from 20 to 25 tons of coal. No timber is used in the roads, the roof being excellent. The timber for the face costs 2*d.* to 2½*d.* a ton.

At the High Park Colliery, Langley Mills, Notts, the Barnsley seam is called the Top Hard Coal and is worked by Longwall. The colliery has been working since about 1861, the Top Hard seam being won by two shafts at a depth of 200 yards. These are both downcast and winding shafts, one engine hauling coals from both. Each shaft is 10 feet in diameter, and carries only one cage. Two trams, each holding 11 cwt. are placed in each cage. About 900 tons a day are landed, all the coal being picked and lifted from the trams into the waggons by hand, and no screens used. The upcast shaft is 12 feet in diameter and 116 yards deep, it being situated 1,100 yards away (on the rise side of the measures) from the downcast shafts. Near it, is another upcast shaft for a separate colliery. Each upcast has a Waddle fan 30 feet in diameter, so placed that, should one ventilator break down, the other can be used to draw the air from both shafts.

The fans run 70 revolutions per minute.

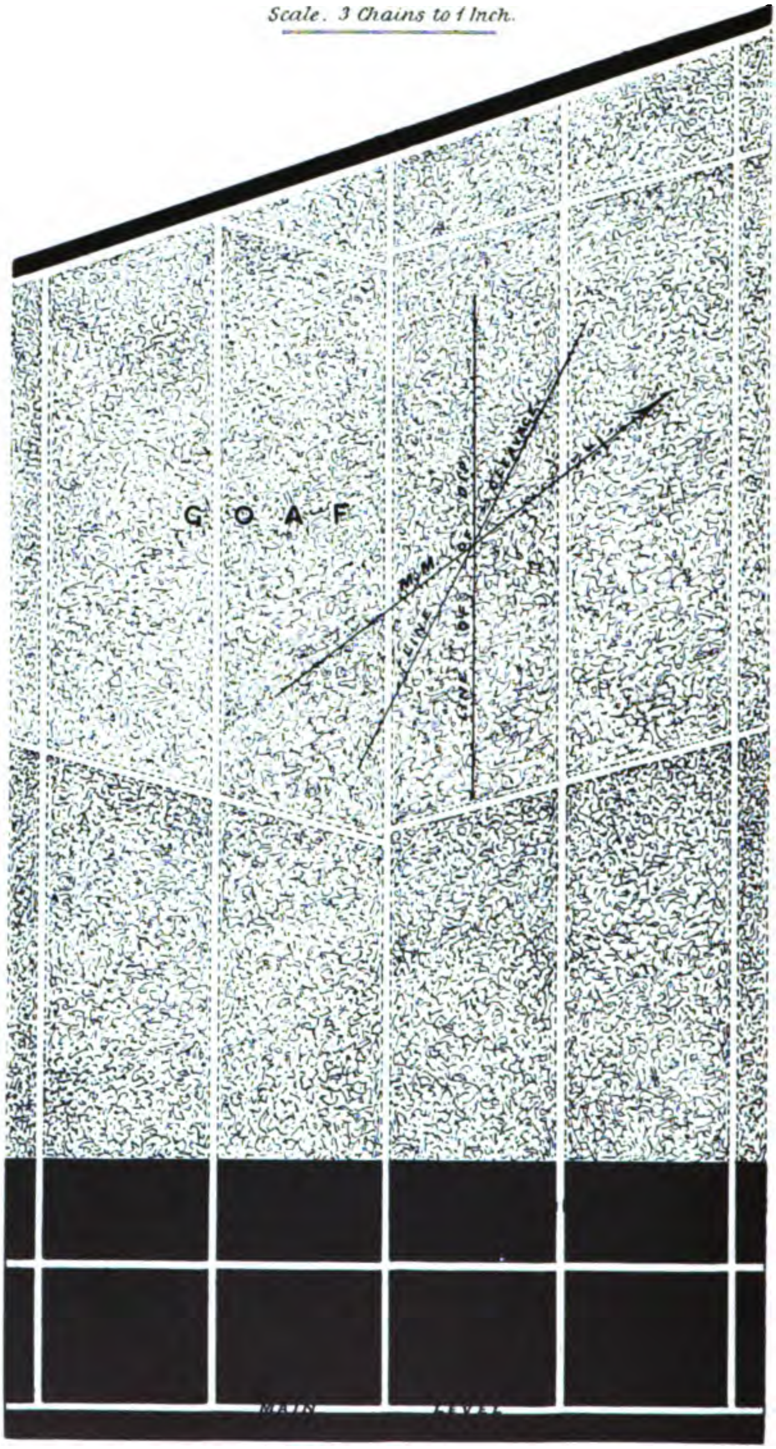


Fig. 214.—Kiveton Park Colliery, near Sheffield. Plan showing Longwall Method of Working the Barnsley Seam.

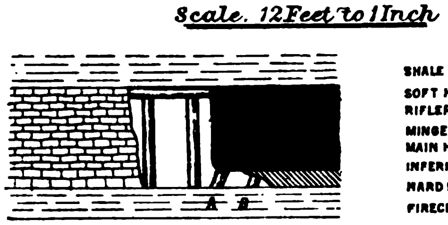
Fig. 215 shows a section of the Top Hard Coal Seam, and the mode of propping and spragging at the face.

The seam is 5 feet 2 inches thick, the dip being 1 in 22.

Over the seam are 15 feet of shale, 3 feet of which next the coal are taken down in the roads. Above the shale is a bed of coal 2 feet 6 inches thick, and over this is "bind" or shale. The thill is composed of fireclay 2 feet thick, and below it is a dark sandstone.

Fig. 216 is a plan showing the method of working. In 1881 there were 48 stalls all in line for the daily output of 900 tons. The distance from centre to centre of the gate-roads is 50 yards. These are carried to the rise, and are cut off every 400 yards by a cross-cut from one of the rise roads, which is made into a "jinney" or self-acting incline. All the coals are sent down the "jinney."

Three men have charge of each stall, and they at the date named had a contract price of 1*s.* 9*d.* per ton for hand-filled coal, all the small being gobbed. The contractors employ loaders and holers, themselves doing other kind of work necessary in the place, including the first ripping of 3 feet in the gate-road, building the packs and drawing the coal out to the first pass-bye, which never exceeds a distance of 50 yards from the face. From this pass-bye the coals are drawn by a pony to the top of the incline, and after being let down the incline reach the shaft by a rope haulage.



	FT.	IN.
SHALE	15	"
SOFT HOUSE COAL	1	..
RIFLER	..	10
MINGE	..	4
MAIN HARD COAL	1	7
INFERIOR COAL	1	5
HARD FIRECLAY	1	..
FIRECLAY	2	..

Fig. 215.—High Park Colliery, Langley Mills, Notts. Mode of Propping and Spragging Coal in the Top Hard Coal Seam.

Fig. 217 shows the practice of building the packs, and timbering at the face. When a holing is begun at the road-head, the packs are within 4 feet of the face, and there is a row of 8-inch props along the face, there being spaces of 4 feet 6 inches between the props. Norway timber props are used, 5 feet 3 inches long, the lids over them being 12 ins. × 4 × 6, made from broken posts. The stall-men set these props. The place being as described, the holers do their work from the road-head along both sides to the extremities of the stall. As they proceed, sprags (A, Fig. 215) 2 feet long are placed every 6 feet. Where required short sprags (B, Fig. 215), are put in under the coal. They are about 15 inches long, and are fixed at irregular distances apart. The holing is carried 4 feet in, the stall-men paying the holer 1*s.* 6*d.* for a fathom in length of such holing. The holing rates are 1*s.* a fathom for 3 feet in, 1*s.* 6*d.* a fathom 4 feet in, and 2*s.* a fathom for 5 feet. The practice here, however, is for the holer to hole 4 feet in, and after this has stood a short time to allow the fireclay to become crushed, the stall-man himself holes the other foot in. When the holing is completed, and the coal kept securely up by the sprags, a cut or shearing, 2 feet wide at the outside and tapering to 3 inches about 5 feet in, is made at the road-head. The stall-man then takes out 3 or 4 sprags, which allows the coal they supported to fall. This is filled into the trams, the roadway being carried along the face, as shown in Fig. 217. Props and sets of timber are placed over the rails at intervals of 4 feet 6 inches. One end of the collar is let into the coal 3 or 4 inches, the other end being supported by a prop. The props vary from 6 to 8 inches in diameter,

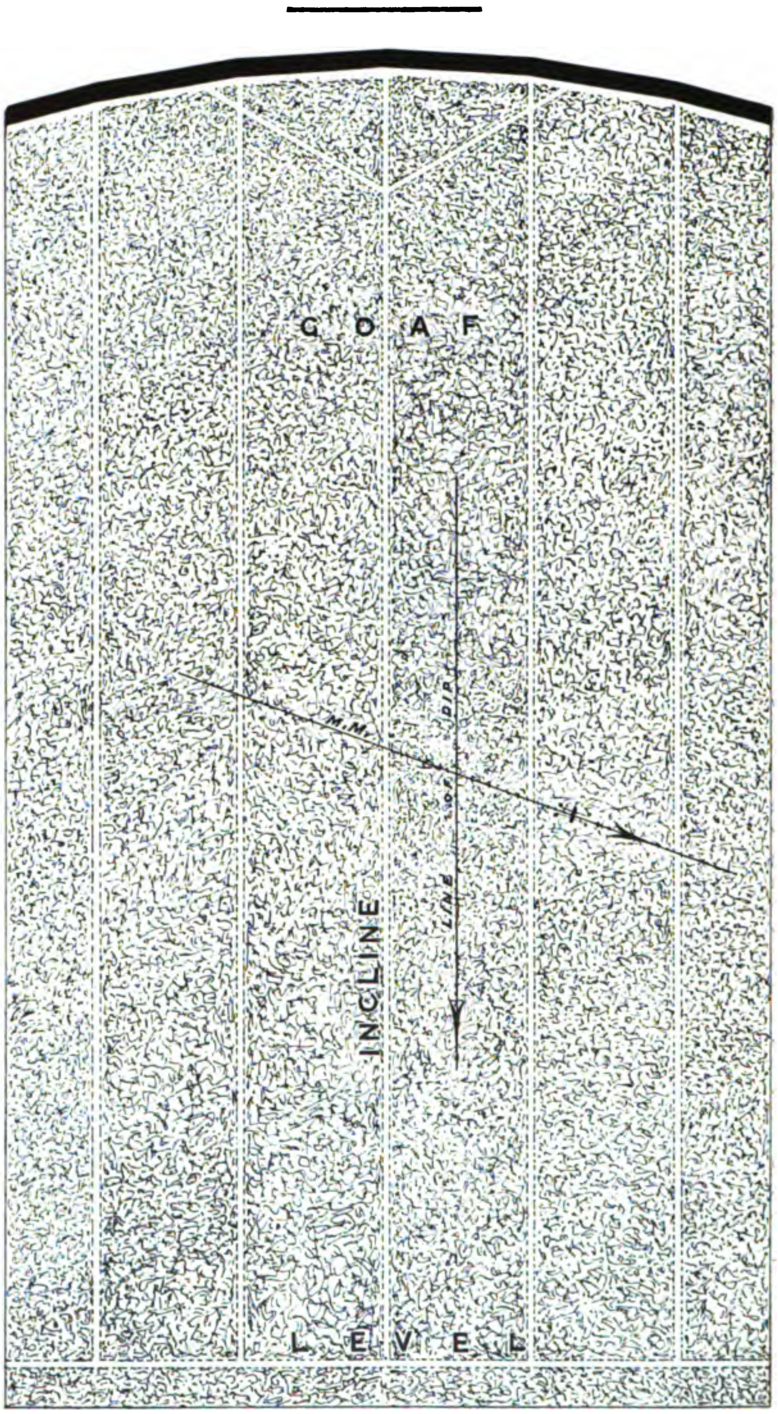


Fig. 216.—High Park Colliery, Langley Mills, Notts. Plan showing Longwall Workings in the Top Hard Coal Seam.

and are about 5 feet 3 inches long. The collars are 4 feet long and 5 inches by 4 inches in section. Having removed the coal first got, other sprags are taken out, allowing another portion of coal to fall, which in its turn is filled into the trams and removed, this process being continued until the end of the stall has been reached on both sides. Before commencing to add a fresh portion of building to the packs there are 2 rows of props about 5 feet apart behind the props, and sets of timber over the roadway. The stall-man draws these props as he builds the packs. The roof in places is tender and much broken, and where this is so the packs are built of the débris from the roof, about 6 feet wide, with intervals of 9 feet between them. The ripping of the road yields material for packing at the sides.

The duties of the deputies in the pit are much the same as that of the firemen in other districts. They superintend the stall-men, examine the places for firedamp, attend to the ventilation, &c.; but, subject to this superintendence, the stall-men set up and draw out all the timber, and rip the roads. Naked lights are used in the workings, but a safety lamp is fixed in the highest part of the ripping, so as to give warning in case any firedamp appears. Powder is used in ripping the roads.

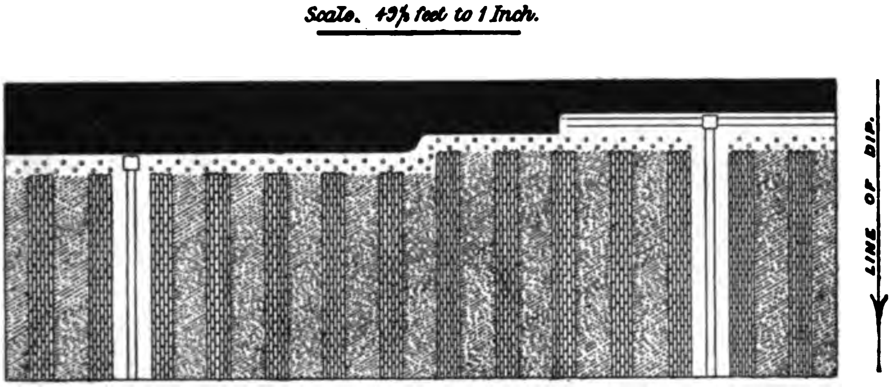


Fig. 217.—High Park Colliery, Langley Mills, Notts. Plan showing part of two Stalls with the packwalls, in the Top Hard Coal Seam.

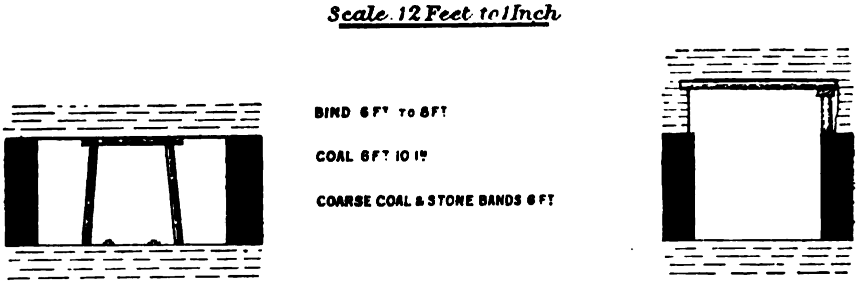
Very few props are used in the main roads, as the latter sink quickly, and in the main levels this necessitates ripping up to the 2 feet 6 inches of coal, which makes a good roof. The extreme subsidence takes place at a point 50 yards back from the face. The roads being dusty are watered, but not regularly.

The method of working the Maudlin Seam, Wearmouth Colliery, Sunderland, is by Pillar and Stall. The colliery has been working coal since 1835, but sinking operations were commenced there in 1826. There are two shafts, an upcast 11 feet 6 inches in diameter, and the downcast 12 feet. Both are winding shafts, and together in 1881 they raised 2,000 tons a day, the day here consisting of 24 hours. The tubs used carry 8 cwt. each. Two 4-decked cages run in each shaft. Two tubs are placed in each deck, so that the cage carries 8 tubs. The furnace which helps to produce the ventilation is not placed at the bottom of the upcast, but in the Maudlin Seam, 44 yards from the bottom. It has a firegrate area of 144 square feet, and is greatly assisted by the furnaces of 6 boilers. The total quantity of air varies from 180,000 cubic feet per minute to 200,000 cubic feet, being highest in the winter months, when the natural ventilation is greatest.

The shafts prove the Maudlin and Hutton seams at a depth of 530 yards and 574 yards respectively, both seams being worked. They lie almost flat, the dip to the east being so slight as to cause no inconvenience in carrying roads in any direction. The working faces in most of the districts are from 2 to 3½ miles distant from the shaft, and the coals are brought out along engine planes. The tail-rope system of haulage is adopted, and there are more than 20 miles of steel-wire rope in daily use in the pits. In one of the longest engine planes, where the empty train is taken in at the same time that the loaded one is brought out, there are 126 tubs on the road at one time.

A section of the Maudlin Seam is shown in Fig. 218.

The openings in the coal are 12 feet and 9 feet wide, the pillars left being 40 yards square. No holing under the coal requires to be made by the workmen, as the coal is tender, and the workmen "scallop" it, that is, they hack it out with picks while standing upright. In the screens used the bars are an inch apart, and 55 per cent. of the coal got passes through the screens. There is no objection to this, as the coal is used for gas making. The colliers do not place their own timber at the working faces. This is done by the deputies, unless in cases of emergency, when the collier puts up timber, which is left in his place for the purpose. In the solid working, sets of timber are placed in the roads every 3 feet. The props are 3½ inches in diameter, 6 feet high, 5 feet 6 inches apart at the bottom, and 4 feet 6 inches apart at the top, as shown in Fig. 218. The rails are laid in the centre of the roads between the two upright props. The collars are made of 5-inch props split through the middle, and the half-round side is placed next the roof.



Gears in working place.
Fig. 218.

Gears in main road.
Fig. 219.

METHODS OF TIMBERING IN THE MAUDLIN SEAM, WEARMOUTH COLLIERY, SUNDERLAND.

The props are put in by the deputies as the places are being driven, and remain in until another end is through. The deputies then draw them, after which the roof usually falls to a height of 2 or 3 feet; but sometimes the fall is sufficient to fill the place up. The pillars crumble off at the sides, owing to the tender nature of the coal.

The Broken is worked by splitting the pillars, and bringing back the jud on both sides. A deputy attends about 12 men, and they send out about 60 tons a day.

The collier receives 10*d.* a ton for hewing and filling, in the whole mine.

The roof over the coal is good, the cost for timbering being 1¼*d.* per ton in the roads and solid places, and 2*d.* in the pillar working.

The main roads are timbered, as they require it, with props, or sets of timber. At some parts the sets are placed 3 feet apart, and are 6 inches square and 9 feet long. The props in the main roads are 4½ inches in diameter, 8 feet long, and have hard wood lids 6 inches × 7 × 2½. Where the sides are good one side of the collar is notched 6 inches into the roof, and receives support on the other from a short prop 3 feet long, this prop being placed in a notch prepared for it in the roof over the coal, see Fig. 219. The timbering of the main roads is done by a special class of men called stone-men. They work in pairs, and receive about 1*s*. for a 9-foot "balk" or collar, but the price varies according to the difficulty of the task in fixing it. Two stone-men working together can put in 10 sets of 9-foot timber in a shift.

The workings are dry generally, although a little water is found in some of the swamps, and this water is very salt, containing as much as 21 ozs. of salt to the gallon. The roads get very dusty; they are watered regularly, and the dust cleared up once a fortnight.

At the Silksworth Colliery, near Sunderland, the same seams are worked as at Wearmouth Colliery. Silksworth Colliery is comparatively new, and consists of two shafts, one of which is 16 feet 6 inches in diameter, and the other 14 feet. Both shafts are sunk to the Hutton Seam, at a depth of 580 yards, the Maudlin being passed at a depth of 540 yards. 2,000 tons of coal are landed daily. Only the larger shaft is used for winding, and it is fitted up with 4 cages. Two of these wind coals from the Hutton, and two from the Maudlin Seams. There are two pairs of winding engines, each having 48-inch cylinders and 6-foot stroke. The engines winding from the Maudlin Seam have 25-foot cylindrical drums; the other pair of engines, winding from the Hutton Seam, have a scroll drum 13 to 28 feet in diameter. One pair of engines was made by Barclay of Kilmarnock, and the other by the Grange Iron Co., Durham. Both are fitted with a very simple and effective automatic variable expansion gear. Mr. Daglish, the chief viewer, in a paper read before the North of England Institute of Mining Engineers, has fully described this expansion gear. The cages are arranged as at Wearmouth Colliery to have 8 tubs on 4 decks, but are here loaded and unloaded at two levels, both at the surface and underground. The smaller shaft is used only as an upcast. A furnace is placed in the Maudlin Seam which has 90 square feet area of firegrate. The total quantity of air passing up the shaft is 195,000 cubic feet per minute. This reaches the shaft by three main returns, one 100 feet area, passing 45,000 cubic feet a minute; one 124 square feet, passing 80,000 cubic feet a minute; and that in the Hutton Seam, 100 feet area, passing 70,000 cubic feet. The dip is not uniform throughout the workings, varying from 1 in 18 to 1 in 6.

The following is a section of the Maudlin Seam:—

	ft.	in.
Blue Metal (Roof)	1	8
Coal	5	8
Sagger (Fireclay)	0	2
Bottom Coal	0	10
Sagger	0	8
Coal	1	6
Blue Metal	3	0

The method of working is by Pillar and Stall, the pillars being formed 50 yards square, the openings round them being 12 feet one way, and 8 feet the other. The 12-foot wide places are commenced 8 feet wide, and after going in 9 feet they are widened to their full width, and are carried on so till they are nearly through. When within 9 feet of being through they are reduced to a width of 8 feet, and carried to the end that width. The working is in the 5 feet 8 inches of coal in the above given section. The roof consists of a "blue metal" 20 inches thick, over which are 3 fathoms of post and blue metal mixed. Below the 5 feet

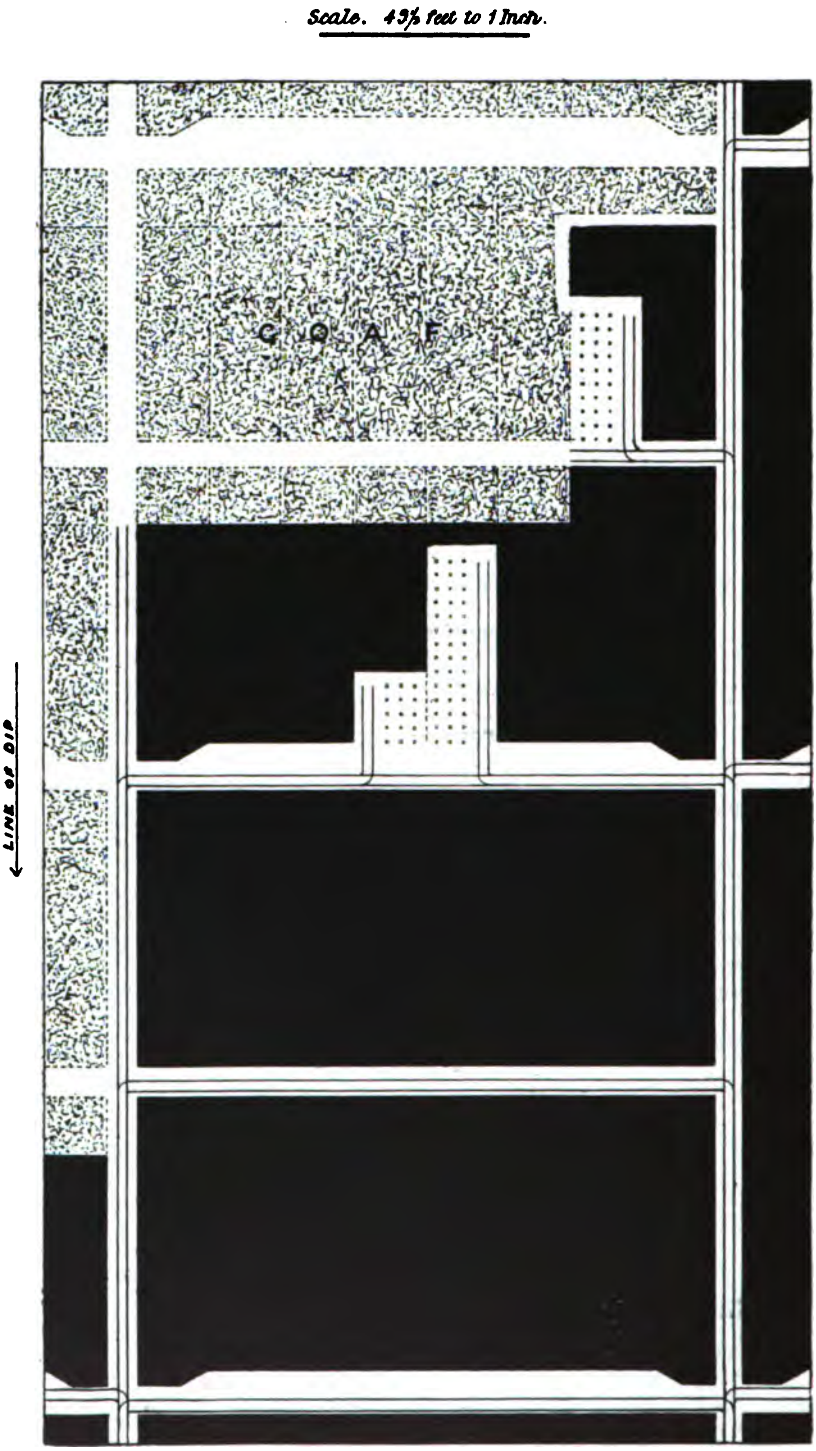


Fig. 220.—Silksworth Colliery, near Sunderland, South Durham. Plan showing the usual method of Pillar Working in the Maudlin Seam

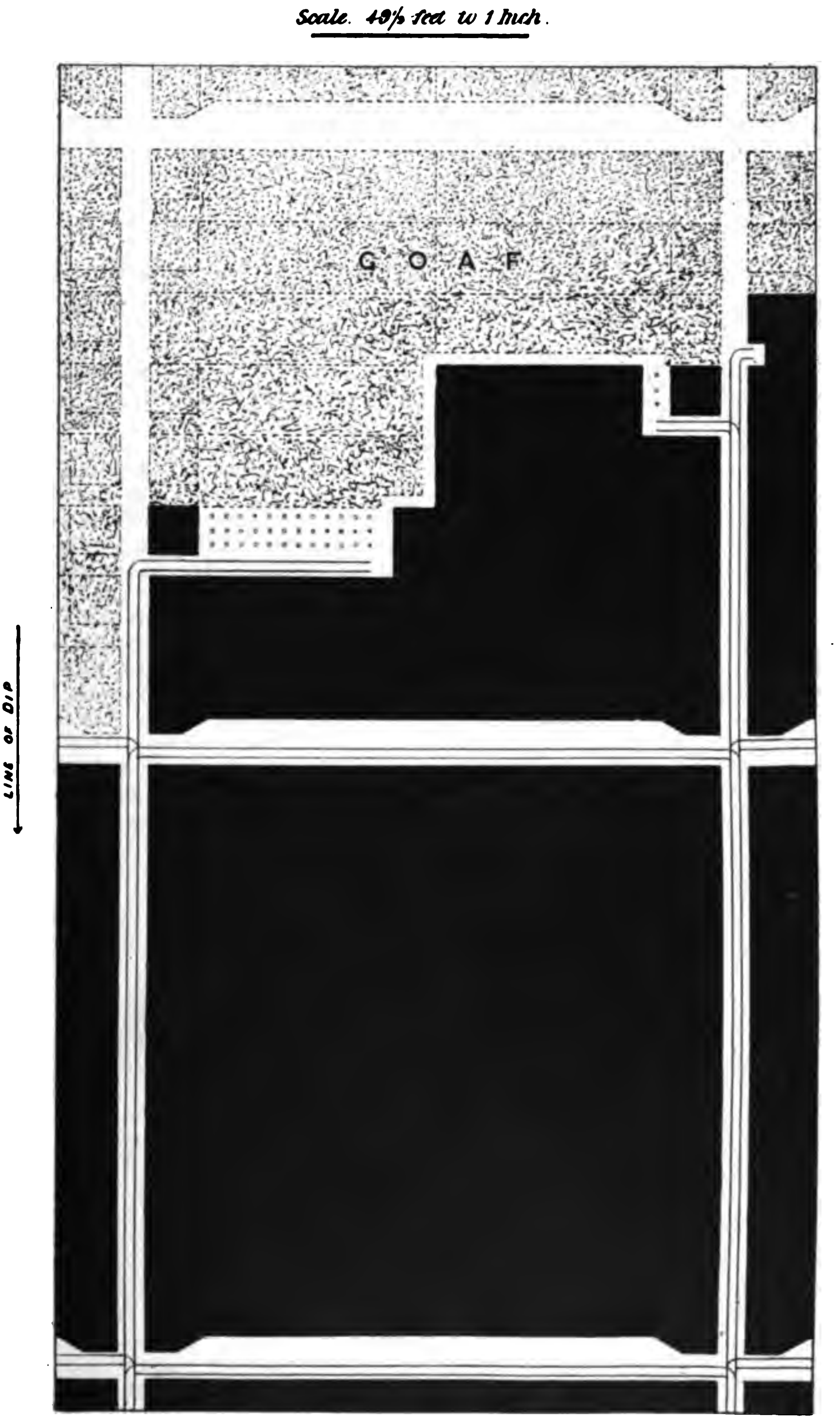


Fig. 221.—Silksworth Colliery, near Sunderland, South Durham. Plan showing Pillar Working under a good roof in the Maudlin Seam.

8 inches of coal is a thin bed of sagger, then 10 inches of bottom coal, followed by 8 inches of sagger, then 18 inches of coal, all of which are taken out in the main roads to make height, but the coals are inferior and unmarketable, and so are left on in the ordinary working. Below come 3 feet of blue metal, and then 3 or 4 fathoms of sandstone. The workings of the Silksworth pit are within half-a-mile of those at Wearmouth, and although the Maudlin Seam is not quite so thick at Silksworth, the coal is very similar at both places. The main roads are timbered as at Wearmouth.

Fig. 220 shows the usual method of working the pillars at Silksworth.. First the 50-yard square pillar is split by means of an 8-foot place. From this split, and also from the ending parallel to it, lifts are taken out. These lifts are 6 yards wide, and each is carried 20 yards to the rise and 5 yards to the dip. In the two lifts being driven at the low side of the pillar in Fig. 220, it is seen that the portion of 5 yards to the dip, belonging to those lifts, is not taken out. That operation would follow the completion of the rise portion of the lift. The lifts are kept "stepped."

Fig. 221 shows the practice of working the pillars at Silksworth, where the roof is good. Here the lifts are driven half-way up the pillar from both sides without splitting it. The place is commenced 6 feet wide, leaving 4 yards of solid coal between the road and the goaf, and is continued this width for a distance of 4 yards, after reaching which it is widened out to the waste. This leaves a "stook" or small pillar, of 4 yards square, between the road and the goaf. The lift is carried half-way across the pillar, or 25 yards, after which the props are drawn by the deputy, and the roof allowed to fall. Two men work in each lift in each shift, there being a fore and back shift. They are paid 8*d.* a ton for hewing and filling the coal. As a fact, the lifts are taken off the pillars on both sides of the split, as shown in Fig. 220, but Fig. 222 shows how they might be arranged to be worked from one side only. The workmen "scallop" the coal, there being no holing, and therefore no spragging. Props and "sets," the same as at Wearmouth, are used, the sets being arranged with the rails between them in the lifts, and placed about 18 inches apart. On the waste side of the road three rows of 6-inch props are placed about a yard apart, as shown in Figs. 220 and 221. There is no rule in force as to the distances apart these props are to be placed, but there is an understanding that they are not to be more than a yard. The deputy sets them, and when the lift is finished he draws the props out, commencing at the inside. The hewers are out of the pit when the deputies draw props. The 4-yard "stook" is then worked off and the props drawn. In cases where the deputies cannot draw the whole of the timber without assistance, timber drawers, who are always spare deputies, are put on to help them. These men receive 8*d.* and 10*d.* a score for taking out the props, but this price is paid only on whole timber, unless the broken timber is required, in which case two broken props count as one whole one. A deputy attends to 10 or 12 men, his duty being to set and draw timber for the men, and see that all the ventilating arrangements are right. He works an 8-hour shift, the same as the colliers, and during his shift visits each place twice. He receives a daily wage.

The whole-mine workings are the same as at Wearmouth. The roof over the coal is good, the cost of timber being 2½*d.* per ton.

At the Florence Colliery, Longton, North Staffordshire, the Great Row Seam is worked by the Longwall method. At this Colliery, which is quite new, two shafts prove the Great Row Seam at a depth of 352 yards. The pits are 12½ and 14 feet in diameter, one being the downcast, the other the upcast. The downcast was sunk 746 yards to the Moss Coal, which is the best household coal of the series, and the sinking of the upcast to the same seam was proceeding in 1881.

The downcast is used to wind coal, for which purpose a pair of high-pressure

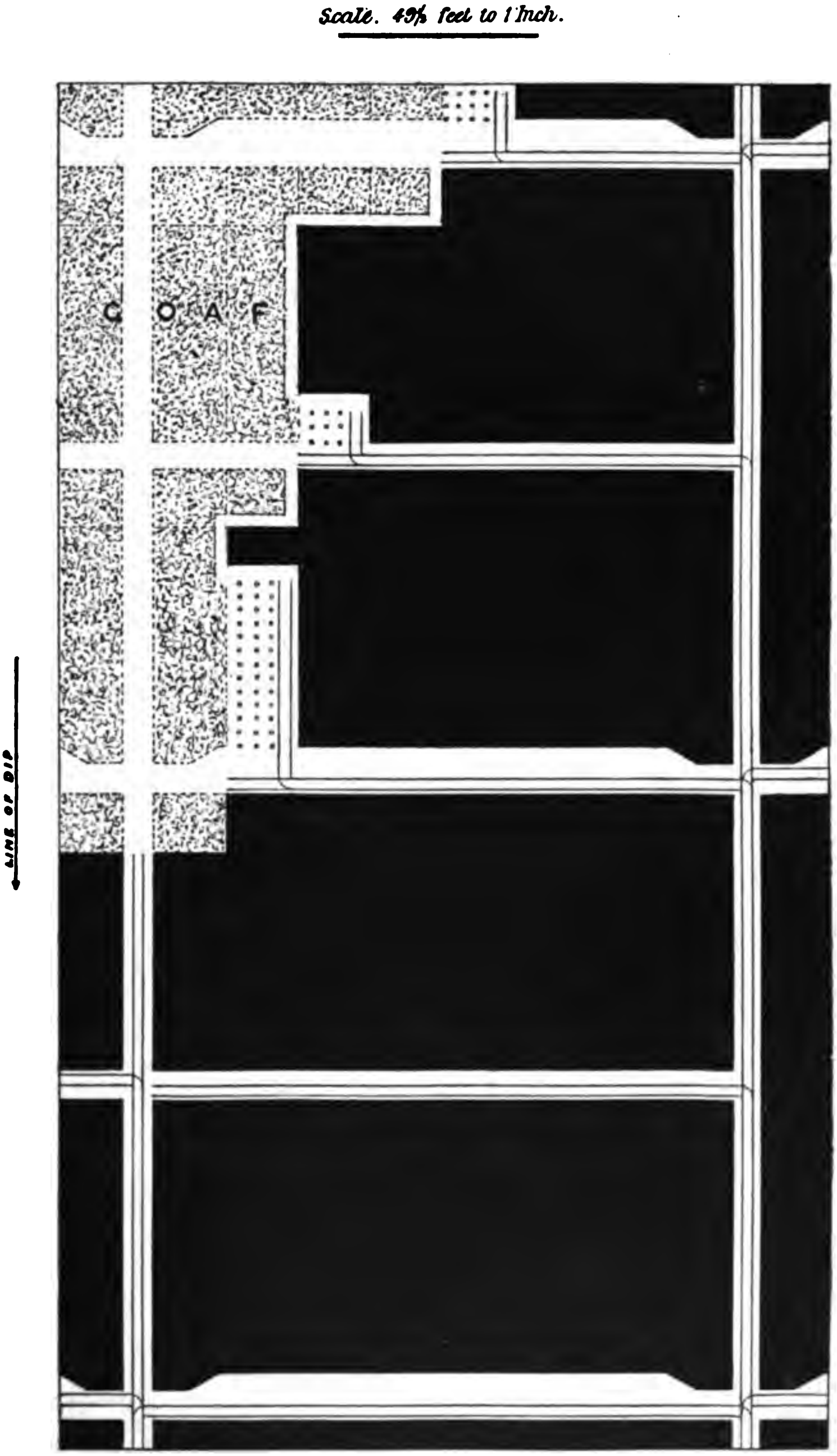


Fig. 222.—Silksworth Colliery, near Sunderland, South Durham. Plan showing a Possible Arrangement of Pillar Working from one side of the Splits, in the Maudlin Seam.

horizontal winding engines, with 42-inch cylinders, 7-foot stroke, double beat valves, and a 28-foot diameter cylindrical drum, are used. The engine is supplied with an automatic cut-off, to stop the cage when it reaches the surface. Coals are placed in the cages at levels or loading stages in the shaft, besides the bottom; these levels being at the Bassey, Chalky and Ash Seams, which are all worked above the Great Row.

In 1881 a Waddle fan, 45 feet diameter, was being erected at the upcast shaft.

Fig. 223 shows a section of the Great Row Seam, the average thickness of which at this colliery is, however, 6 feet. Over the coal are 8 feet of fireclay, and above this a bed of coal 2 feet 6 inches thick. Resting on this coal are beds of fireclay and bass (hard dark shale), 8 feet thick. Above this again come 8 feet of coal and partings, and then higher, 32 feet of "Binds, Slums, and Marls." The thill is composed of 37 feet of "Bass, Binds, and Clod."

The seam dips 1 in 7. Fig. 224 is a plan showing the method of working. From the pit bottom, for a distance of 350 yards, a pair of levels 10 yards apart were driven, and, on reaching a certain point, a "breasting" or face of coal 25 yards wide was taken out, and a "gob" road formed in it 4 yards from the deep side-rib. The stall-roads are turned out of this gob road every 88 yards, and carried to the rise. They are cut off by a level every 120 yards. The faces are "stepped," one being 15 or 20 yards in advance of another. The coal is conveyed along the main level by horses, but a "jig," or self-acting incline works in every going stall-road, by means of which the coal is let down to the level.

Scale. 12 Feet to 1 Inch.

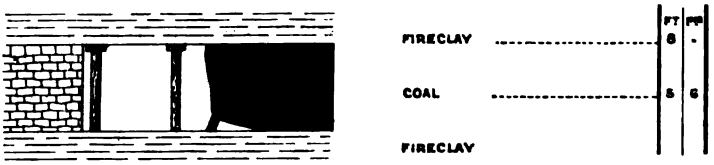


Fig. 223.—Florence Colliery, Longton, North Staffordshire. Mode of Propping and Spragging at the Working Face on the Great Row Seam.

For this purpose a 12-inch wheel is placed in a fork, and the end of the fork is passed through a prop at the road-head. A brake, consisting of a piece of iron, is pressed on the rim of the wheel by a handle working a dumb screw fastened to the fork. Only one tram is run at a time; each tram holds 8 cwts., and the brake is just powerful enough to stop it at any point during the run. A chain is used on the inclines, and as the face advances the wheel is easily moved forward.

The bottom level advances, taking a "breasting" or face of 25 yards. The road is 6 feet wide, and on either side of it a pack-wall, 3 yards wide, is built. The face is protected by two rows of props, which are 6 inches in diameter, the rows being placed 4 feet 6 inches apart, with a 5½ or 6-foot space between the props. The lids used over the props consist of broken props, when there are sufficient for that purpose. Fresh building is put in every 5 feet, the rear row of props being drawn and re-set in advance. Sprags 6 feet apart are used under the coal. The space between the pack-wall and the coal on the deep-side is kept open as long as possible, but when this can only be done with difficulty, a hole is driven through the building from the road, and a fresh air-course carried on from this point. A chock of broken timber is fixed at the corner of each hole, and there is an interval of about 40 yards between these holes.

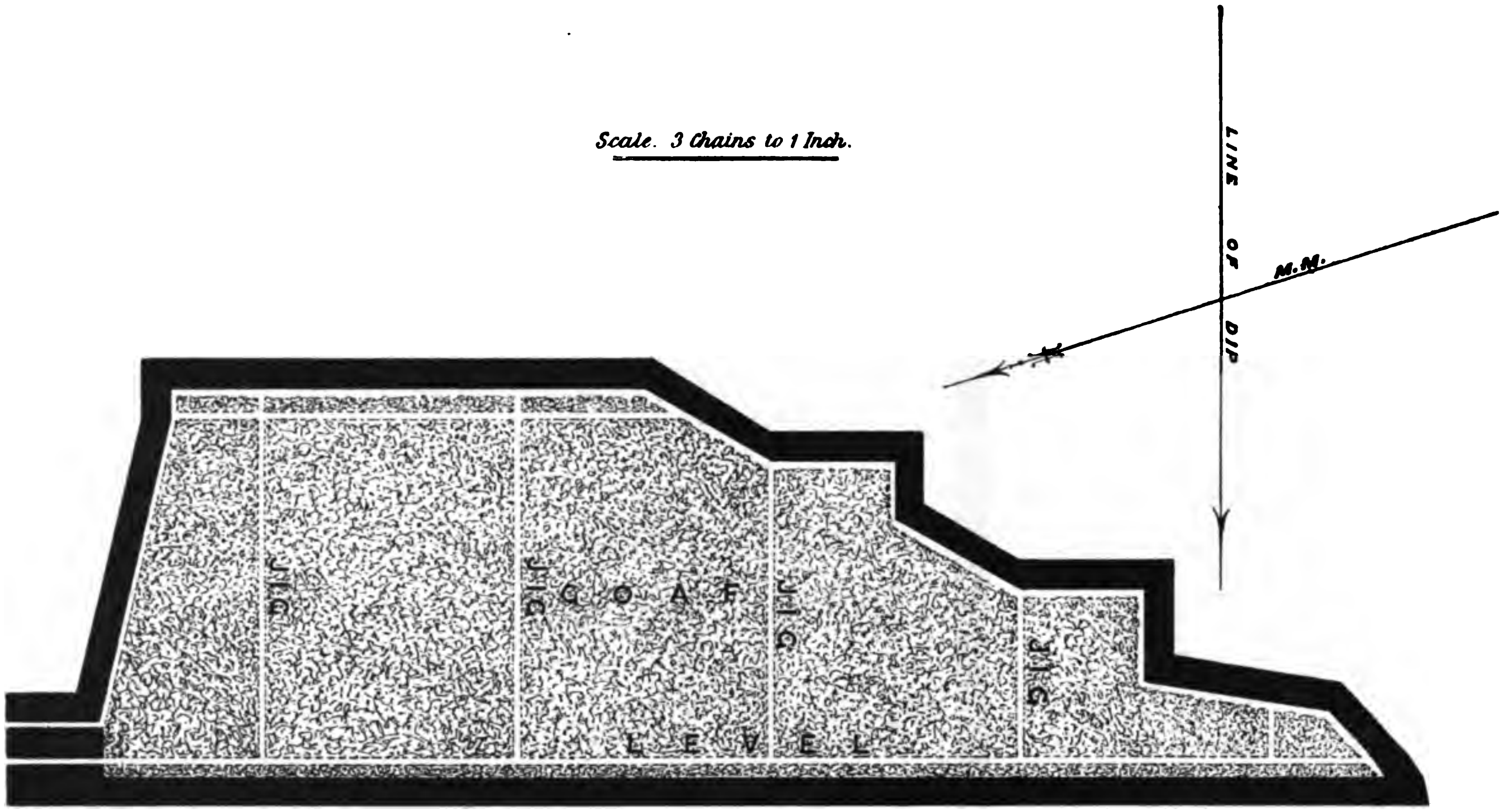


Fig. 224.—Florence Colliery, Longton, North Staffordshire. Plan showing Longwall Method of Working the Great Row Seam.

The stalls are 88 yards wide, and packs 3 yards wide are built parallel to the road (which is wide enough to admit of a double road for the self-acting inclines) 7 yards apart all the way from one stall-road to another.

This will be seen in Fig. 225, which is an enlarged plan showing the packs, timbering, &c. The packs formed next the roads, it will be observed, are wider than those in the waste, being 4 yards. The stones obtained from the 3 feet of ripping are used to build the packs on both sides of the roads. In every 88-yard stall, are two stall-men, one for each side of the road. In their employ are 4 holers, 2 buttockers, and 2 packers.

The stall-men build the packs, set and draw the timber, and despatch the coals down the incline to the level, for which they receive 15*s.* per score of tubs of coal—7*s.* being allowed for slack—all of it being raised. The workmen rake the coal into iron boxes or trays 2 feet square and 6 inches deep at the one end, tapering out at the other. For carrying, these have two handles, which also enable the workman to empty the coal into the tubs conveniently. The men in one of these stalls send out during the day, or shift, 100 tubs, each of which holds 8 cwt. Besides the score price referred to, the stall-man receives 5*s.* 6*d.* a yard, for top ripping and building the road packs. Any other stones required for packs are obtained from the waste. The holing under the coal is made 4 feet in, being 2 feet high in front, the holer placing sprags under the coal every 6 feet, as shown in Fig. 223. The sprags are 2 feet long and 6 inches in diameter. The holing across the stall being finished, the sprags are knocked out; the coal is then blown down by powder. The stall-man fires the necessary shots. The coal is shot down in advance of the point where the rails are laid, but for the convenience of filling, the rails are kept very close up to the "buttock" or piece of coal, next to be blown down, and advanced as required. The holers are not obliged to remove from one side of the stall to the other, as the shots are fired, but only those working on the side where the firing takes place. After the shot is fired, they return to their work and continue as before. The props at the face are in two rows, 4 feet 6 inches apart, there being 6 feet between the props forming a row. They are from 5½ to 6 inches in diameter at the thin end, and are put in by

Scale. 49⅓ feet to 1 Inch.

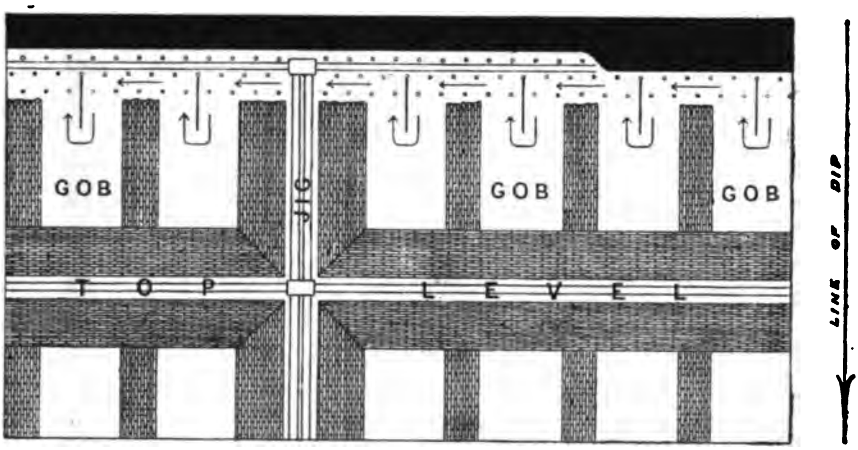


Fig. 225.—FLORENCE COLLIERY, LONGTON, NORTH STAFFORDSHIRE. PLAN OF LONGWALL FACE WITH THE GATE-ROAD AND PACKWALLS CONNECTED WITH IT, IN THE GREAT ROW COAL.

the stall-man, who also draws them out, using a "dog" and chain (as previously described and shown in Fig. 163).

The pack-walls are added to every 5 feet. Where the roof is tender, a chock made of broken timber and set on small coal is put in. As shown in Fig. 225, a sheet is fixed at intervals extending from the face several feet back into the waste lying between any two of the packs, the object of which is to cause the air to travel back into the goaf. Safety lamps are used in the workings, the responsibility of examining which rests with the fireman, who superintends the stall-men, but does no timbering.

In that part of the main level now being carried in the gob, little timber is required, but in that section of it situated nearer the shaft where the pillars remain, there is great side pressure, necessitating much timber.

The sets put in consist of props 8 inches at top and 7 inches at bottom, 6 feet 6 inches long, notched with the Welsh notch (shown in Figs. 181 and 182) into an 8-inch collar, which is notched to receive the props. The sets of timber are placed from 3 to 4 feet apart. Over them, reaching from collar to collar, 3-inch props are laid in the direction of the road and laggings are laid across these. Two men work together in main-road timbering, the notches having been cut before the timber is brought underground. In fixing the sets of timber, the men use an iron bar called a dog. It has prongs at both ends, and when one prop is put in place the dog is used to steady it. The other prop is then placed in its position and the collar put on. When the parts are nicely adjusted the whole is struck with a hammer. Many ingenious contrivances are practised at this colliery for repairing the sets of timber. One is a set of timber in the form of an upright "cocker," as shown in Fig. 226. The props are 5 feet 6 inches apart at the bottom, and 4 feet at the top, while the rise of the span is 8 inches. Again, where the Welsh timbering is broken, a diagonal strut is notched into the prop and the collar as shown in Fig. 227.

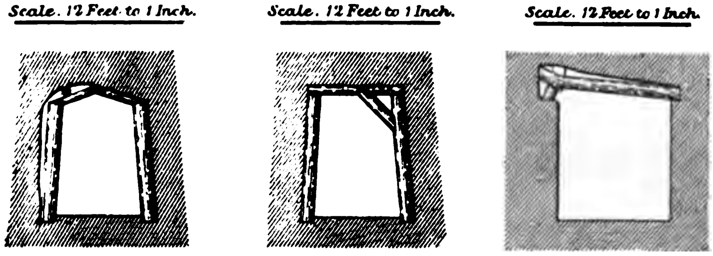

Scale. 12 Feet to 1 Inch.
Scale. 12 Feet to 1 Inch.
Scale. 12 Feet to 1 Inch.

Fig. 226. Fig. 227. Fig. 228.

Methods of Repairing the Timber at Florence Colliery.

As a third instance of repairing, where collars have been notched into the coal without the use of upright props and afterwards broken or partly broken by the weight on them, a short strut is put up under the point of fracture and wedged tightly to the collar, as shown in Fig. 228. The roads are dusty, but as watering causes the floor to rise, it is not carried out.

The roof over the seam, although soft and loose, does not require a large quantity of timber.

At the Great Fenton Collieries, Stoke-upon-Trent, in North Staffordshire, the Great Row Coal Seam is worked on the Longwall system. It is the same seam as that worked at the Florence Colliery, last described. The colliery comprises four pits, two, named the Pender and the Bourne, having been in operation since 1876, working the Blackband Ironstone and the Great Row Coal Seam; the other two, called the Homer and Sutherland, are sunk to the Ash Coal and Littlemine Ironstone. These two have been working since about 1880. The pits are about ¼ of a mile apart. The Pender Pit is 13 feet in diameter, the Bourne Pit 8 feet, both reaching the Great Row Coal Seam at a depth of 308 yards.

The ventilation is produced by a furnace in the Pender Pit, which is fitted with wire-rope guides, two ropes passing through the outside of each of the two cages. At meetings, there is a clearance of 15 inches between these cages. The Pender Pit winding engine has a 36-inch vertical cylinder, with a 6-foot stroke, and 20-foot drum. About 500 tons a day are landed.

The seam dips at an inclination of 1 in 10. A section of it is shown in Fig. 229. The 3 feet 6 inches of top coal is left for a roof in the roads. Immediately over it are 5 yards of moderately strong shale, above which again is a bed of coal 7 inches thick. Resting on this thin bed of coal is shale. This 7-inch bed of coal emits a large quantity of fire-damp. Forming the thill under the coal are 4 feet of hard "sagger," or fireclay, which rests on grey metal. In the Longwall method here adopted, there are 100 yards between the gate-roads. A "Butty" contracts to put the coal into the waggons on the surface for a certain rate per ton or per score. The proprietor supplies all labour and materials. In a face of 100 yards, there are 9 or 10 holers, 4 loaders, and 2 buttockers. The buttockers break out the coal for the loaders, removing the sprags from under the coal for the purpose. There are also four packers whose duty is to build the packs. Each butty has a man specially appointed to take charge of the timber. The duties of this man are to set the chocks and draw the timber out, but the holers set all the props necessary for their safety. The holers and packers work during the night, but the fillers work in the daytime from 6 A.M. to 3 P.M., while coal is being wound in the pit.

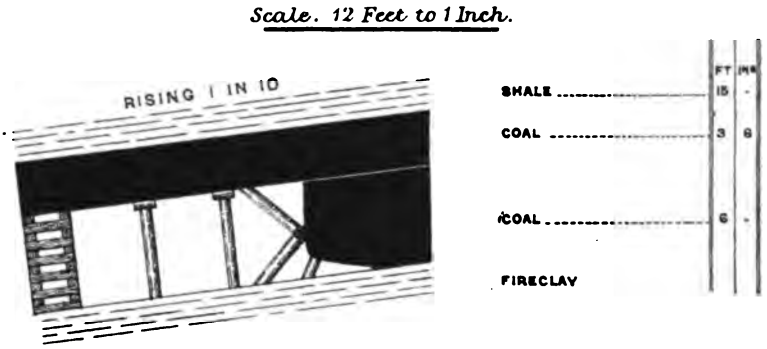


Fig. 229.—Great Fenton Collieries, Stoke-upon-Trent. Cockersprag, Sprag, Props, and Chocks at a working face in the Great Row Seam.

Explosives are used to break down the coal, but no shots are fired till after 10 o'clock P.M. Fig. 230 is a plan showing the method of building the packs, timbering, &c. The roads are made 9 feet wide, and have built on both sides of them a pack-wall 4 yards wide. There are, besides these, other packs built in the waste, parallel to the roads, these packs being also 4 yards wide and separated by 8 yards of space. There is no ripping in the roads, so that stones for all the packs have to be obtained from the waste. Before commencing to hole, the cockermegs, shown in Fig. 229, are placed all along the face. They consist of two 3-foot props, and an old 6-foot prop, the latter being held in a horizontal position against the middle of the coal by the former, which are slightly notched into the roof and floor. The holing is then begun and holed 5 feet in, and as the holing advances along the face, sprags are placed under the coal every 6 feet as sdown in Fig. 229, in addition to the cockermegs already fixed. The packs are kept up within 6 feet of the face, as required by the Special Rules in force, but interposed between the packs and the face, and placed parallel with it, are set two rows of 8-inch props, having 6-foot spaces between the props, the two rows being 4 feet 6 inches apart. The rails are laid along the face between these rows. Besides this double row of props, two rows of timbering are placed between the pack-walls in the waste, these being set 6 feet apart under the top coal. Each of these rows of timbering comprises three props and a chock. The collier shears the top coal which is supported by these props, then draws the timber out, allowing the top coal to fall. He next sets props with a lid and a sill on the rubbish (see Fig. 162), thus securing the roof from which he has just taken down the top coal, and he is

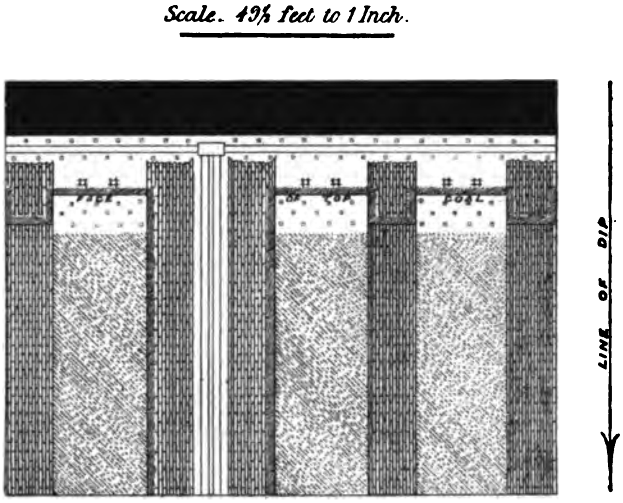


Fig. 230.—Great Fenton Collieries, Stoke-upon-Trent. Plan of Longwall face in the Great Row Coal Seam, showing Props and Chocks.

now able to work at that portion of the top coal remaining over the packs. To obtain this coal, he holes in on the 4-yard packs, for 2 yards on each side, thus allowing the coal to drop. All the coal over the packs is not got, but as much of it as it is prudent to take without running risk from the roof. The props set on the sills over the rubbish are now withdrawn with a dog and chain, by the man specially appointed for the purpose, the chocks preventing the breaking of the roof, which immediately follows, from reaching the face. The top coal is not taken down in the roads, nor from over the packs formed at the sides of the roads. See Fig. 230. In time, however, as the roof sinks, this top coal is ripped in an arched form, leaving a good road which requires little timbering.

There is nothing special at this colliery in the manner of timbering the main roads. Props 8 inches in diameter are used, having lids 6 inches by 5, and 2 feet 6 inches long. In some of the main roads 7-inch collars are notched into the coal on one side and supported by a short 3-foot prop resting on the coal at the other. The roads are dusty and are watered as they require it. The roof generally is very good and safe to work under, requiring little timber.

At the Cannock and Rugeley Colliery, Hednesford, South Staffordshire, the Deep Coal Seam is worked on the Longwall system. The colliery has been in operation since 1865, and in 1881 was working both the Shallow and Deep seams, the former being 22 yards above the latter. The Deep Seam is the lowest of the series in this district, and is only separated from the Silurian measures below by a distance of 46 yards. A peculiarity of the South Staffordshire coalfield is the fact that the Coal Measures repose on the Silurian Rocks, the Old Red Sandstone, the Carboniferous Slates, the Mountain Limestone and the Millstone Grit being absent. Another peculiarity which may here be mentioned is the existence of the "Ten-Yard," or Thick Coal Seam, which is 30 feet thick.

The Cannock and Rugeley Colliery consists of the Cannock Wood Pits and the Pool Pits, the two concerns being separated by a distance of about 1½ miles. At the Pool Pits are two shafts, each 15 feet in diameter and having 70 yards of metal tubbing in it, preventing feeders of water equal to 3,000 gallons per minute finding their way into the shafts. The depth here to the Shallow and Deep seams is 326 yards and 348 yards respectively. Coals are wound in one shaft, 4 tubs, each holding 13 cwts., being placed in the cage each time of winding. The engines used for winding are a pair of horizontal high-pressure cylinders, 32 inches in diameter. As much as 1,051 tons of coal have been landed during a day of 8 hours, but about 900 tons is the average daily output. A 40-foot Guibal fan, 12 feet wide, running 33 revolutions per minute, produces a current of 150,000 cubic feet per minute, with 1·5 inch water-gauge, at the other, or upcast shaft.

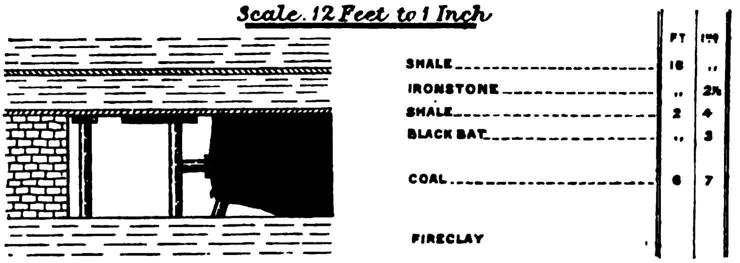


Fig. 231.—Cannock Wood Collieries, Hednesford, South Staffordshire. Section of Deep Seam, and Mode of Propping and Spragging in a working face.

The Cannock Wood pits comprise 3 shafts, 2 of which, 12 feet in diameter, are downcasts, the other, 16 feet in diameter, the upcast. The 3 shafts, after passing through the Shallow Seam at a depth of 178 yards, reach the Deep Seam 200 yards from the surface. The two downcasts are used to wind coal, 800 tons being landed in a day of 8 hours. Two tubs, each holding 13 cwts., are placed in the cage to be wound at one time. All coal is hand-picked, for which purpose the trams are run alongside the waggons, where the coal is sorted into 7 different qualities. Some 50 men on the pit-head sort and place 800 tons daily into the waggons. The sorting is let by contract, the price paid being 3⅞*d.* per ton. A Guibal fan 40 feet in diameter and 12 feet wide produces the ventilation. The fan has 2 engines, each having a 36-inch cylinder with a 36-inch stroke, and working the fan alternately for a month at a time. The fan runs 36 revolutions per minute, exhausting 180,000 cubic feet of air per minute, with a 1·5-inch water gauge. The colliers use the Williamson Safety Lamp. A borehole 30 feet in the coal of the Deep Seam showed a pressure of gas of 35 lbs. to the square inch.

The Cannock Wood pits are situated on an anticlinal axis, the coal dipping on either side of the shafts at an inclination of from 1 in 24 to 1 in 18. The Shallow and the Deep Seams are both worked by the Longwall system. The practice is to keep the workings of the lower seam 300 yards in advance of those in the upper. This plan is found to work better than either advancing them together or keeping the upper seam workings in front of the lower.

Fig. 231 shows a section of the Deep Coal Seam at Cannock Wood, and the mode of timbering at the face there. The coal is 6 feet 7 inches thick. Immediately over the coal are 3 inches of black bat, then 18 feet 6 inches of dark shale with thin ironstone in it, above the shale being 21 feet of white rock. Under the coal is fireclay.

Fig. 232 shows the method of working adopted. The gate-roads are from 30 to 35 yards apart, cross-roads cutting them off every 100 yards.

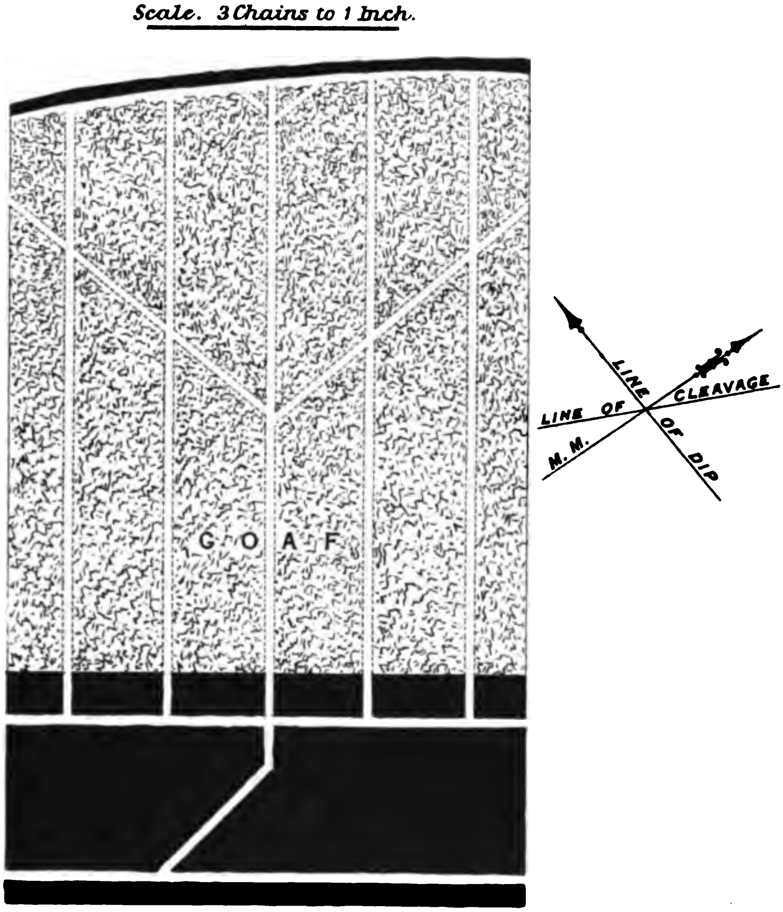


Fig. 232.—Plan showing Longwall Method of Working the Deep Coal Seam at the Cannock Wood Collieries, Hednesford.

Fig. 233 is an enlarged plan showing two gate-roads, with the manner of building the packs, and setting the timber. On each side of the roads the packs are built 8 yards wide. Next these packs are 5 yards of vacant waste, then built parallel to the roads is a central pack 6 yards wide. The road is carried 11 feet wide, having a plate at the road-head, and the rails laid along the face.

Besides the Special Rules in force in the district, regulations are issued by the manager, stating distinctly the distance props are to be set apart, as follow:—

"The distance for props to be apart on the face must not exceed 3 feet, and each prop must have a suitable lid on it. The distance between the props for a tram-road in between must not exceed 5 feet, and a prop must be set within 4 feet 6 inches of the buttock of coal.

"The sprags under the coal must not exceed 5 feet apart, and they must be well set.

"The stall-men or contractors to see that the timber is set as above, and to see that the workmen employed by them do not work in danger.

"The stall-men or contractors to examine the way end, and all parts of the stall, and if there is any danger the men must not be allowed to work.

"The chain and bar must be used in drawing timber. Workmen found endangering their lives by neglecting to use the chain and bar, or by not setting timber to secure the roofs and sides whilst drawing timber, will be sent out of the mine at once and summoned before the magistrates.

"The competent men must see that the timber is set, as above, and examine the coal, roof, gob road and wastes of each stall, and if they find any danger, to stop the stall, and report the same at once to the underground manager. The underground manager to go and examine the stall, and if he finds it in a dangerous state for want of the timber not being properly set, the stall must be stopped until the danger is removed, and the stall-men to be fined not less than the sum of two shillings and sixpence for each offence."

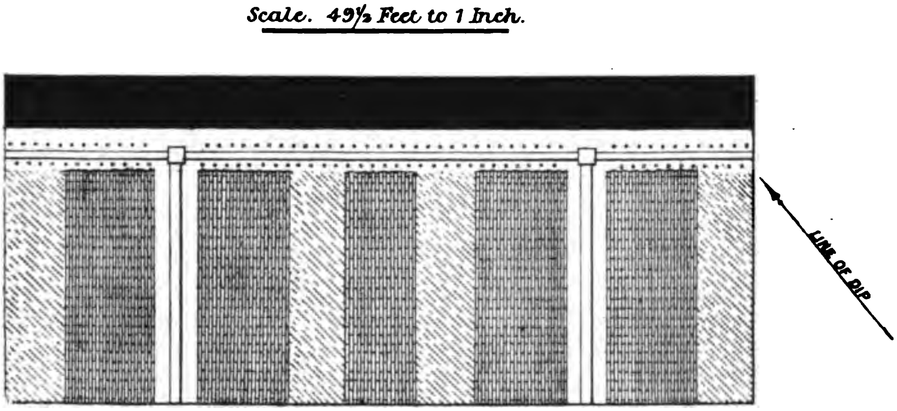


Fig. 233.—Cannock Wood Collieries, South Staffordshire. Plan showing Longwall Face in the Deep Coal Seam, and the Stalls, Packs, and Props connected with it.

These regulations apply to the Deep Coal, and a corresponding set are framed for the Shallow Seam, in which the props at the face must not be more than 4 feet apart and not more than 5 feet between the first prop and the building.

The coal in the Deep Seam is holed under, 5 feet 6 inches in, and is supported by sprags placed 4 feet 6 inches apart. The sprags are 2 feet long and 8 inches in diameter. 2 feet back from the face, a row of 8-inch props, having broken props for lids, are set 3 feet apart parallel to the face. See Fig. 231. From the centre of each prop on the side towards the coal, is placed a short 8-inch sprag, at the other end of which is a lid bearing on the face of the coal. 5 feet back from this, another row of props with lids is set, the props being 3 feet apart, and parallel to the row in front. Close behind the last-named rows are the packs. The rails are laid along the face between the two rows of props as shown in Fig. 233. This method of getting the coal gives a large percentage of small coal, and an experiment is being made in another way. This is called "banicking."

Instead of holing in the bottom of the coal, or in the coal at all, the men hole 3 feet in the shale reposing on the coal and break the coal off with crowbars. In this way it parts at the planes of cleavage and gets into a series of steps. This method works more satisfactorily than the other, yielding 80 per cent. of large coal. In carrying out the system sprags are put in between the roof and the coal.

In each stall two stall-men or small contractors, employ two holers, and two fillers. They place the coals into the tubs, receiving 1*s.* 4½*d.* per ton on coal for doing so. The holers receive 2*s.* 3*d.* a fathom for holing 3 feet under the coal, and for putting in the sprags every 6 feet as they go. Each of these men can hole 2 fathoms in a day of 8 hours. The wages of the fillers or loaders vary from 2*s.* 9*d.* to 3*s.* 9*d.* per day. The stall-men set up, and draw the timber out; in the latter operation, they use the "bar and chain." The roof is bad and falls when the props are withdrawn from under it. Part of what falls is used to build the packs, the rest being obtained from the ripping in the roads. The stall-men rip the roads and build the packs. The roads are 11 feet wide and are ripped 3 feet, powder being used in the process. The stall-men receive 2*s.* 10*d.* for ripping the road, which is generally, but not always, done at night.

When the holers have finished the holing on one side of the road, they remove to the other, the coal on the side they have left being shot down without interrupting their labour.

The powder is used in cartridges; the drills are all made to a suitable gauge, so that the cartridges may fit accurately the holes made by the drills. The shot-firing is carried out by a competent person, appointed by the proprietor. He superintends 8 stalls, and before commencing to drill a hole, the men consult him. The roads are timbered as required. The sets of timber in the horse-road are placed 3 feet apart. The props are 6½ inches at the thin end and 7 inches at the thick end, the thin end being placed uppermost. The collars are 7 feet long and are 8 inches square. The props are 7 feet high and are placed 4 inches nearer to each other at the top than at the bottom. In the engine planes the collars are 9 feet long. As shown in Fig. 232, the cross-roads are not carried at right angles to the stall-roads but on a slope. Chocks are placed at the corners of the buildings, 9 feet long, 2 feet wide at one end, and 5 feet 6 inches at the other, old timber being utilised in building them.

Timber costs from 3*d.* to 4*d.* per ton, the price paid at the colliery being 32*s.* 9*d.* per 100 lineal feet for props not less than 6½ inches at the thin end. The roads are dry and dusty and are watered at irregular periods.

At the Pemberton Colliery, near Wigan, in Lancashire, the Orrell Five-feet Seam is worked by Longwall. The colliery has been working since about 1870, and consists of 4 pits, which are separated by a short distance. They are the King pit, an upcast, the Queen pit, a downcast, a shallow pit which is also a downcast, and the pumping pit. Of these, only the King and Queen pits are used in which to wind coal. The winding engines are placed between the two pits, the two engines being in one house. They are similar engines, each having a pair of 36-inch cylinders with Cornish valves. The Queen pit winding engine has a scroll drum, the diameter of which commences at 19 feet 8 inches and increases to 30 feet 10 inches. Its weight is about 48 tons. The cages have 3 decks and receive two tubs in each. 6 tubs, each holding 8 cwts., are thus raised at each winding. The drum of the King pit engine on the Wigan Four-feet rope side is 15 feet 2 inches in diameter, and that on the Cannel side is a scroll from 15 feet 2 inches to 19 feet 4 inches.

The method of removing the stones from the coal is worthy of remark. A band of flat hemp ropes is fastened together by iron bands and placed at the foot of each screen. It is 14 feet long and is carried round pulleys at either end. A shaft by the side of this band is kept constantly revolving, and by means of a clutch the band may be thrown in or out of gear with the revolving shaft. The band is endless, and when thrown into gear moves at the rate of about 35 feet per minute. When about to discharge the coal from a tram over the screen the wheel carrying the band is thrown into gear. On the end of the screen are doors, worked by a lever, which allows the coal to fall thinly on the band as it moves. When all the coals are on it, the band is stopped by being thrown out of gear, and two women, one at each side, pick out the stones very readily. After the coal is well cleaned of impurities, the band is thrown into gear again, and the coals placed into the waggon. From 100 to 120 tons are thus daily passed over each screen, but this quantity could be greatly increased if necessary. The small coal passes through the screen bars and is led into a pit below. From here it is taken by means of a screw working in a trough, and is lifted by an endless pitch chain fitted with iron buckets, like a dredger or elevator, the coal being dropped on a screen and the duff taken out.

A Guibal fan, 46 feet in diameter and 14 feet 10 inches wide, is placed at the top of the upcast shaft. The fan is run 36 revolutions per minute, and gives 275,000 cubic feet of air with the water-gauge reading 2·5 inches. A direct-acting pumping engine is used to raise the water from a depth of 190 yards. Two sets of pumps are used, a lifting set in the bottom, having a 20-inch brass bucket, lifting the water 70 yards, and the one above a forcing set with a 20-inch ram raising the water the remaining 120 yards. The pumping-engine works 10 strokes a minute.

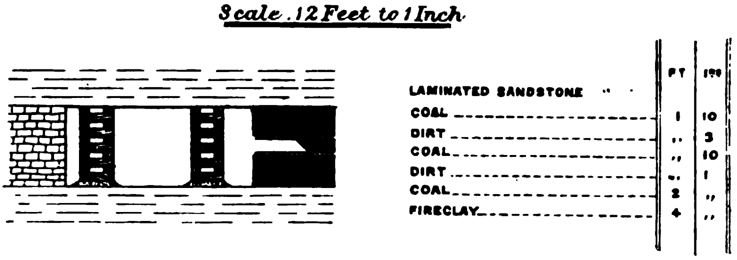


Fig. 234.—PEMBERTON COLLIERY, NEAR WIGAN, LANCASHIRE. METHOD OF TIMBERING A WORKING PLACE ON THE ORRELL FIVE-FEET SEAM.

The Queen pit cuts the Orrell Five-feet coal seam at a depth of 566 yards, and the Arley Seam at a depth of 626 yards. The Arley Seam corresponds with the Silkstone Seam in Yorkshire. The coals worked in the Orrell Five-feet Seam are not put into the Queen pit cages at the 566-yard level, but at the bottom in the Arley Mine. They are transferred from the one seam to the other by means of a "blind pit," or "drop staple."

One of Fowler's clip-pulleys is placed over the blind pit at the Orrell Five-feet Seam. A wire rope attached to one cage at the bottom of the blind pit in the Arley Seam passes up the pit, round the pulley, and is attached to the other cage at the Orrell Five-feet Seam. A full tub is placed in this cage and an empty one in the other, and on lifting the brake from the pulley, the weight of the full tram causes that cage to descend and the other to be raised. The speed is regulated by the brake.

By this arrangement there is only one loading-stage in the Queen pit, all the coals being brought to the pit bottom and caged there.

The Orrell Five-feet Seam is tender in its nature, and yields a household coal. When passed over an inch screen it gives 61·3 per cent. of large coal.

Fig. 234 shows a section of the seam and the method of timbering at the face.

The coal is 5 feet thick. Over it, as a roof, is a "lea stone," or laminated sandstone, with ironstone balls, and in the roads 2 feet of this is ripped down. The coal rests on "sagger," or fireclay. Holing the coal is not regularly carried on, but the 10 inches of coal in the middle of the seam is used for the purpose when holing is done. The bottom coal is then lifted up, and the top coal supported on sprags placed under it by the collier. Afterwards, on knocking away the sprags, the top piece of coal falls. The collier sets props as well as sprags when they are required. The props used are from 3 to 4 inches in diameter.

The seam dips at an inclination of 1 in 12, and was formerly wrought by the Pillar and Stall method, the pillars left being from 20 to 30 yards square.

This method is giving way to the Longwall, as shown in Fig. 235, but some pillars are still being worked off. The coals are brought from the level next the face to the top of the blind pit down a "spunney," or self-acting incline. The stall-roads are driven to the full rise, as shown in Fig. 235, there being 30 yards from the centre of one road to the other. Levels running at right angles cut off the old stall-roads every 100 yards. These levels are made 8 feet wide, and have a pack-wall 4 yards wide on each side. The stall roads or brows to the face are made 7 feet wide, and have a pack-wall 3 yards wide on either side of them, as shown in Fig. 236. Four men work in a stall, two on each side of the road, and they sometimes take their own tubs of coals out to the "spunney." In other instances, two men on one side of a stall employ a younger man as their putter, and he draws the coal, and also assists to fill it.

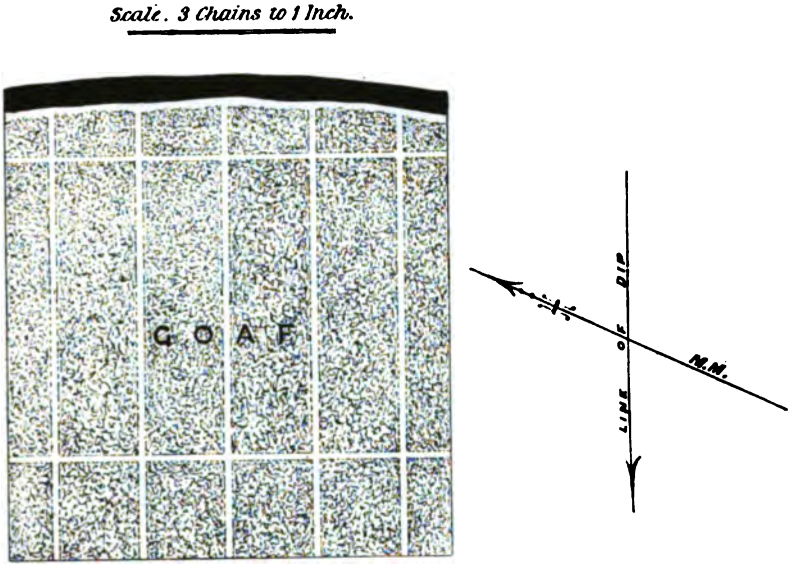


Fig. 235.—Pemberton Colliery, near Wigan, Lancashire. Plan showing Longwall Method of Working the Orrell Five-Feet Coal Seam.

The price paid the collier for holing and drawing is 1*s*. 3*d*. per ton. The methods of building the packs and timbering at the face are shown in Figs. 234 and 236. The latter consists in placing two rows of chocks 6 feet apart all along and parallel to the face. The distance between the two rows is 5 feet, the rails being laid between them for the tubs to be taken along.

As a third row of chocks is set, the rear row is drawn and shifted forward. The drawing of the chocks is followed by the roof breaking off behind. The fireman sets the chocks, and for the convenience of after-removal they are built on 4 or 5 inches of small coal. They consist of billets of wood 6 inches square in section, and 2 feet long, and are a convenient size to be quickly fixed. A fireman puts one up and wedges it tightly in 5 minutes. He receives a daily wage, and attends to from 20 to 30 men. The ripping is done by a contractor, and this man takes out 2 feet from either the roof or the floor of the roads, builds the packs, which are kept within 3 or 4 yards of the face, and draws the chocks and other timber. For ripping and building the packs he is paid 10*s*. per yard; for taking out the props and chocks he receives 2*s*. 6*d*. a score for large props, but nothing for the small ones, and 9*d*. for every chock. Occasionally, where the roof is tender, he puts up a chock, receiving 3*d*. for so doing, but the fireman is expected to put in all necessary chocks. The props which the collier puts in are removed when the chocks are set. Very little timbering is required in the main roads. Where necessary, props and lids, or sets of timber, are placed. The sets are put up without any kind of notch. A special class of men put up the main-road timber during the night. They receive directions from the man in charge of the night-shift as to what repairs are necessary, &c. The roads are dusty, but are only sprinkled with water, as a more copious supply causes the floor to rise. No blasting is done in this seam, nor in any other at the Colliery, except the Pemberton Five-feet Seam.

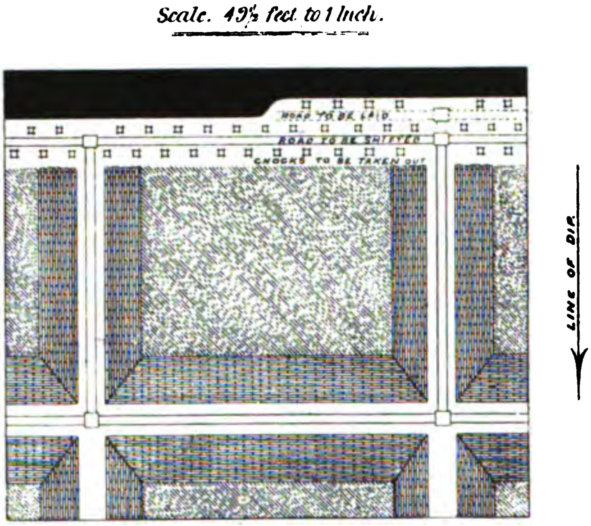
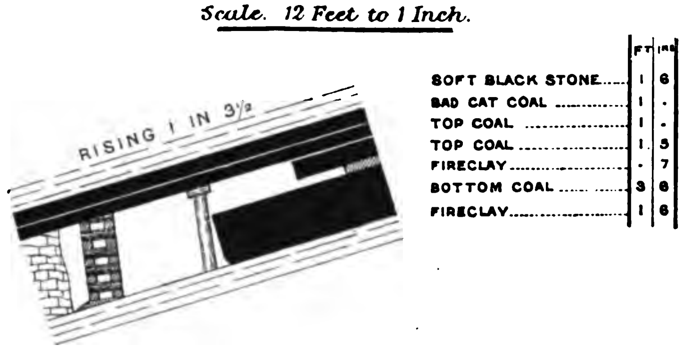


Fig. 236.—Pemberton Colliery, near Wigan, Lancashire. Plan showing Longwall Face in the Orrell Five-Feet Seam, with Packs, Chocks, and Roadway along Face.

In the Pillar and Stall workings before alluded to, pillars are left 20 yards by 30, the level roads being 3 yards and the drifts 4 yards wide. In working off the pillars, lifts, or "ratches," 5 yards wide, are worked along the whole 20 yards of the pillar. When three lifts have been thus taken off, lifts are taken out up-hill. Whilst a 5-yard lift is advancing up-hill a line of props and sets of timber are kept over the road. On the waste side of the road are 3 rows of props and lids, there being 3 feet between the rows and 3 feet between the props in a row. The props are 4 feet 6 inches long and 6 inches in diameter at the thick end. The lids are 18 inches × 5 × 2½, being made out of old props or collars. In a 20-yard lift are 100 props, 3 rows of 20, and the double row over the road of 40. The colliers fix these props, which are afterwards withdrawn by the fireman. As soon as the colliers have finished a lift, the fireman does not wait till the men have gone out of the pit, but draws the props at once. Arrangements are being made to draw the props at night, but if it is thought that a lift will not stand till then, it will be drawn without delay. In cases where the pillars are fallen close, a strip 3 yards wide, called a "ribbing," is carried to the top of the pillar, the lifts afterwards being taken out in the ordinary way. The cost for timbering at the colliery is 1*d.* per ton in the Longwall, and about ¼*d.* per ton in the Pillar and Stall whole workings; in the Pillar and Stall broken, about the same as in the Longwall. Pit timber at the colliery costs 25*s.* per 100 lineal feet or 37*s.* per ton, and the billets of wood for the chocks cost 7*d.* each.

At the Clifton Hall Colliery, near Manchester, the method of working consists in first driving levels to the boundary, marking out blocks of coal into pillars, and working back towards the shaft.

The Colliery has been working since about 1854, the Doe, Five Quarters and Trencherbone Seams being now worked. An upcast and a downcast shaft are both sunk to the Trencherbone Seam. The latter solely is used for winding. The Trencherbone is the lowest of the three seams worked, being 540 yards from the surface in the downcast pit. All coal is caged at the Trencherbone Seam level. A cross-measure drift is driven water-level from the Trencherbone Seam to the Five Quarters and the Doe Seams, and the coals are brought along it. Three-decked cages are used, and the engines are fitted with spiral drums. To overcome the difficulty of changing tubs at the top and bottom of the shaft simultaneously, a balanced scaffold is placed at the bottom. The onsetter there first changes the lowest tier of tubs; he then lowers the scaffold by means of a brake and changes the other tiers. 600 tons of coal are thus daily raised.



	FT	INS
SOFT BLACK STONE	1	6
BAD CAT COAL	1	·
TOP COAL	1	·
TOP COAL	1	5
FIRECLAY	·	7
BOTTOM COAL	3	8
FIRECLAY	1	6

Fig. 237.—Clifton Hall Colliery, near Manchester. Section of Doe Coal Seam, and Timbering in it.

A furnace, assisted by the steam from an underground hauling engine, and also from the furnace of the boilers supplying steam to the engine, causes the ventilation. The quantity of air passing up the upcast shaft is 100,000 cubic feet per minute.

Fig. 237 is a section of the Doe Coal Seam. The seam, including the 18 inches of soft black stone and the 7 inches of dirt, is 9 feet thick, but of this the top 3 feet 6 inches is left as a roof. This 3 feet 6 inches includes the 12 inches of top coal, 12 inches of bad cat coal, and 18 inches of soft black stone. Above this are 3 inches of ironstone, 3 feet 6 inches of hard, brittle (short) shale, and above this white metal. Under the coal are 18 inches of warren earth or fireclay, and below this is a hard grey metal. The 7 inches of "Daugh" or fireclay is used to hole in, and the holing is carried a yard in, after which the 1 foot 5 inches of top coal is taken down. The holing is continued another yard, and the top coal over it taken down. This is continued until 9 feet of bottom coal is bared, which is then got by blasting.

The seams dip at an inclination of 1 in 3½.

The method of working the Doe Coal Seam will be understood by reference to Fig. 238. From the extremity of the cross-measure drift between the Trencherbone and Doe Coal Seams a pair of levels separated by 30 yards of coal are driven to the boundary. Openings or cut-throughs connect the levels every 40 yards. At a point 200 yards to the rise another pair of levels is driven parallel to the first pair, the coal between the two pairs being at first left solid. On the lower pair of levels reaching the boundary a pair of places, separated by 30 yards of coal, is driven out of them to the full rise from a point 200 yards back from the boundary, and these places are continued until they hole into the upper levels Levels and throughers, all 7 feet wide, are then driven so as to divide or split up

Scale. 3 Chains to 1 Inch.

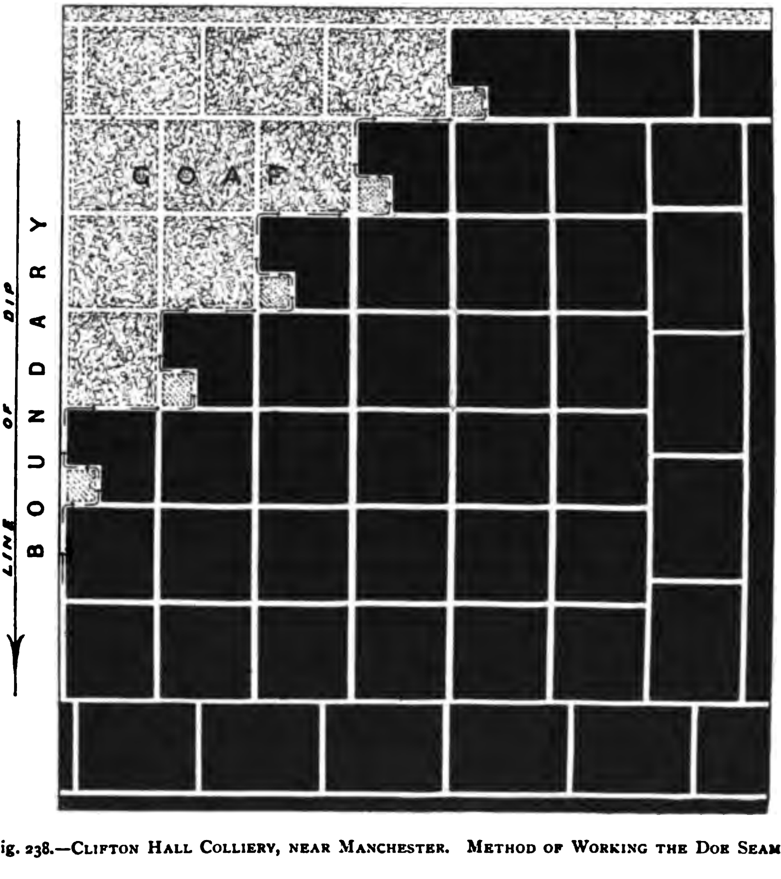


Fig. 238.—Clifton Hall Colliery, near Manchester. Method of Working the Doe Seam.

the 200-yard square block into pillars of 30 yards square. The pillars are then worked off by taking lifts 12 yards wide across them, the coals being brought down to the levels. During the time of working these pillars the next 200-yard block outside is being split up into the 30-yard pillars, so that at no time is there more than one range of pillars being worked.

In the levels being driven to the boundary the men get 6*s*. for every 3½ tons of coal left from riddling through a ¾-inch riddle. The levels are 7 feet wide, and the throughers from 5 to 6 feet. The air is carried to the face of the levels by brattice from the last througher. It is so placed as to form an intake of 5 feet wide on the dip side, and a return of 2 feet on the rise side of the road. The coal left on makes a very good roof, and no timber is required at the face, but further back where this top coal is taken down and the fireclay up, a good deal of timber is required, as the road is then secured with sets of timber. There is no specified distance between the sets, but they are put in where required by the men specially appointed for the purpose. Usually the distance apart is 3 feet. The props and collars used are 8 inches in diameter, the tops of the props being rounded to receive the unnotched collars. Where the weight on the collar is excessive it splits the prop down the centre like a wedge. Rafters are placed reaching from collar to collar in a line with the road, and laggings in between the rafters. Parallel to the strike are cleavages called "shuttles" in the coal. Frequently these "shuttles" set free large blocks of coal, which fall out into the roads, and thus constitute a source of danger.

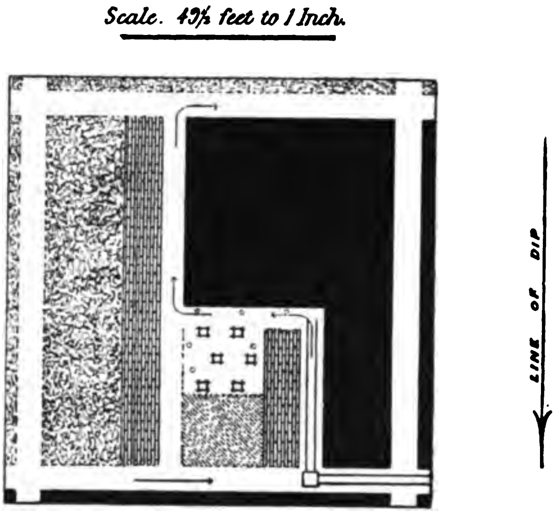


Fig. 239.—Clifton Hall Colliery, near Manchester. Plan of Lift on the Doe Coal Seam.

In the pillar workings all the seam is worked without leaving the top coal on. Fig. 239 is an enlarged plan of a pillar showing the method of taking off the lifts. Here, one 12-yard lift has been taken off across the pillar, and another is proceeding. A 6-foot wide road is carried up the side of the coal, and on the waste side of the road a 9-foot pack is built of the cat coal. Two chocks are built 6 feet back from the face and parallel to it, the chocks being 5 feet apart. Parallel to these 2 chocks and 5 feet behind them are built other two, shown in Fig. 239, whilst the 2 forming a third line of chocks are about to be removed.

The chocks are built of billets of wood, 6 inches square in section, and 2 feet long, upon 4 inches of small coal. When, owing to the great inclination of the seams, the billets do not close nicely against the roof, slips of wood are put in between the last billets and the roof to make the required close. Between the chocks and the face a row of props is set, and other props are put up as required between the different chocks. The props are of larch, 3 inches thick and 5 feet 6 inches long. The lids placed over them are 18 inches × 5 × 2½. When the rear row of chocks is withdrawn the top coal falls in the waste, and is taken out. Three men and a lad work in the place. They hew the coals, fill and let them down the "jig brow," build the packs, and set and draw all the timber required. They are paid 6*s*. 3*d*. for getting 10 waggons, or 3½ tons of coal, and 4*s*. a yard for building the packs. The coal is riddled through a ¾-inch riddle, and for 3½ tons of small coal they receive 2*s*. 6*d*. Twenty per cent. of the total quantity got is small. The 3 men and lad send out about 10 tons a day. The fireman superintends these men, and sees that the necessary props and chocks are put up.

The roof in the roads is not good; here the top coal is left on, but is taken down in the waste. Timber costs 2½*d*. a ton in this seam.

Fig. 240 shows a section of the Five-Quarters Seam, which is 24 yards below the Doe Seam. The coal is 3 feet 4 inches thick with the fireclay partings included. Of this only the 1 foot 10 inches, which is of excellent quality, is sent out. Over the coal is hard sandstone reaching up to the Doe Coal. The 6 inches of "daugh" or fireclay is used to hole in. Under the coal is "warren earth" or fireclay. To make height in the roads, water is put on this to render it soft, and it is afterwards easily taken up.

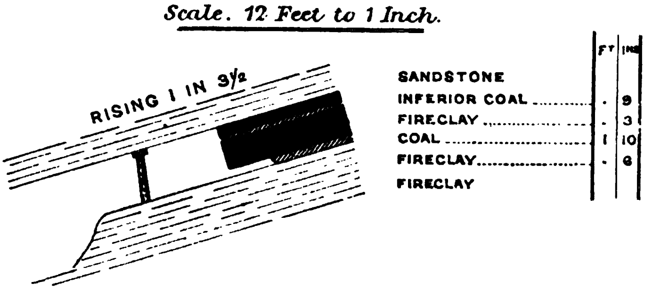


Fig. 240.—CLIFTON HALL COLLIERY, NEAR MANCHESTER. SECTION OF FIVE-QUARTERS COAL SEAM.

The method of working pursued on this seam differs slightly from that last described on the Doe Coal. Here, as shown in Fig. 241, instead of driving a pair of 7-foot wide levels separated by 30 yards of coal, a face or breasting of coal 13 yards wide is driven, and a single level formed in it. This road is not carried in the middle of the breasting, but to one side, in a manner similar to that in the Ram's Mine at Pendlebury Colliery, shown in Fig. 247, and fully described later on. A building 6 yards wide is formed on the rise side of the road, and another 3 yards wide on the dip side. Between the pack and the ribside on the rise side is an aircourse 3 feet wide, the aircourse being carried alongside the coal as the level advances.

A similar level is driven from a point 140 yards to the rise of this (see Fig. 241), the levels being carried thus to the boundary. When this is reached a place is driven to the full rise between the lower and the upper level from a point 140 yards back from the boundary. A block of 140 yards square is thus left. Commencing at the face of the level this 140-yard square pillar is now worked by lifts 20 yards wide being carried to the rise across it between the two levels. The road is carried in the centre of the lift, and protected on each side by a facing built of the 9 inches of inferior coal. The rubbish yielded in ripping the roads and in holing the coal is sufficient to fill the whole space in the waste. An aircourse is formed along the ribside. No chocks are used, but props are set 6 feet apart along the face. The props are 3 inches thick, and 3½ feet long, the lids over them being 2 feet × 5 inches × 2½ inches. The props are not set at right angles to the floor of the seam, nor in a vertical position, but at an inclination between the two, so that as they afterwards sink they assume a position more nearly at right angles to the floor. Only one lift at a time is taken out across the pillar. The collier receives 8*s*. 8*d*. for 3½ tons of riddled coal, but he has to draw the coal to the foot of the second jig brow for this, some 200 yards distant. He puts in all the props, and receives 3*s*. a yard for the pack facings in the road. Four men work in a lift, and they send out about 7 tons of coal a day. From experiment it is found that this seam works badly if the Doe Coal is first worked over it. The rock roof is excellent.

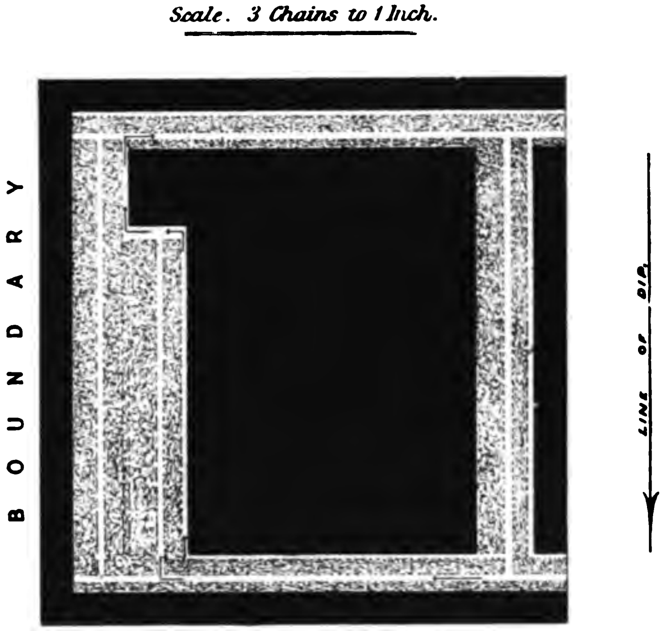


Fig. 241.—Clifton Hall Colliery, near Manchester. Plan showing Method of Working the Five-Quarters Coal Seam.

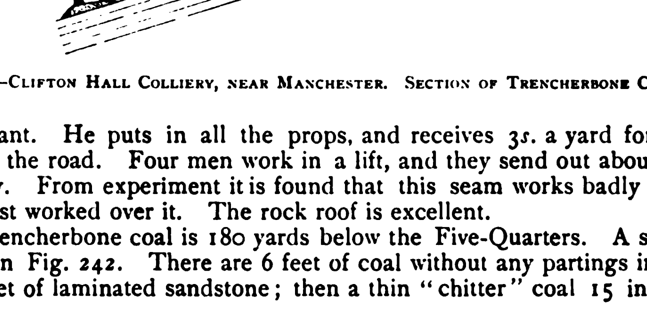


Fig. 242.—Clifton Hall Colliery, near Manchester. Section of Trencherbone Coal Seam.

The Trencherbone coal is 180 yards below the Five-Quarters. A section of it is shown in Fig. 242. There are 6 feet of coal without any partings in it. Over it are 4 feet of laminated sandstone; then a thin "chitter" coal 15 inches thick, called the California coal; over this, again, is a very hard red sandstone, 24 yards thick, called the Trencherbone rock. Under the coal are 18 inches of "warren," and beneath this are 4 feet of "warren," 4 yards of shale, a thin coal, and then shale.

In working the Trencherbone Seam in the Manchester district, boulders have been found in the coal, in the roof over the coal, and sometimes half embedded in both coal and roof. The Trencherbone is by no means the only seam of coal which has yielded these boulders; they have been discovered in the Arley Mine at Burnley, in the Roger Seam of the Astley Pit, Dukinfield, in the lower coal measures (the gannister coal or Mountain Mine) at Bacup, Lancashire. The last mentioned are 1,000 feet below the Arley. Some coal seams of the United States yield boulders, having the same character and composition, and occupying the same position in the coal seams as those in the Lancashire coalfield. As a rule, they are hard, siliceous grits or quartzites of a pale or dark grey colour. They vary much as to size and form, but all are smooth, and often polished, with corners rounded off by abrasion, giving evidence of their being water-worn before being deposited in the coal strata. These boulders are well worthy of notice, as they have caused much speculation among geologists as to how they were brought into their present position, and from whence. They are older than any

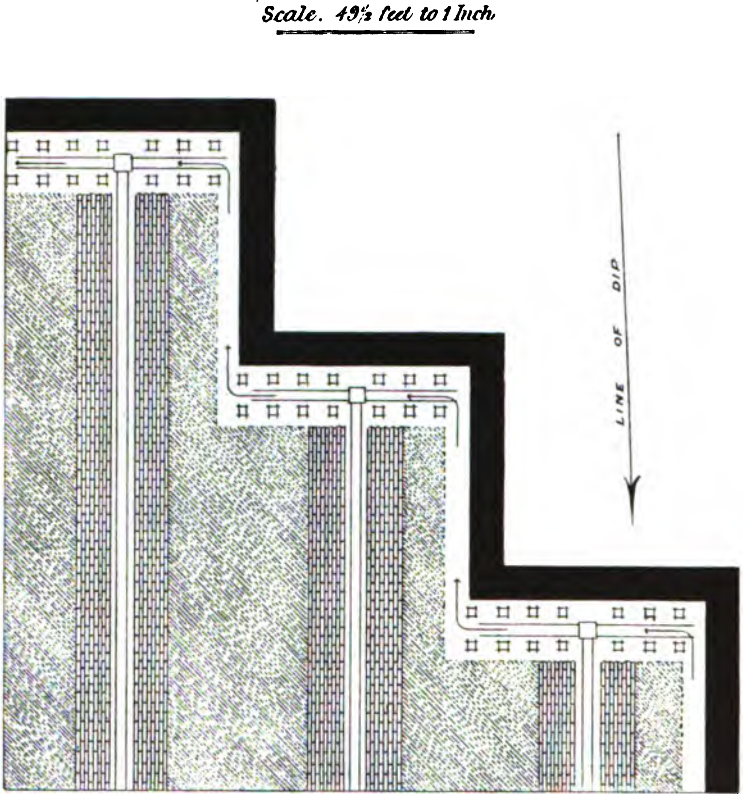


Fig. 243.—Clifton Hall Colliery, near Manchester. Plan of Three Lifts Advancing at the same time in the Trencherbone Coal Seam.

rock of the Carboniferous period (one found at Bacup being of granite), some of them are of great size, and there must have been some unknown means of transport, to distribute them over areas so widely separated as England and America. The whole of this interesting problem is as yet unsolved.

The method of working the Trencherbone Seam at the Clifton Hall Colliery is similar to that of the Five-Quarters last described. The levels are driven in precisely the same way, but the pillars are only 100 yards square, and instead of only one lift being taken off the pillar at a time, at least two, and sometimes three, are taken up-hill at the same time, as well as one or two going down-hill. The collier uses a windlass to draw the coals up from those places going down-hill. Fig. 243 shows three lifts proceeding at the same time to the rise. They are driven 20 yards wide, the face of each being kept from 20 to 28 yards in advance of the one behind it. The road is formed in the centre of each lift, and on either side of the road a pack 3 yards wide is built. A double row of chocks is kept next the face and parallel to it. The rails are laid in between the chocks, which are 5 feet apart as shown in Fig. 243. As the face advances sufficiently, another row of chocks is put in and the rear one withdrawn. Before proceeding to take down the rear row, props are set around each chock. The rubbish on which the chock was built is cleared away, and the chock knocked out at the bottom. The props giving security round the chock are then taken out, and the roof falls. The sprags shown in Fig. 242 are placed to prevent the coal riding over. They are 5 feet 6 inches long, and the lids over them are wedge-shaped.

The roads are dry and dusty. Generally the roof is good, and the main roads do not require much timbering; in places where the roof is bad props and sets of timber support it, cross-pieces and laggings being placed over the collars. The two props of a set are not of the same thickness, the one to the dip being 5 inches in diameter, and that to the rise 4 inches. The collar is 7 inches in diameter. It is said that an advantage arising from this system of working is the fact of keeping the weight on the face and off the roads.

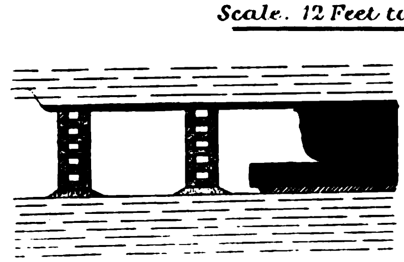
No blasting is done in the pillar working.

The Pendlebury Colliery adjoins the Clifton Hall, the two collieries belonging to the same company. It has been working since 1848, and at present works the Shuttle, Crumbouke, and Ram's Mines, by means of an upcast and a downcast shaft. The latter is 400 yards deep to the Shuttle and Crumbouke coal. At the pit bottom, a direct-acting pump raises the water in one lift to the surface.

The upcast, Furnace, or No. 2 Pit, is used to wind coal from the Ram's Mine. The workmen also ride up and down it, and as it is very hot, to guard against serious consequences through the cage being stopped by accident in the shaft, a signal is provided by which communication is made with the pit bottom from the cage. The onsetter on receiving such signal can open the separation doors and let the fresh air into the shaft.

A section of the Ram's Mine is shown in Fig. 244. The coal is 5 feet thick, besides the 4 inches of inferior coal which is left on in the working, but taken down in the roads. Over this inferior coal are 7 yards of blue metal, 3 feet of which are taken down to make height in the roads. Under the coal is blue metal. The coal is first holed in the middle at the parting, and the top coal then taken down, and kept 3 feet in advance of the bottom coal, which is blasted. A competent person is employed to fire the shots. In some places, the seam is holed in the 4 inches of daugh under the coal, but this does not run continuously through the seam. Where holed under the seam, sprags 18 inches long and 6 inches high are put in with wedges on the top.

The Ram's Seam is 200 yards above the Doe Seam, being worked at Pendlebury in the same way as the Trencherbone Seam is worked at Clifton Hall. An underground hauling engine, supplied with steam from boilers placed at the bottom of the upcast, hauls coal up a road driven 500 yards to the dip. The whole of this road is secured with brick arching. Levels are driven out of this dip road. At the far end of the dip, and for a short distance along the lowest level leading out of it, are some double-headed rails used as props, see Figs. 245 and 246. Two uprights 6 feet long support a curved crown 10 feet long.



	FT	INS
BLUE METAL		
INFERIOR COAL	·	4
COAL	3	5
PARTING	·	1
COAL	1	6
FIRECLAY	·	4
BLUE METAL		

Fig. 244.—Pendlebury Colliery, near Manchester. Section of Ram's Mine Coal Seam, and Timbering in it.

The uprights are set on sills, 12 ins. × 5 × 7. The crowns are curved slightly, having a versed sine of 6 inches. To keep them securely in place, short props 6 inches long and 6 inches thick are wedged in between the end of the rail and the side as shown in Fig. 245. These sets of rails are fixed every 4 feet along

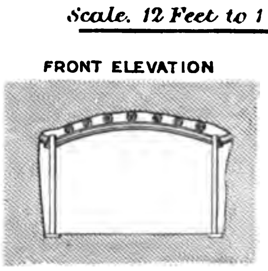


Fig. 245.

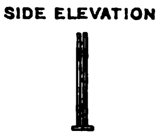


Fig. 246.

Pendlebury Colliery, near Manchester. Method of Securing Roof by Double-Headed Rails.

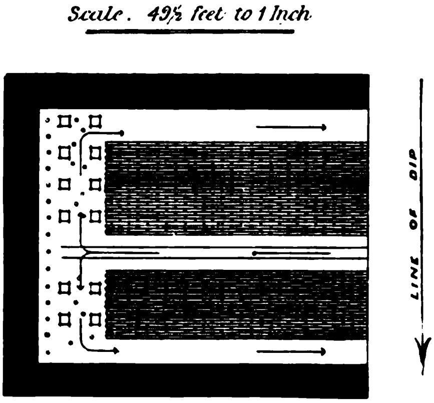


Fig. 247.—Pendlebury Colliery, near Manchester. Plan showing the Face of Level in the Ram's Mine.

the level for some distance, and rafters placed over the crowns. Little or no timber is used in the rest of the level. Owing to a fresh arrangement the boundary line was altered after this level had been driven and the coal worked back. It became necessary, through the boundary extension, to drive the level on again. At the point it had been standing, the roof was tender and required props and sets of timber to secure it at that point, but not elsewhere. The level is driven 20 yards wide, as shown in Fig. 247. On each side of the road, which is 9 feet wide, packs are built. The pack on the rise side of the road is 8 yards wide, and above it is an aircourse 9 feet wide extending from the pack to the coal. The pack on the low side of the road is 6 yards wide, and beyond the pack on the low side is an aircourse 6 feet wide. The face is protected by 2 rows of chocks, a row on the rise side consisting of 4 chocks and on the low side 2

The rows are 6 feet apart, and as the face advances sufficiently the rear row is drawn out and set up in front. When about to draw a chock, props are set up round it, and after the withdrawal of the chock, the props are taken out. Besides the double row of chocks, a row of props is set up next the coal, and others again between the chocks, as required. These props are 4 inches thick. Altogether there would be 12 chocks and about 30 props protecting the face, but the number of props varies. Occasionally chocks are set in the road between the rails and the packs, at intervals of 6 feet.

The stall-man sets the props and chocks, and they are taken out by the person who fires the shots. The stall-man receives 7*s.* 7*d.* for 3½ tons of coal, and 4*s.* 7*d.* for 3½ tons of small coal. The advantage claimed for this method of driving the levels is that it is safer in a fiery mine, as the face advances more slowly and gives the gas time to drain. The roof is fairly good, and the timber costs 2*d.* a ton.

At the Sovereign Pit, West Leigh, near Manchester, the West Leigh Seam is worked on the Longwall principle. It corresponds with the Wigan 9-feet, and the Trencherbone at Clifton Hall. The colliery consists of an upcast and a downcast shaft, each 376 yards deep. A pair of 36-inch cylinder high-pressure engines, fitted with Cornish valves and having a 6½-foot stroke, winds coals in the downcast pit. Flat wire ropes are used, the drums being 15 feet in diameter. At the upcast shaft is a Guibal fan, 40 feet in diameter and 15 feet broad, the pit being used solely for ventilation. A horizontal engine, having a 30-inch cylinder and 3-foot stroke, with Cornish valves, drives the fan. A duplicate engine also drives the fan, the two working 8 weeks alternately. The fan runs 35 revolutions a minute, at which speed it exhausts 200,000 cubic feet of air per minute with 1·6 inch water-gauge.

The West Leigh Five-feet Seam is very fiery, and, like the Barnsley Seam in Yorkshire, subject to sudden outbursts of fire-damp from the floor. The gas is emitted from a thin seam of coal 14 yards below. To lessen the danger arising from an overwhelming discharge of this gas, boreholes are put down from the workings, which allow the gas to pass up in smaller quantities, which may be regulated and controlled.

In 1881 there were two such boreholes, each having a tube with two branches on top of the borehole. A pressure-gauge on one of them indicated a pressure from the gas of 15 lbs. per square inch.

On removing the pressure-gauge and opening the cock the gas rushed with great velocity from the tube, accompanied by a roaring noise like that of steam issuing from a boiler under great pressure. After allowing the gas to escape for a few seconds, a test of the pressure-gauge showed a diminution of the pressure equal to 5 lbs. per square inch. On closing the cock, the pressure gradually increased again until it had attained its original pressure. During the hours the workmen are in the pit, the cocks on these boreholes are shut and the gas not allowed to escape, but when the men are all out, the fireman turns the cocks and the gas issues from them for 10 hours till he makes his morning examination, at which time he closes them for the day. There being no work on Sunday, the gas is allowed to issue all that day, and in consequence the pressure is much lessened on the Monday morning. The gas in a borehole, once left open during a strike of the men, became completely exhausted; but on work being resumed the gas again accumulated and issued from the borehole.

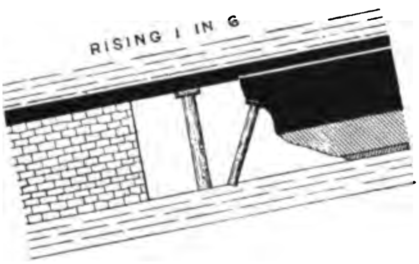
Much fire-damp is emitted from the roof also, and a fall is always succeeded by large quantities being given off. The gas is much more explosive than ordinary fire-damp, and it is almost impossible to get a cap on the lamp with it. Three per cent. of it mixed with air and charged with dust is explosive. Mueseler safety lamps are used by the workmen.

The strata dip at an inclination of 1 in 6.

Fig. 248 shows a section of the seam. There are 4 feet of coal, and above it 10 inches of inferior coal. This inferior coal is left on for a roof. Over this are 19 yards of laminated tender shale. If the coal is taken from under this shale in narrow work, the shale has a tendency to break down in gutters over such excavations, and therefore makes a bad roof; but when the coal is worked on the Longwall system, it makes a very fair roof. The thill is 2 feet 3 inches of shale, under it being 4 inches of fireclay; then 2 feet of fireclay and blue metal, below being 42 feet of blue metal with bands of coal; then the thin seam of coal before alluded to and to which the boreholes are pierced.

The 2 feet 3 inches of shale and the 4 inches of fireclay are taken out along with the coal. The 4 inches of fireclay are used to hole in, and the shale is taken down to give height while the operation proceeds. Sprags 5 feet 3 inches long and 6 inches in diameter are placed against the coal to prevent its riding over as the holing goes on, as shown in Fig. 248. The holing is carried 6 feet under the coal, after which the sprags are removed and the coal allowed to fall. The 2 feet 3 inches of shale are used to build the pack-walls at the sides of the roads, the whole space in the waste between roads being filled tightly and closely with rubbish.

Scale. 12 Feet to 1 Inch.



	FT	INS
LAMINATED SHALE	57	-
INFERIOR COAL	-	10
COAL	4	-
SHALE (BLAES)	2	3
FIRECLAY (DAUGH)	-	4
FIRECLAY & BLUE METAL	2	-

Fig. 248.—Sovereign Pit, West Leigh, near Manchester. West Leigh Five-Feet Seam, showing Props and Sprags used there.

The method of working is shown in Fig. 249. The levels are driven every 80 yards, and the stall-roads are from 12 to 20 yards apart. If there is too much rubbish yielded the roads are kept more than 12 yards apart, so as to allow more stowing room for the rubbish. The roads are not ripped. If a road sinks till it is too low, it is not repaired but abandoned, and a level road is driven through the gob cutting it off. This is found preferable to driving a road from the face, because the roof having settled back in the waste, a road afterwards made in it is easier maintained. The object of driving the stall-roads in the direction shown in Fig. 249, rather than at right angles to the levels, is to render the gradient easier for the tubs. One road is however always kept at right angles to the levels and used as a "jig brow." A chock is built in the acute angle of the building in the road near the face. The sprags are placed every 4 feet against the coal, the props a similar distance apart and a little back from the face between the sprags. At the road-head are two chocks placed 8 feet apart.

The roads are secured by props and sets of timber placed 3 feet apart, and chocks are placed at the corners of buildings where stall-roads are turned. Some of the sprags have an iron band 1¾ inches wide and ⅜ inch thick on either end, to prevent their splitting. These last much longer than those not so protected. A blacksmith puts rings on 30 props in a day.

Two colliers work in a 20-yard stall, each having a working face of 10 yards from the road. Each employs a drawer. The collier receives 1*s.* 4*d.* per ton for large coal and 1*s.* per ton for small. For this he holes the coal, sets the sprags, gobs the rubbish, and delivers his coal at the bottom of the jig brow. Of the coal got 95 per cent. is large.

A collier sends out an average of 7 tons of coal in a shift of 9 hours. Special contractors put in the packs and chocks. They are paid 6*d.* a ton on the coal sent out for building the packs, setting and drawing the props and chocks. The roads from the pit bottom inwards are under their charge, and they work on them both by day and night. Four-fifths of the work is, however, performed during the daytime. There are two pairs of contractors in the pit, one to each set of faces.

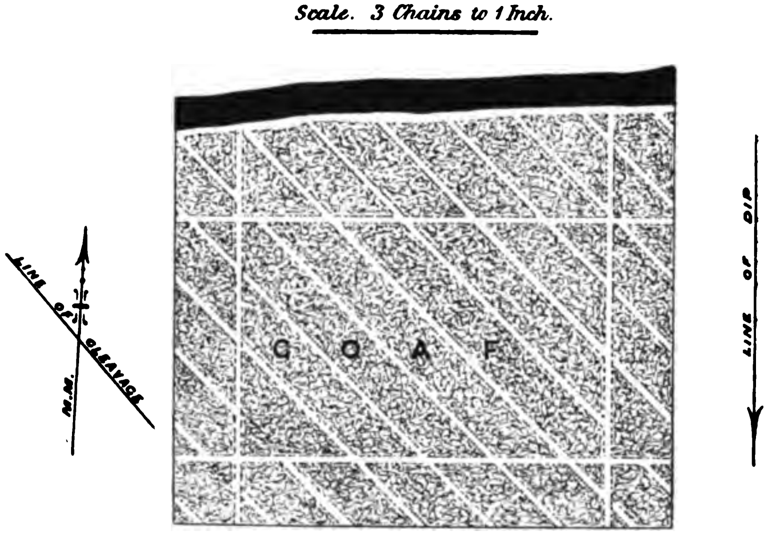


Fig. 249.—SOVEREIGN PIT, WEST LEIGH, NEAR MANCHESTER. PLAN SHOWING THE METHOD OF WORKING THE WEST LEIGH FIVE-FEET SEAM.

Each pair employs 18 or 19 men to look after a face 700 yards in length. There is a proportion of one of these contractor's men to 3 colliers. About 85 per cent. of the faces work on end. In the pit are 74 colliers, who send out 400 tons of coal a day. About 53 of these employ a drawer, and 21 draw their own coal. Their working day extends from 5 A.M. to 3.30 P.M. One overman directs operations in the pit, in which, besides the workmen, 7 horses are employed. The cost for timber at the face is ¼*d.* per ton.

The price paid for ordinary pit timber at the colliery is 9*s.* 8*d.* per 100 lineal feet, and 1*s.* 3½*d.* per cubic foot for larch. The roads are not dusty, but somewhat dry.

At the Radstock Collieries, Somersetshire, the thin seams of the Radstock group are worked on the Longwall system. The Somerset Coal Measures consist of two productive coal series, divided by a thick mass of almost unproductive hard grey sandstone and grit called Pennant. It is from 2,000 to 2,500 feet thick and separates the upper or Radstock from the lower or Kingswood and Bristol series of coal seams.

The upper series is locally subdivided into two groups, the upper group constituting the coalfield of Radstock and Camerton, and the lower the coal district of Farrington, Parkfield, and Coalpit Heath.

Similarly the lower series is subdivided into two groups, an upper one worked at Kingswood and Newbury, and a lower one worked at one time at Vobster and Twerton, and at present around Bristol.

The upper or Radstock group of the Coal Measures attains a thickness of about 1,000 feet and contains six workable seams varying from 1 foot to 2 feet 4 inches each in thickness. Those 1 foot thick are called *thin* at Radstock, and those two feet thick and upwards are called *thick*.

Between the Radstock and Farrington groups is a mass of red shale about 150 feet thick, making a very marked line of division. The Farrington group averages 750 feet thick, and contains six workable seams of coal.

The Pennant contains two or three workable seams.

In the lower series there are about twenty-six seams in its central area, decreasing to seven in the northern area. In the Nettlebridge valley, the coal measures of the lower series are from 3,000 to 3,500 feet thick, whilst in the Kingswood district they average 2,700 feet, and in the Bristol district 2,000 feet.

The total thickness of the coal measures is at least from 6,000 to 7,000 feet.

A peculiarity of this coalfield is that four-fifths of the whole of it is covered by newer formations, resting unconformably on the coal measures. These newer formations consist of New Red Sandstone, Lias and Oolite, and the coal seams worked under have their outcrops hidden by them. The New Red Sandstone, Lias and Oolite are nearly conformable to each other, and at Radstock are found almost flat.

At the base of the New Red Sandstone, there is a conglomerate, locally called "Millstone," which is believed to correspond with the Magnesian Limestone in the North of England. It consists of blue stones, which in size vary from half an inch to a foot thick, bedded in a red matrix. The stones have a rounded, water-worn appearance. This conglomerate is extremely hard and impervious to water. A pit in being sunk to the coal measures first passes through the overlying formations, and in the New Red Sandstone large quantities of water find their way into the shaft. These are pumped out until the conglomerate is reached, when the water is "tubbed back." The wedging crib is laid in the conglomerate and the metal tubbing run up through the water-bearing strata.

Whilst the newer formations are almost free from faults, the coal measures beneath present a marked contrast. Faults, varying in size from a few inches to a hundred fathoms, are found running through the district in every possible direction, the seams of coal lying in disconnected pieces between, sometimes level, at others at angles more or less inclined up to vertical, and even folded back with the floor of the seam uppermost.

An anticlinal axis at Kingswood divides the coal measures into two basins, the Northern or Gloucestershire being the smaller, and the Southern, constituting the Somersetshire, the larger basin.

The most remarkable of all the faults is the Overlap or Slide Fault of Radstock. This fault displaces the seams of coal vertically about 36 fathoms, and some of the pits sunk have proved the same seam twice, thus multiplying the coal seams in the shafts. In the manor of Radstock where the workings have been carried against this Overlap Fault and the facts carefully ascertained, the amount of folding over or overlapping of the seams varies from 120 yards on the highest of the 6 workable seams to 330 yards on the lowest.

The Radstock district yields a beautiful variety of fossil plants.

The method of working the six seams of the Radstock group is shown in Fig. 250. The bottom seam of the group lies more than 300 yards below the surface. No pillars are left either to support the shafts or the levels. Where the coal is taken out at the shafts, strong arching is thrown over to secure the road. Owing to the disturbed nature of the ground, levels are not carried from the shafts as in most districts, but cross-measure drifts or " branches " are continually being driven to win pieces of coal lying between faults.

Fig. 250 shows the method pursued where the seams are rising 1 in 3½, which inclination prevails over a certain area. The roads going to the full rise out of the bottom level are from 35 to 40 yards apart. One of them is made into an incline, the top or bottom being ripped for the purpose, and it is at first made 6 feet high. The other rise roads are only made "topple height," or about 4 feet. A level out of the incline on either side cuts off the old topples every 100 yards. The levels are made 6 feet high. When the face has advanced 100 yards, another incline is made above the lower one. The coal is taken out for 5 or 6 yards below the bottom level, as shown in Fig. 250.

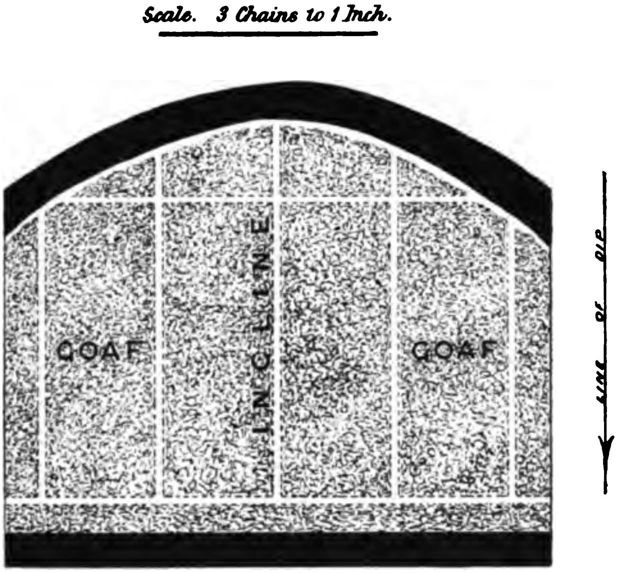


Fig. 250.—Longwall Method of Coal Working in the Upper Series of the Somersetshire Coalfield at Radstock.

Two colliers and a lad work together in a road, and their operations are confined to 20 yards on either side of it. Open lights are used throughout, no firedamp having ever been seen in the Radstock group of seams.

The men hole in the soft shale either above or below the coal, as may be most convenient, and use wedges afterwards to get the coal. Very little spragging under the seam is done. The coal, after it is holed under 3 feet for a considerable distance, being of a hard unyielding nature, requires wedging before it separates from the roof. The colliers set their own props, a double row being used along the face with lids or " traps " over them. There is no rule as to the distance apart of these props. They are 3 or 4 inches thick, and vary from 1 foot to 2 feet 6 inches long according to the thickness of the coal. The boy working with the two men hauls the coal out on a sledge or a board along the face to the road-head. The lad doing this work is called a "carting boy," and his sledge a "put." He brings the coal 20 yards from either side of the road, and in the case of the bottom level he is expected to haul 30 yards from the high side, and 10 yards from the low side of the road if required.

The " carting boy " fills the sledge, or, when he has large lumps, a board, and by means of a "guss" and chain hauls out his load. The "guss" is a rope band worn round his middle, the chain is suspended from it in front, and a "crook" of iron is used for readily hitching it to the sledge or board ring. The chain is passed between his legs, and he goes on his hands and knees in a "thick" seam, but serpent-like, clutching at the props with his hands and bearing against them with his feet to help him on in the thin seams.

From the road-head larger sledges, which are piled up much higher than they can be along the stall, are taken out to the top of the incline by "twin boys," or boys running in the "twinway."

A small pulley is secured at a road-post or at one of the sleepers, and a chain passed round it, with one end attached to the back part of the loaded sledge near the road-head, and the other to the front end of the sledge at the foot of the topple. Only one tramway is laid in the topple, and that not of iron rails but wooden "crease." It is made similar in shape to the tram rails for keen-edged wheels, but in being laid is reversed, the vertical part being placed outside the horizontal. The sledges are shod with iron, and this "crease" forms a groove in which they slide, without running on wheels down steep places. When the loaded sledge is started at the top of the topple it proceeds slowly, followed by the "twin boy" till it comes into collision with the empty. The latter is then turned on its side and dragged on a yard or two whilst in this position by the full sledge in its farther descent. When it is clear of the loaded sledge the twin boy turns it over fairly into the "crease" and then leaves it to follow the load down as the empty proceeds upwards. At the foot of the topple, the sledge is "carriaged." A carriage is a skeleton frame running on wheels, made for the reception of the sledge. The twin boy pushes the carriage sufficiently away from the foot of the topple for a large board or stage to be dropped over the rails (iron) running along the level road, and called a "twinway." The carriage is then brought against the board, their upper surfaces being nearly on a level when so placed, the sledge drawn from the foot of the topple on to the board, and from there placed on the carriage. Here it is run out to the incline, where it is taken off the carriage and let down the incline accompanied by and attached to two or three others without wheels. A double line of wooden rails is used in the incline, and to prevent the lumps of coal dropping off the puts which are piled high, a chain is passed over them. At the foot of the incline the coal is transferred into tubs and taken out to the shaft by horses.

The top shot down in the roads is used to build the pack-walls on either side. These are mere facings, and the rubbish made in holing is stowed in the gob. In a thin seam, the waste will not hold all the rubbish, and some of it is loaded and sent out to the shaft. The thick seams hold their own rubbish, and in cases where it is not sufficient to closely fill the waste, it is thrown into "tumps," that is, it is built in the waste with alternate spaces and rough packs. The collier receives a rate per ton, which varies according to the thickness of the seam. He sets all his own face timber, and throws his rubbish back. He also rips the road and builds the packs there, but for doing so he receives in addition to the tonnage rate on coal a yardage price in proportion to the height he makes the road. The carting boy is not paid by the collier, but he and the twin boy both receive a tonnage price, varying in the one case in proportion to the thickness of the seam, and in the other to the distance the coal is taken. Timbermen are appointed who receive a daily wage for withdrawing the props at the face and building the tumps. A large number of the props are not recovered. When they are, they are struck out by blows from a hammer, and not by a "dog" and chain. The roads crush very much after being made, and require frequent shooting down to maintain them at their height. Sets of timber are used in the main roads. They are placed without any kind of notching, and are of various sizes according to the difficulty of keeping open any section of the road.

No cogs are used at the face or in any of the main roads.

In those areas where the seams lie flat, or at an easy inclination, instead of the puts used in topples all the roads are made 6 feet high and the horse takes the tubs to the face. If a pit at Radstock raises 150 tons of coal in a day of 8 hours, it is considered good work, but it must be remembered that rubbish is hauled as well, and the natural disadvantages to contend with are very great. The shafts are small, usually not exceeding 8 feet in diameter.

At the Kingswood Collieries, near Bristol, the seams of the lower series are worked by Longwall.* The seams are steep and have a bad roof, making them expensive to work.

Fig. 251 is a plan showing the method of working the Great Seam or Vein at Kingswood, the seam being about 4 feet 6 inches thick here. From the bottom, or main level, "hatchens," as they are called, are carried to the rise. At the corners of these hatchens, packs 4 feet square are built as shown in Fig. 251 on either side of the road. These points are made filling places, plates being placed so that the tubs can be pushed in round the corner clear of the rails. The top is not shot down in these hatchens, but the coal is brought down in one of three ways, determined by several conditions such as, inclination of the hatchen, scarcity of boys, &c. First, where lads are not employed, the coal is let down in shoots, and the supply of coal through it into the tubs is regulated by a hopper. Secondly, by making the hatchen a self-acting incline, a small pulley being used round which the chain passes and the pulley is shifted as the face advances. Thirdly, by lads taking "sleds" up and down. In this case a chain is secured at the top of the hatchen and lies in the middle of the road, the lads using it as a hand-rail to pull by in ascending the road, and to act as a "drag" in bringing the loaded "sled" out.

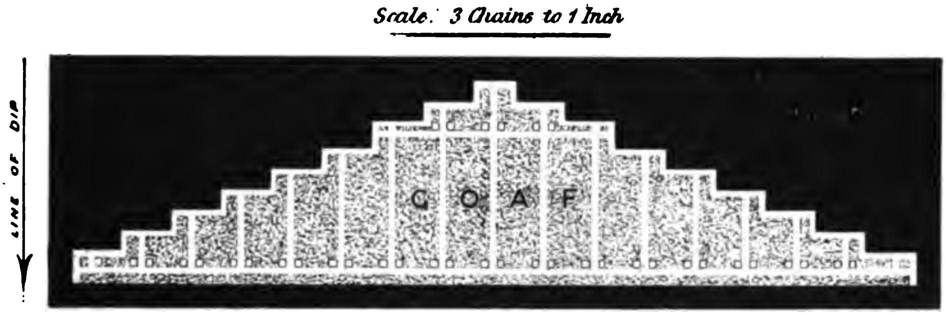


Fig. 251.—Longwall Method of Working the Great Vein of the Lower Series in the Somersetshire Coalfield at Kingswood, Bristol.

After being driven 44 yards, the hatchen intended for an incline is ripped and the rails are laid, a drum is fixed at the top, and the hatchen becomes a "Gug" or incline. A level road, one on each side from the top of this incline, cuts off the old hatchens from the lower level.

The levels are ripped right into the face, by one shift of rippers following two shifts of colliers. The face in a level road is carried 9 yards wide, 5 yards above and 4 yards below the road; the pack on the rise side is built 4 yards wide, and that on the deep side about 3 yards.

The packs built on each side of the hatchens vary from 3 to 4 yards wide. From the lower side of the main level a cross-cut is turned to the low side, and continued parallel with the main level as soon as there is room enough to give 5 yards of coal on the rise side. This cross-cut (not shown in Fig. 251) is ripped and packed precisely the same as the levels. Its chief use is as an intake for the air going inbye.

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* See Transactions, North of England Institute of Mining Engineers, vol. xxvii., pp. 96–97.

Two colliers work in one place. The air, after circulating round the face, passes into a higher district or by a cross-measure drift or branch into the overlying seam, called the Thorofare, on which, the roof being excellent, the returns are carried.

The Upper and Little Toad Veins are worked in a manner similar, differing chiefly in the length the hatchens are carried, which are 80 yards in the former, and 60 yards in the latter seam.

The roof of the Great Vein is so bad that the main roads are carried in the Little Toad Vein, and these roads are connected to the Great Vein workings by cross-measure drifts of 120 yards in length as occasion requires, the roads on the Great Vein being then abandoned and allowed to fall.

At the Allanshaw Colliery, Hamilton, Scotland, the Ell Coal Seam is worked by the Pillar and Stall or "Stoop and Room" method.

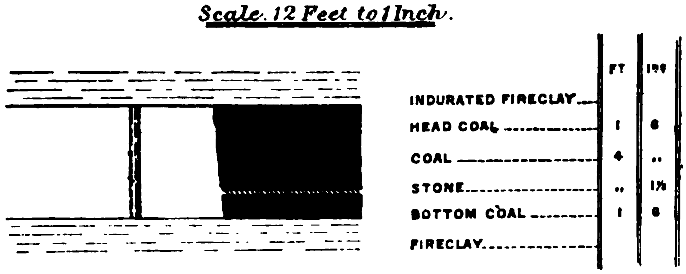

Fig. 252.—Allanshaw Colliery, Hamilton, Scotland. Section of the Ell Coal Seam.

The colliery has been working since about 1876, and consists of an upcast and a downcast shaft, each being 234 yards deep to the Ell Coal Seam. The shafts are circular in form, although the usual practice in Scotland is to sink rectangular pits, and then to divide them into two compartments, one for each cage; or if a set of pumps is to be placed in the shaft, it would have three compartments, the additional one being for the pumps. The Allanshaw pits are each 13 feet 6 inches in diameter. In 1881 the downcast solely was used for winding, although a pair of horizontal, high-pressure winding engines, with 26-inch cylinders and 5-foot stroke, had been placed at each shaft. In 1890 each shaft was used for winding. Single-decked cages, each carrying one tub or hutch which holds a ton, are used.

A Guibal fan 20 feet in diameter and 5 feet wide, exhausts the air at the upcast, and runs 40 revolutions per minute. At this speed it gives 40,000 cubic feet per minute with ·5-inch water-gauge. About 450 tons of coals are landed in a 10-hour day.

Fig. 252 shows a section of the Ell Coal Seam. There are 7 feet of coal, which parts badly from roof and thill or pavement. Over the coal are 4 feet of indurated fireclay, above which are 20 feet of rock. The pavement is composed of fireclay 6 feet thick.

Fig. 253 shows the method of working. The pillars are 20 yards wide by 30 yards long. The openings round them are 9 feet wide the short way and 12 feet wide the long way. The short way of the pillar faces the cleavage or cleat. The seam lies very flat, the dip being 1 in 20 to the North.

In the solid workings on a 12-foot wide place, a row of props or trees is set up, the props being 5 feet apart, on each side of the roadway. The collier puts up these props, and he is obliged to place them so that they are not more than 5 feet apart, whether the roof is good or bad. The trees or props are 4½ inches in diameter at the

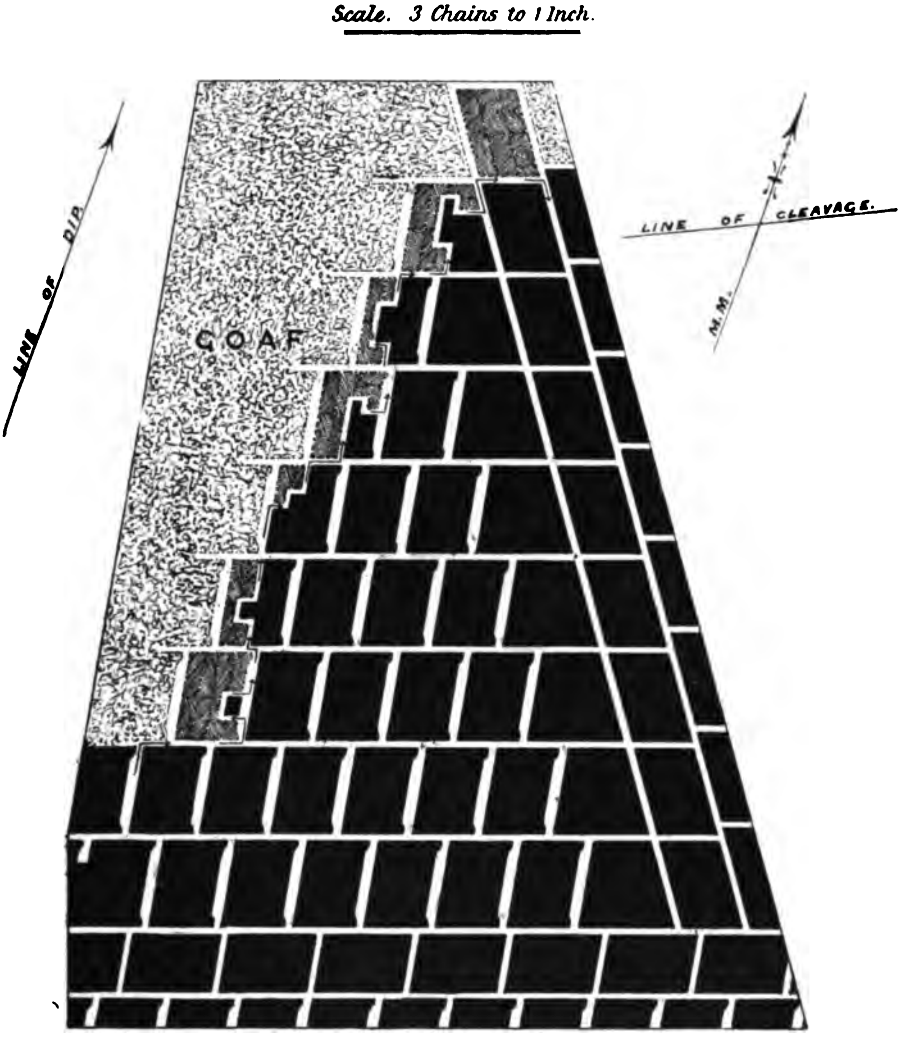

Fig. 253.—Allanshaw Colliery, Hamilton, Scotland. Plan showing Pillar and Stall. Method of Working the Ell Coal Seam.

thin end, and cost at the colliery 7*s.* 3*d.* for 100 lineal feet. Two men work together in one place. Their practice, after the coal is holed, is to shear down one side of the coal and put a shot in the other, blowing out the coal. Open lights are used in the solid workings, and the colliers fire their own shots. They receive a tonnage rate for large coal, and another for dross or small, which includes payment for the timbering.

In the broken mine, the pillars are worked off in lifts of 5 yards. Three men work together in a lift. They hew the coal, place it in the hutches, and set up all the timber. The colliers working off the pillars, or "stooping," as it is called in Scotland, use "Scotch gauze" lamps, and no blasting is allowed. The rails are laid in the road to about 3 feet from the coal, and a row of props set 3 feet apart, between the rails and the coal. On the waste side of the road are placed rows of props 14 inches apart, and about 3 feet between the rows. These trees are 4½ inches in diameter and have small lids, 6 inches square and 1 inch thick, over them. Three men draw the timber from a lift, one of whom must be the fireman. In compliance with the Special Rules, he draws the props in the afternoon when his examinations are completed. Two of the men strike out the trees by blows from a hammer, whilst the third removes them to a place of safety on a timber tub made for the purpose. In a lift containing 300 trees, it takes about 3 hours to draw the timber.

At the Cowdenbeath Collieries, in Fifeshire, the Dunfermline Splint Coal Seam is worked on the Longwall system.

The collieries have been in operation since about 1851, the work at present being carried on at three shafts, the Nos. 3, 7, and 8. The last two are downcasts, and the No. 3 an upcast. The upcast is ¼ of a mile distant from Nos. 7 and 8 which are within 120 yards of each other. All the pits are of a rectangular form; No. 7, being 17 feet × 10; No. 8, 14 feet × 6; and No. 3, 14 feet × 5½. The No. 3 Pit is 216 yards deep to the Dunfermline Splint coal, whilst No. 7 Pit is 270 yards deep to the same seam; and No. 8 Pit is 180 yards deep to the Lochgelly Splint and Parrot Seam.

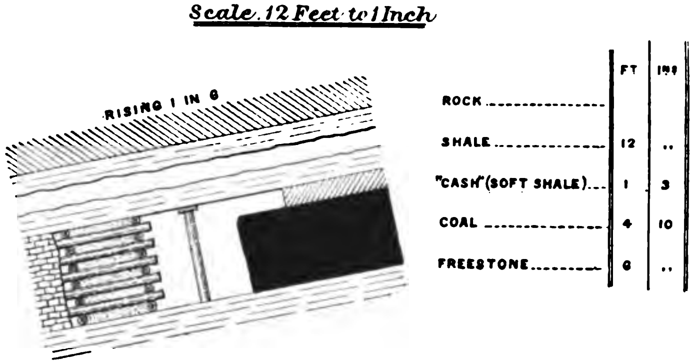

Fig. 254.—Cowdenbeath Collieries, Scotland. Section of the Dunfermline Splint Coal Seam.

A Guibal fan, 24 feet in diameter and 8 feet broad, is placed at the upcast shaft. It is driven at 60 revolutions per minute, and gives 50,000 cubic feet of air with 1·3 inch water-gauge.

Heavy pumping machinery is erected at both Nos. 7 and 8 Pits. The No. 7 Pit pumping-engine raises 800 gallons of water per minute, and that at No. 8 Pit raises 600 gallons. All the shafts are used to wind coal in, the total landings being about 800 tons a day. In 1881 the following seams were worked at the No. 7 Pit: the Dunfermline Splint 4 feet 6 inches thick; Five-feet, 4 feet 8 inches thick; Mynheer, 4 feet thick; and the Lochgelly Splint and Parrot Seam, 12 feet thick. The strata dip to the North at an inclination varying from 1 in 6 to 1 in 3.

Fig. 254 shows a section of the Dunfermline Splint Seam. There are 4 feet 10 inches of coal. Resting on it are 15 inches of "Cash," or soft shale, used by the colliers to hole in. Above this is the roof, composed of shale, which is 12 feet thick, and over this is rock. The pavement is freestone, about 6 feet thick. The shale over the seam makes a bad roof.

Scale. 3 Chains to 1 Inch.

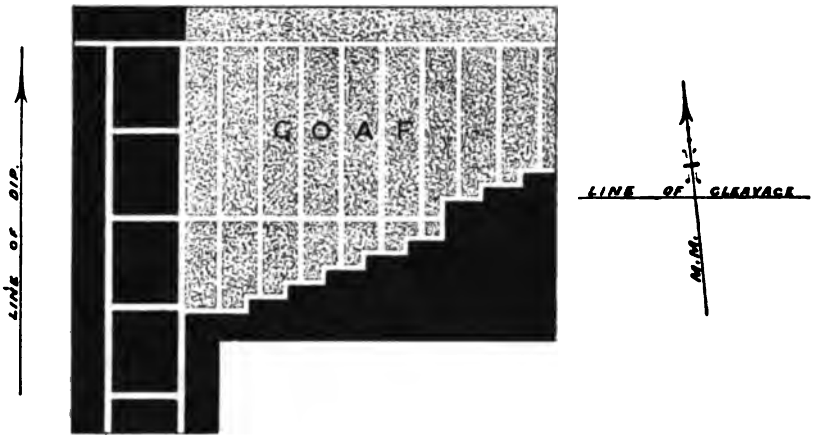

Fig. 255.—Cowdenbeath Collieries, East Scotland. Plan showing Longwall Working of the Dunfermline Splint Coal Seam.

Fig. 255 shows the method of working this seam. The levels are driven on end, or nearly parallel with the planes of cleavage, and the faces advance against the cleavage. The roads to the rise are driven at right angles to the levels, and they are turned every 14 yards from the level. Level roads cut off the rise roads every 60 yards.

Scale. 49½ feet to 1 Inch.

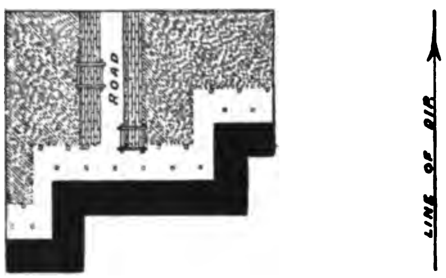

Fig. 256.—Cowdenbeath Collieries, East Scotland. Plan showing Face of Longwall Working in the Dunfermline Splint Coal Seam.

The men draw the coals from the faces to the wheel-braes or inclines, thence they reach the shaft by self-acting incline planes. Cut-chain inclines (fully described in Chapter IX. of this work) are used to let the coals down from the upper levels to the main wheel-braes.

It is found better in working the seam, to step the wall faces, keeping them 5 yards in advance of each other as shown in Figs. 255 and 256, on account of the heavy bad roof. The weight is thus confined to the limit of each wall, and the roof settles down more gradually than when a straight line of face is kept.

Two men work in a place, and besides hewing the coal, they put up all props, chocks and build the packs. They send out about 5 tons in a day of 8 hours, and receive 2*s*. a ton on large or round coal. Powder is not used in working this seam. Under the direction of the fireman, the colliers set their props where they think they are most required, there being no specified distance between them. The roof is so bad that withdrawing the props is dangerous and most of them are left in. The props are 3½ inches in diameter, the lids over them being of broken props. The "cash" obtained in holing is used to build the packs, but it makes poor buildings, and, to strengthen the packs, chocks, 6 feet square, made of props and filled inside with rubbish, are placed on the sides of the roads, as shown in Fig. 256. When the roads are finished and abandoned, these chocks are taken out.

The fireman visits each place three times during his shift, and although he sets up no timber himself, directs the colliers to do so where he thinks they are necessary. The main roads are secured by props and sets of timber, or gears placed at no specified distance apart, but at distances considered necessary. The roof being bad makes the cost of timbering high. The cost per ton on round coal is 5*d.*, the price of pit timber at the colliery being 5*s.* per 100 lineal feet.

In the working of THIN SEAMS the most advantageous system is the Longwall, and an arrangement of roads should be designed to suit the inclination of the seam or seams to be worked. The cost of working must of necessity be higher than in the case of thicker seams. Still, thin seams are successfully and remuneratively worked both in Great Britain and in other countries.

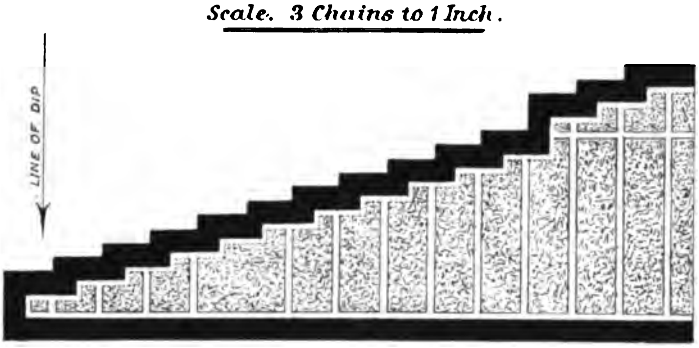

Fig. 257.—No. 1 MODE OF WORKING THIN COAL SEAMS IN NORTHERN FRANCE AND BELGIUM, ADOPTED WHERE THE INCLINATION DOES NOT EXCEED THIRTY DEGREES.

In Northern France and Belgium three different systems of working thin seams are employed.* No. 1 system, shown in Fig. 257, is an arrangement for seams in which the inclination does not exceed 30 degrees. Roads are carried to the full rise out of the chief level, the distance between these rise-roads being from 12 to 14 yards. The faces follow each other in step-like order, as shown on the drawing. At first the coal is taken down each of these rise-roads by means of a small self-acting inclined plane. When the faces have advanced a certain distance a new level is turned, which cuts off all the longer inclined planes, except one or two, which are retained as main roads. The lower level is at the same time advancing and opening out fresh ground, thus supplying more working faces. The main roads are ripped, and on the inclined planes tubs carrying 10 cwt. are used. A full description of the self-acting inclined plane in operation here is given in Chapter IX. of this work.

No. 2 system, shown in Fig. 258, is suitable for seams whose inclination ranges from 30 degrees to 60.

In this case, a succession of faces, in step-like order, are driven. These faces advance in the direction of the strike of the seam, not towards the rise, as in the No. 1 system. Where the seams are so highly inclined and the face advances to the rise, getting the coal is attended with a considerable amount of danger to the colliers, as large masses of coal fall out upon them during their work. A considerable loss arises, too, from the fact that the loosened coal lying on the floor slips downwards towards the waste by its own weight, and portions become lost amidst the rubbish.

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* See Transactions, North of England Institute of Mining Engineers, vol. xxvii., pp. 174-180.

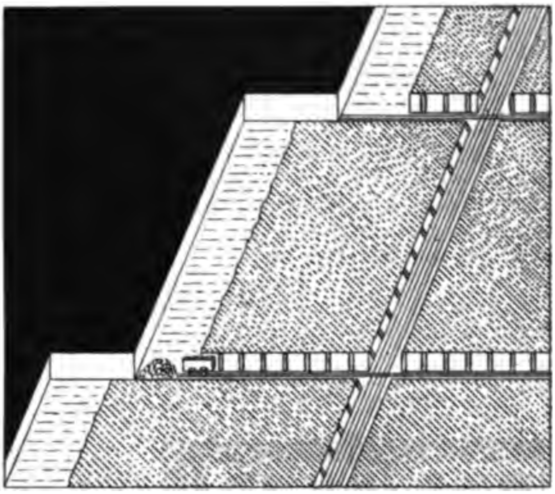

Fig. 258.—No. 2 Method of Working Thin Coal Seams in Northern France and Belgium, adopted where the Inclination ranges from Thirty to Sixty Degrees.

In the No. 2 system, however, the danger and loss referred to are avoided. A reference to the sketch shows that the tramway is laid along the main level, and any coal loosened by the collier, or accidentally falling, rolls along the face to the tramway, where its progress is arrested. It is afterwards filled into the tubs and sent out. Each face is 20 yards long (measured along the slope), and four men work in it. For their own comfort and convenience these men place pieces of board across the floor horizontally from prop to prop, by this means regulating and controlling the descent of the coal along the face.

No. 3 system is suitable for seams in which the inclination varies from 60 degrees upwards.

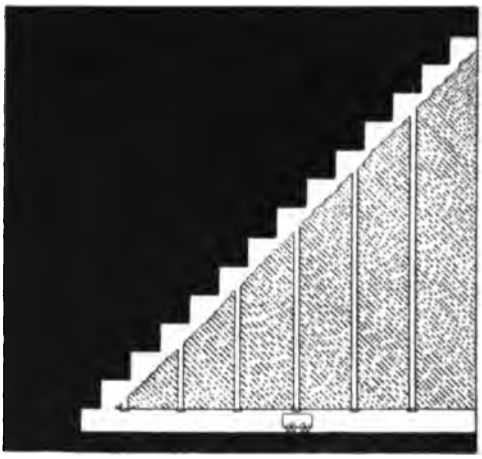

Fig. 259.—No. 3 Method of Working Thin Coal Seams in Northern France and Belgium, adopted where the Inclination varies from Sixty to Ninety Degrees.

The work here is stepped out into a series of faces one above the other. Each face is 6 feet high, and forms the working place of one man, as shown in Fig. 259. In this arrangement of steps the lower workmen are in advance of the higher, so that each man is protected against danger of anything falling on him from above, by the ledge of coal projecting 6 feet behind where he is working.

The refuse yielded in working the seam and the inferior coal are thrown back behind the colliers, to fill up the waste. Where the seam does not yield sufficient rubbish to pack the goaf, the workmen stand upon scaffolds, formed by placing planks horizontally across the props.

The whole range of work consists of about a dozen steps, and at the bottom there is a level road, to which the coals descend by means of small shafts or "chimneys." These chimneys are formed vertically in the goaf, by timbering with square frames and longitudinal poles, and are provided with a kind of sluice at the lower extremity. As the colliers hew their coal it is filled into the nearest chimney, to be afterwards withdrawn from below by the putters, who bring tubs under the chimneys, and for a time remove the sluice, thus allowing sufficient coal to rush into the tubs to fill them.

In many instances the seams of coal have been tilted over through more than a right angle, so that what are properly the floors of the natural beds have become the roofs.

As the floors are very friable, it is necessary to supplement the ordinary timber frames in the chimneys with poles, or lofting, and also with brushwood and wickerwood. This renders the cost of timbering high, sometimes as much as 1*s.* 7*d.* per ton. It is a matter of great difficulty to maintain the timbering in proper working order.

Excessively Thick Seams are often difficult to work, and require special methods according to the thickness of the seam and the inclination at which it lies.

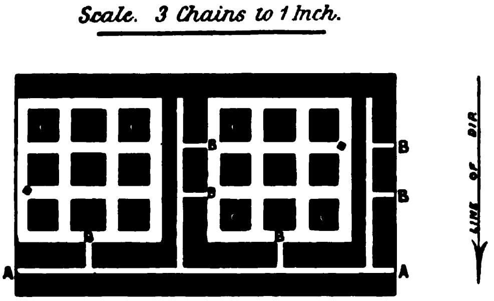

Fig. 260.—Plan showing South Staffordshire Square Work.
A A, Gate-road ; B, Bolt-holes ; C, Pillars.

In South Staffordshire, where the Dudley Thick or Ten-yard Coal Seam is found over a considerable area, a method of working called "Square Work" is sometimes adopted. The seam varies from 25 to 36 feet in thickness. The main road from the shaft, called a gate-road, is carried in the lower coals of the seam, and is ventilated by means of a separate air-head or drift of small dimensions, opened in the coal either to the side of or above the gate-road. From the gate-road (see Fig. 260) the main workings are opened, called sides of work, consisting of a square of 50 yards or more, with a narrow bolt-hole 8 yards long, connecting the gate-road and the sides of work. Stalls are then driven across and forward in the lower coals, leaving square pillars of about 10 yards in the side, to support the roof, and where the unsoundness of the coal or roof appears to require it, additional supports in men-of-war pillars of 3 or 4 yards square are left.

Sometimes these sides of work, instead of being 50 yards square, are larger, and in the form of a parallelogram, the longer side being parallel to the gate-road. To get the upper divisions of the seam, the men have to stand on the coal already cut, or on light scaffolding. As this process goes on, the rubbish and timber stop the communication to the first opened bolt-hole, and a narrow road is driven from the gate-road up the middle of the rib of coal between the two sides of work, and other bolt-holes cut through at intervals.

The pillars in the square-work are at the finish thinned till the roof falls, when the side is abandoned.

This seam being very liable to spontaneous combustion, dams are put into the bolt-holes to exclude the air from the small coal.

The coal seams of Dombrowa, in Poland, are of great thickness, and dip at a considerable angle. They form an extension of the seams of the Upper Silesian basin at Zabrze and Myslowitz. The coal is dry, very inflammable, and unfit for coking. The slack is of little use, and is liable to give rise to spontaneous combustion. A special method is required to get the coal as large as possible. At

Scale. 3 Chains to 1 Inch

SECTION

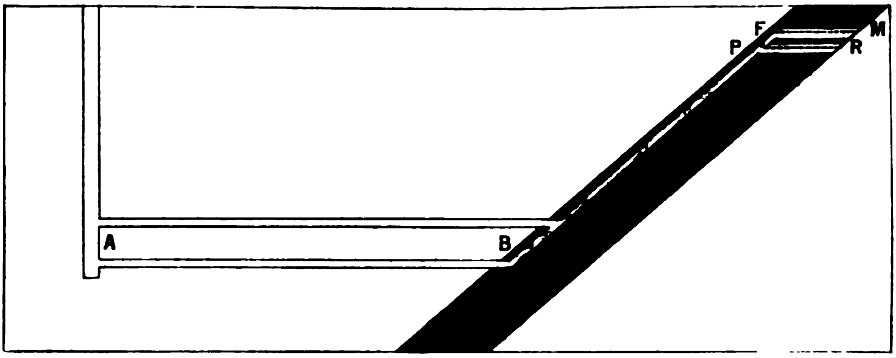

Fig. 261.

PLAN

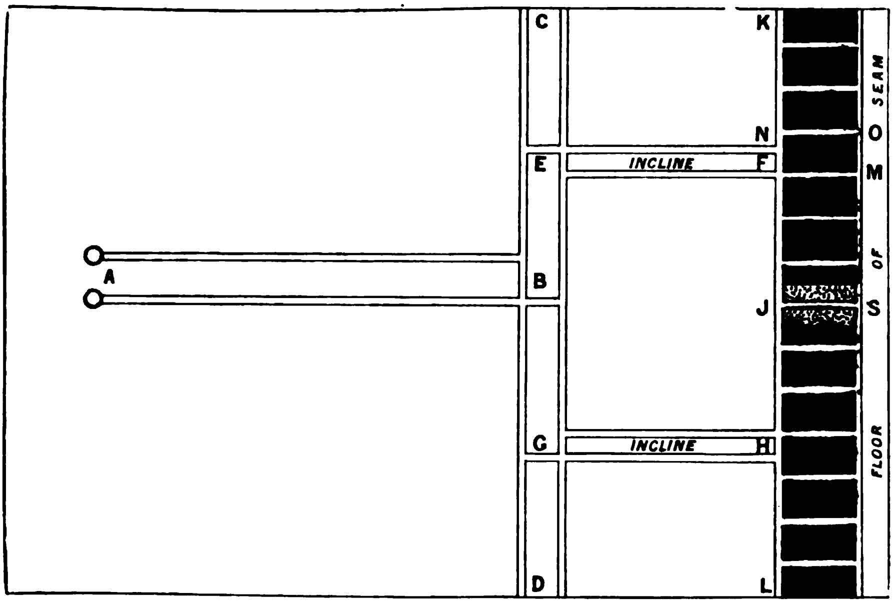

Fig 262

Section and Plan illustrating the Working of Highly Inclined Coal Seams of great thickness at Dombrowa, in Poland.

Nowo Labenzky and Chieshkowsky the seam is from 52 to 59 feet thick, the dip varying from 15° to 40°. The method adopted at these places will be understood by referring to the accompanying section, Fig. 261, and plan, Fig. 262.

The seam is shown dipping towards the shafts at an inclination of 40°. From the shafts, two water-level stone drifts, A B, on the section and plan, Figs. 261 and 262, are driven 33 feet apart vertically, which reach the coal seam at B. From this point the water levels are continued right and left along the course of the seam, keeping close to the line of intersection with the roof, and preserving their relative positions, B C and B D on plan. The higher one of these is used as a main drawing road, while the lower serves as a drainage and return airway. At intervals, varying from 260 to 360 feet, rise headings (B F on section and E F and G H on plan) are driven in pairs to the full rise of the seam and parallel to each other, about 23 or 26 feet apart, but of unequal section, the larger ones being afterwards converted into self-acting inclined planes, while the smaller are used for ventilation and as travelling roads, being deemed safer to traverse by the workmen than the main inclines. In driving these headings, a thickness of 3 feet of coal is left next the roof as shown on section, the roof over the coal seam being a black carbonaceous shale often very combustible, and even liable to spontaneous ignition.

From the top of the rise headings, F on section, and F and H on plan, a road is driven right and left on the roof side, F J, F K, and H J, H L on plan, and the coal is divided for working by driving headings across to the floor of the seam at regular intervals out of these roads, leaving pillars from 40 to 43 feet thick between the headings F M on section and F M, N O, &c., on plan. These headings crossing the coal seam are from 8 to 10 feet in height, the coal being worked in stages about 13 feet thick. A thickness of solid coal of at least 3 feet is left as a temporary roof to support the waste of the upper stage previously worked out, as shown between F M and P R on section. Working the pillars is commenced in the centre of the panel formed between any two of the inclined planes, after removing the roof coal by bringing back the pillars from S on plan in a direction parallel to the cross headings to J for a breadth and length of 20 feet and a height of 13 feet, the working face being protected by timber props. A part of the pillar working is shown at S J on plan. When the pillar has been worked off the timber props are withdrawn so as to allow the roof to come down, and if necessary, where the posts are tightly held, charges of dynamite are used to liberate them. Before removing the timber, however, a layer of small wood and props is placed on the floor, forming a kind of cushion, which distributes the pressure of the fallen rock and waste over the surface of the next lower stage. In this way the floor, which in due course becomes the roof of the lower stage, is sufficiently coherent to be kept up by the use of timber props.

It is usual to work three stages at different levels at one time; while the pillars are being removed in the uppermost, the headings are driven in the one below, and the roads corresponding to F J, F K of the first stage on plan, are being driven in the third out of the rise headings E F, and G H.

By this method the coal, being always supported on a firm bed, is not liable to crushing, and thus one of the principal causes of spontaneous combustion is avoided. In case of fire, the pillars, being only 40 feet wide, give facilities for extinguishing the fire. In order to obtain the full benefit of the method the excavations formed by the first four or five stages at the top should be closely packed, so as to have a protecting cushion for the workings against falls of rock from sides of the old excavations.

About 65 per cent. of the total quantity raised by this method is large.

In the coal mines of Upper Silesia, seams of coal varying from 20 to 30 feet thick are worked.*

The principal seam at the Königin Luise Mines near the village of Zabrze, near the frontier of Poland, is called the Schuckmann Flötz, and varies in thickness from 19 to 26 feet. At the Von Krug shaft it is 23 feet thick, is in one solid bed without bands or partings, and being of anthracitic quality is unsuitable for coke-making.

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* See Transactions, North of England Institute of Mining Engineers, vol. xxvii., pp. 190–193.

The method of working this exceptionally thick seam is shown in Fig. 263.

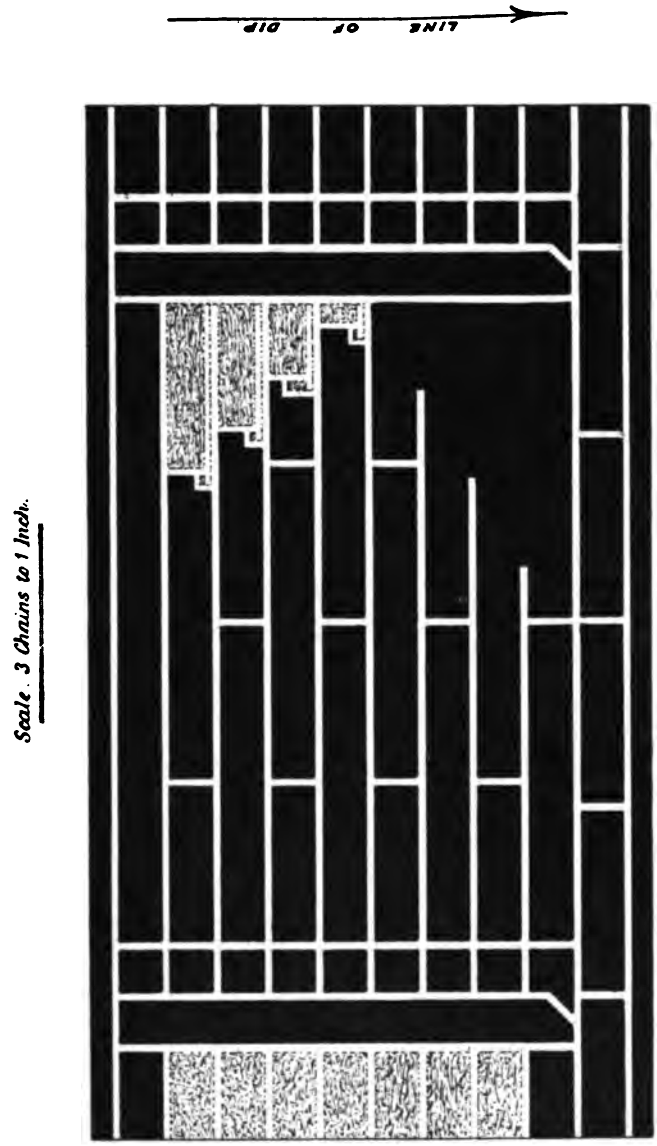


Fig. 263.—Plan showing Method of Working the Schuckmann Flötz Coal Seam at the Königin Luise Mines in Upper Silesia

It is first divided into blocks of 220 yards by 14, with the necessary cross-holings for ventilation, by driving a preliminary network of roads in the lower portion of the coal, viz., next the floor. These roads are driven 8 feet square in cross-section, and the pillars or blocks formed lie with their longer sides parallel to the strike of the seam. Two or three men, assisted by boys, work together in each of these roads. The 220 yards by 14 block of pillars is next removed by taking a series of lifts 5 yards wide at a time, across the pillar, towards the rise, and taking all the coal to its full height of 23 feet. In order to get the coal, the colliers stand at different elevations on ladders, and the roof is supported by means of long timbering reaching from the floor to the roof, lofting being placed over the uprights. When a lift has advanced right across the pillar and is holed into the gallery beyond, it is finished, and as much of the timber as possible withdrawn. Two lines of these trees are, however, always left, one along the lower side of the lift, designed to protect workings coming from the dip against an influx of stones from the goaf, and another line up the near side of the lift, intended to give protection to the next succeeding lift.

Besides these precautions, it is frequently necessary to leave ribs of coal several yards in thickness, so that a considerable loss attends the working of this thick bed, besides which the coal left behind is liable to spontaneous combustion. Five men work together in a lift, assisted by three boys who separate the large and small coal by means of rakes, and fill it into the tubs, which they afterwards take out. It is the duty of the colliers to set the timber.

At the Mines at Kladno, in Bohemia, a thick seam of coal from 23 to 37 feet in thickness is worked.

The method of working, in the first instance, is similar to that pursued in the mines of Upper Silesia. Here the preliminary network of roads driven in the lower portion of the seam, divides it into a series of large blocks, having their longer side parallel to the strike, the dimensions of each of which are from 60 to 100 yards long by 10 yards wide. The seam contains two bands of stone, one of which is 3 feet and the other 6 feet above the floor. The roads in the preliminary work are driven of the height of the upper band, and are, therefore, 6 feet high. A more intricate, but also more economical process of working the pillars, is practised than that adopted in Upper Silesia.

First, a stall, about 4 yards wide, is driven across the pillar, under the upper band, as shown in Figs. 264 and 265. The stall is not commenced at the extremity of the pillar, but a safety rib, of 2 or 3 yards in width, is left towards the goaf. This stall is rendered secure by timbering with stout 6-foot posts. After advancing across the pillar it holes into the road beyond, which is the goaf. The colliers then by a backward movement extract the safety rib, and also bring down the whole of the 20 or 30 feet of coal overhead. This is effectually done by removing the props with which the stall was timbered. When it is desired to bring down a quantity of top coal, firemen come to the stall and place cartridges of dynamite near to the top of several of the posts. The safety fuses attached to the cartridges are then lighted, and the men retire to a safe place. When the dynamite explodes, the posts are blown out, and the coal in the bed over falls with a tremendous crash. The colliers work in companies of two, and they have little more to do than fill the coal thus brought down: they can get from 15 to 20 tons a day.

Thick seams may also have the ordinary methods of Longwall and Post and Stall applied to them. The seam would be taken out in a number of different lifts formed of layers parallel to the stratification, the top being worked first and the roof allowed to subside on the coal. A 12-foot seam might be divided into two 6-foot lifts, an 18-foot seam into three, and so on. After an interval succeeding the working of the first lift the second might be proceeded with, but some time should be allowed to elapse between.

The author is indebted to the Council of the Mining Institute of Scotland for permission to extract from their Committee's Report "On Methods of Working and Timbering at the Face." The descriptions given of Celynen, Risca, and the

Scale 49½ Feet to 1 Inch

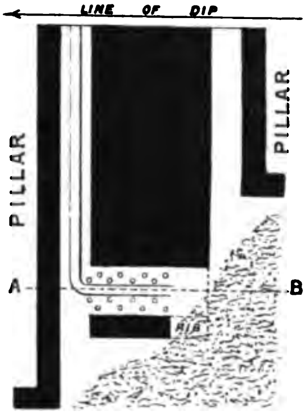


Fig. 264.

SECTION THROUGH A.B

Scale 12 Feet to 1 Inch

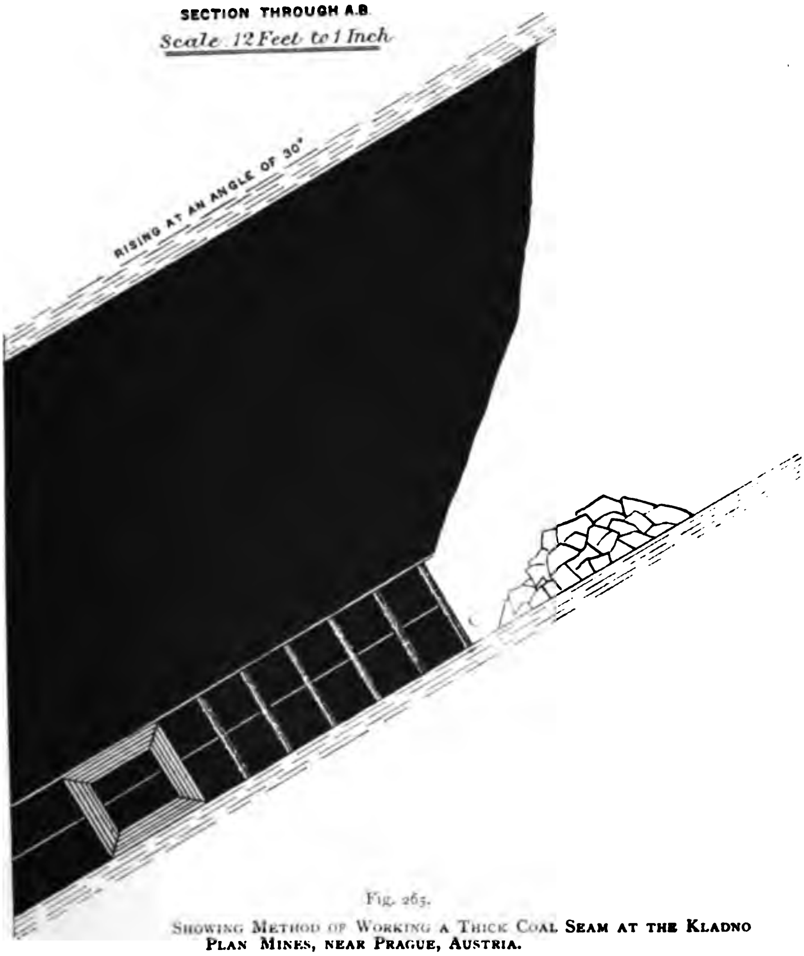


Fig. 265.

Showing Method of Working a Thick Coal Seam at the Kladno Plan Mines, near Prague, Austria.

Ocean Collieries, South Wales; the present method of working Lundhill, and Kiveton Park, Yorkshire; High Park, Notts; Wearmouth and Silksworth, Durham; Florence and Great Fenton, North Staffordshire; Cannock and Rugely, South Staffordshire; Pemberton, Lancashire; Clifton Hall, Pendlebury, Sovereign, near Manchester; Allanshaw, Hamilton, Scotland; and Cowdenbeath, Fifeshire, are extracted from their Transactions as given in vol. III. pp. 51 to 124.

This chapter will now be brought to a conclusion with a few questions and answers having reference to the working of coal seams, &c.

> *Question* 45. In a seam of coal worked by Longwall the coal is of a soft nature—whether would you work on plane or end? Why? How would you keep the weight off the face?

It is generally advisable to work coal of a soft nature on end, because it carries the weight better and is produced in better-shaped and larger blocks, while the roof is not so liable to fall, owing to the weight crossing the natural planes. To keep the weight off the face the back timber should be drawn.

> *Question* 46. Suppose you have a seam dipping to the South, and you meet with a downthrow fault on the East level, which way would you turn the road to get to the coal?

The road should be turned towards the North or in the direction of the rise, as may be seen by the sketch, Fig. 266.

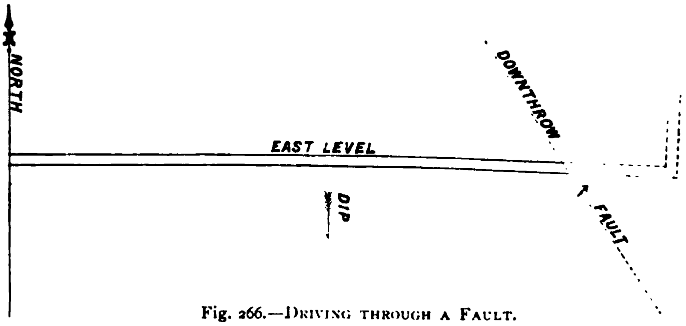


Fig. 266.—Driving through a Fault.

> *Question* 47. Suppose you have a seam dipping to the South and you meet with an upthrow fault on the East level, which way would you turn to get to the coal?

The road should be turned towards the South or in the direction of the dip, as may be seen by the sketch, Fig. 267.

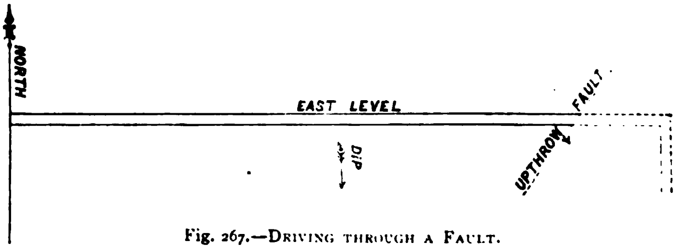


Fig. 267.—Driving through a Fault.

Question 48. In a road where the inclination is 1 in 7, a fault of 10 yards down is met with; what is the shortest length of level stone drift necessary to gain the coal?

The length of the stone drift would be 10 yards downthrow fault × 7 the dip of seam = 70 yards, and it may be driven either from the point where the fault was met with or from a point "out bye," as shown in sketch, Fig. 268.

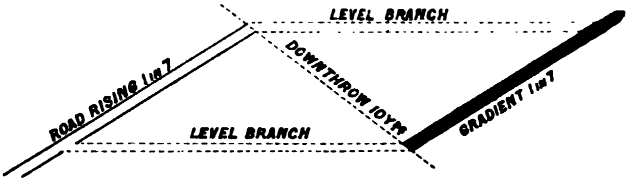


Fig. 268.—Driving through a Fault.

Question 49. In driving the West levels in a seam of coal dipping to the South at the rate of 1 in 6, an upthrow fault of 15 yards is met with, what would you do to recover the coal? If driven through, what length would the driving be?

I should drive in the direction of the dip or towards the South until cutting the coal, keeping the driving as truly water level as the West levels already driven were. The length of such driving would be 15 yards upthrow × 6, the dip of the seam, = 90 yards. See sketch, Fig. 269.

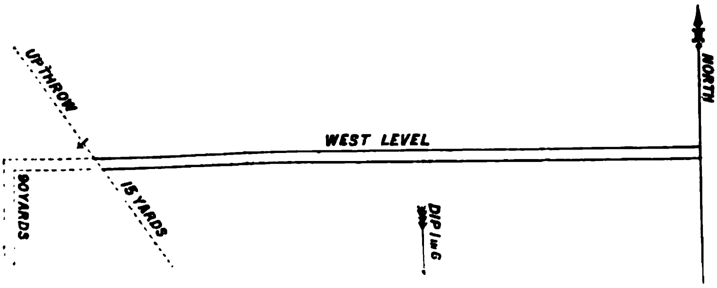


Fig. 269.—Driving through a Fault.

CHAPTER IX.

SURFACE RAILWAYS AND UNDERGROUND CONVEYANCE.

Making a Surface Railway or Tramway—Sleepers, Rails, Points, and Crossings—Rails used on Main and Minor Roads Underground—Steel Sleepers—Portable Tramways—Construction of Tubs—Inclination suitable for Main Levels and Self-acting Inclines—Arrangement of Rails and Friction Rollers on Inclines—Blocks at the Bank-head—Reumaux's Safety Catch for Inclined Planes—Mortier's Safety Catch—Method of Fixing the Incline Drum—The "Seizer" placed at the Foot of Inclined Planes—Accidents on Self-acting Inclines—Arrangement for Stopping Runaway Tubs down Inclines—Cut-chain Haulage on Inclines—Counterbalance Tram for Inclines—Different Systems of Engine Planes—Direct Haulage—Tail-rope—No. 1 Endless Rope—No. 2 Endless Rope—Endless Chain—Signals on Engine Planes—Size of Hauling Engines necessary—Compressed Air as a Motive Power—Compound Hauling Engines—Determination of Gradient for Horse Road and Inclined Plane—Calculating the Friction of Tubs.

THE important question of the disposal of the produce at a colliery receives consideration before the shafts are sunk, although the exact route of the tramway or railway may not then be decided on. The necessities of small collieries may be met by a cart trade or by a neighbouring canal. Where the output is large, and especially if the coal raised is for steaming purposes, a ship canal or a railway to carry the coal to a seaport, or to the staithes alongside a navigable river, becomes an absolute necessity for inland collieries.

The district may be served by a public railway more or less distant from the colliery, and the nature of the connection between the two will depend upon the physical features of the country, and the power to take land for any particular route. The public railways are themselves affected by the physical conditions of the country. For instance, there are many railways in the mountainous districts of Wales with steep gradients and sharp curves which increase the difficulties of working, and reduce the speed of trains as compared with railways having easy gradients and curves of large radius. In some countries where high altitudes have to be reached by the railways, it is only practicable to construct them with a reduced gauge and a rack rail or by a rope haulage on account of their extremely steep gradients. Sometimes, owing to difficulties presented to railway construction, minerals are transported a considerable distance by means of aërial wire ropeways.

Some collieries have surface tramways leading to screens at the sidings adjoining the railway, the tramway being worked by an endless chain, or a rope, or a small locomotive. It is always better for the surface tramway to have the same gauge as that adopted in the underground workings, in order to avoid the breaking of the coal in transfer to a different size of tram and the extra labour involved by a change of gauge. In some instances, where the gauge of a surface tramway is greater than the underground, breakage of the coal is avoided by using trolleys or platforms on wheels on the surface, each platform being made to receive two or more trams of coal, kept securely in place while being conveyed from the colliery to the screens.

Whether it is better to make a connection to the public railway by tram or rail, and the best method of working it, can only be determined after a careful survey and levelling of available routes taken in conjunction with the nature of the ground, whether mossy, soft and yielding, or rocky, level, or mountainous.

If the railway can be economically constructed without sharp curves and with easy gradients, there are great advantages in bringing the railway trucks by locomotives to the colliery sidings. If the branch line is worked by a railway company's locomotives the gradients and curves must be such as to comply with the regulations of that particular company, as must also the weight per running yard of the rails used, heights under bridges, &c. The public railway may be much above the colliery, and the connecting branch be so steep as to render a rope haulage desirable. Again, it may be considerably below, allowing the use of a self-acting incline all or part of the way, according to the configuration of the land and the expenditure incurred in the making. If power is obtained to cross public roads on the level and a locomotive is used, gates worked by a watchman at the crossing will be necessary, but if the crossing is worked by horses, protecting gates may not be required. Bridges over or under roads and over streams must be constructed to give the desired width for a single, double, or more lines of rails as may be desired. Soft yielding ground, deep cuttings through hard rocks, bridges, viaducts, tunnels, &c.. add to the cost of railway making and must therefore be avoided as far as possible.

The plan of a proposed railway shows its course throughout, the beginning, ending, and radius of all curves, the position and dimensions of all sidings, and the area of ground required to form the cuttings and embankments necessary for the proper width of the railway. If there are bridges these are shown, as also are the authorized road or stream diversions.

The section which accompanies the plan shows the surface line along the route in one colour, while in another the railwav formation level is marked, and the height above datum line to both are lettered so that the heights of all cuttings and the depths of all embankments are clearly given. In the calculations made it is customary as far as possible to let the amount of cuttings equal what will just form the embankments, or, if not, provision must be made for obtaining or disposing of extra material. Each gradient and the length over which it extends is plainly marked throughout the section extending from the beginning to the termination of the railway.

The course and levels of the proposed railway are staked out on the ground and the construction is usually undertaken by a contractor.

After the cuttings and embankments have been made to their specified width, the latter usually subside a little, and the levels must be tested before the ballast is laid.

The ballast consists of gravel, broken stone or slag, or any other substance that is pervious enough to ensure perfect drainage. It must be hard enough not to be crushed or powdered by the passage of trains and to resist the action of frost. The size to which the stones or slag are broken is such that no stone shall exceed six ounces in weight. A uniform depth of about twelve inches of ballast is laid over the formation level of the railway, the sleepers are laid on this and the rails on the sleepers. The spaces between sleepers and at the sides are afterwards filled in with ballast, six or nine inches of upper ballasting being thus laid. The object of the ballast is to keep the road dry and hold the rails firmly in place. If water were allowed to remain between the rails they would sink. Longitudinal and cross drains are made where required within the railway ballasting, the latter communicating with drains leading into the side ditches. Side drains must invariably be formed in the excavations, and if found necessary in the embankments, they are formed at intervals down the slopes.

The rails are laid upon wooden or metallic sleepers, which are placed transversely on the under ballast and beaten firmly into place with wooden mauls. When the upper ballast is laid, if properly packed about the sides and ends of the sleepers, it forms a compact mass which offers strong resistance to displacement. If. however, the packing escapes at the ends, a lateral displacement takes place which alters the gauge at that point. If the sleepers are not well bedded but rest on an uneven surface of irregular stones, the weight of the rolling stock will cause them to sink. Soft sleepers allow of canting under lateral pressure from compression of the timber. Wooden sleepers for an ordinary 4-foot 8½-inch gauge are 9 feet or 9 feet 6 inches in length, 10 inches in width, and 5 inches in depth. They are usually of oak or larch, and when creosoted weigh about 150 lbs. each.

The rails are made of malleable iron or steel, the latter making much the stronger rail, but there are some disadvantages in its use. Different sections are used as shown at C, D, and G in Fig. 270, the first being a Vignoles, the second a double-headed, and the last a bull-headed rail. The Vignoles rail is flat-bottomed and has a solid head, a neck and a flanged base. The double-headed rail is reversible, and when first designed it was thought there would be economy in its use, because when one side was worn out it could be turned. In practice, however, it is found that unless these rails are frequently reversed, their lower heads become bruised where they rest on the chairs, and are thus rendered unfit for use. On first-class railways this form of rail has now given way to the bull-headed section, in which the upper head is larger than the lower. The radius of the tops of these rails varies from 12 to 40 inches, and the depth of the double-headed or bull-headed, which are used for the best railway work, are 4½ to 5 inches in depth.

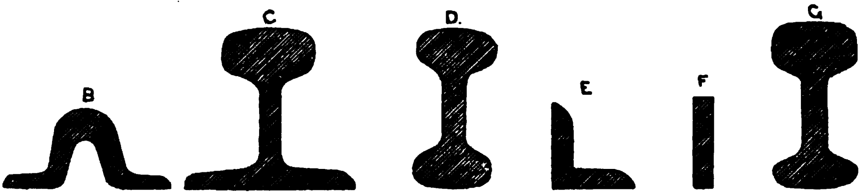


Fig. 270.—Sections of Rails.

The Vignoles rail, although not used on the main railways of this country, is much used about collieries, above and below ground, because it is not necessary to use chairs to secure it to the sleepers.

The weight of rail in lbs. per yard is such as is calculated to be most serviceable for the weight of trains passing over it. Lighter sections are frequently used on branch lines than on the main, where they often weigh from 80 to 90 lbs. per yard. They are rolled in different lengths, and when being laid require great care at the joints, which must be secured by either joint chairs, or fish-plates.

The chairs for securing the rails to wooden sleepers are of cast iron. They hold the rails firmly in place, and distribute the weight by their increased bearing surface. Each chair for a permanent surface railway weighs from about 22 to 45 lbs., lighter ones being used for lighter rails. There are two chairs to each sleeper, each being secured by two iron spikes and two iron screws, and wooden keys are driven between the rail and the chair outside the rail. When the key is driven home it wedges the rail firmly in its place. Holes for the spikes, which are chisel-shaped at the bottom, are first bored quite through the sleepers. The holes bored are only half the diameter of the spikes. Sometimes three iron screws or bolts are used to secure a chair to the sleeper, one being on the inside and two on the outside of the rail, and the keys are placed inside the rails. Bolts require constant attention because they are apt to work loose. The bottom of the rail rests on the chair as shown in Fig. 271, with the metal at one side of the chair in contact with that of the rail below the head and with a space left on the other side between the chair and the rail into which the wooden key is driven to steady the whole. Chairs of wrought-iron or steel composed of one piece are used with metal sleepers, to which they are riveted or bolted. Fig. 271 shows Keeling's patent cast-iron chair secured to a metallic sleeper. In it the arrangement of bolts and rivets is designed to take the strain off the rivets.

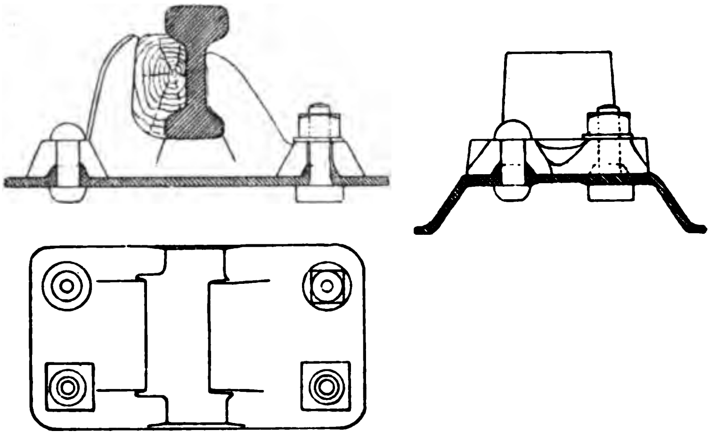


Fig. 271.—Keeling's Patent Cast-iron Chair.

Where fish-plates are used the joint of the rails is suspended, a sleeper being placed on each side at such a distance apart as just allows of the fish-plates being bolted to the rails without touching the chairs (see Fig. 272). The sleepers are evenly spaced throughout the railway, except at rail-joints, points and crossings. For an ordinary surface railway the pair of fish-plates used to join the ends of two rails weighs about 20 lbs. They are short strips of steel rolled, to correspond at the top and bottom, with the head and base of the rail against which they fit nicely as shown in Fig. 273. Sometimes, instead of straight sides the fish-plates are made with the middle portion slightly bent outwards, as shown in Fig. 274. They are secured by four round or square-headed bolts and four square or hexagonal nuts to the rails, the two ends of one fish-plate being placed equi-distant from the rail-joint. The holes punched through the rail for the bolts are oval to allow for expansion, and those in one of the plates are either oval or square in

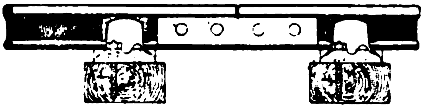


Fig. 272.—Fish-plates, Wooden Sleepers, and Cast-iron Chairs, at a Joint between two Lengths of Bull-headed Rails.

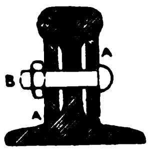


Fig. 273.—Fish-plates.

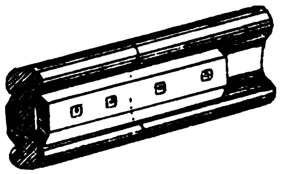


Fig. 274.—Fish-plates.

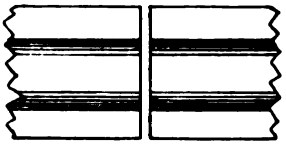


Fig. 275.—Space left between two Lengths of Rails.

order to prevent the bolts from turning round when the nuts are being screwed up. The bolt is made a suitable shape near the head to suit the oval or square hole punched in the one plate, but are round elsewhere.

Joint-chairs are sometimes used instead of fish-plates, the sleeper then being placed under the joint, and a special chair, heavier than those used between the joints, is secured to the sleeper. The abutting ends of the two rails then rest in the joint chair. All the main railways use fish-plates now, which are considered the simplest form of joint, effectually preventing any springing up of the rail.

Whether joint-chairs or fish-plates are used there is always a slight space left between the ends of rails, varying with the temperature when laid, so as to allow for expansion, see Figs. 272 and 275. The two parallel lines of rail forming one way are laid on the same level throughout straight portions of the railway. At a curve, however, the outer rail is always slightly raised above the inner, to balance the centrifugal force of a train running round it. The difference in the level of the two rails is determined by the sharpness of the curve and the intended velocity of the trains.

When one railway intersects another, crossings are used. If the intersection is at right-angles, as shown in Fig. 276, the construction of crossings is very simple.

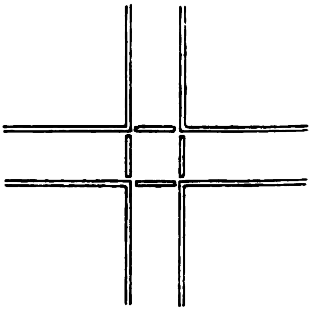


Fig. 276.—Crossing of Two Lines of Rails at Right-angles.

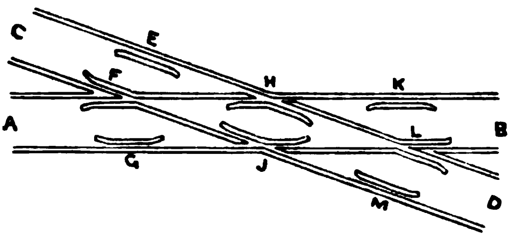


Fig. 277.—Crossing of Rails, not at Right-angles.

Here it is only necessary to have sufficient space or break in the continuity of rails to allow of the passage of the wheel flanges. There are no means for trains to pass from one line of rails to the other, and this kind of crossing is but seldom required. With any other than a right-angle in the intersection the crossings have to be constructed with wing rails, opposite which are placed check rails to guide the wheel flanges in the right direction. In Fig. 277, a line of rails, A B, is shown crossing another, C D, at a slight angle, and the arrangement of crossings necessary may be seen in the illustration. They are of two kinds. When the external angle of intersection is acute, the crossing is single as at F and L. These are commonly called " V " crossings. Where the angle of intersection is obtuse as at H and J, it forms what are called "diamond" crossings. These can only occur in pairs.

Check rails E and G are provided for the " V " crossing at F, and K and M for that at L. The object of each of these check rails is to ensure one wheel of a passing train being kept on the rail if the opposite one should have a tendency to take the wrong side of the point of the crossing. It will be noticed that in whichever direction a train may be passing one of the check rails will be sure to guide the wheels aright.

Fig. 278 shows the intersection of two railways where the crossing is from the opposite side, but the arrangement of crossings and check rails is the same as that already described.

At the junction of roads points or switches are used (see Figs. 289 and 290). They consist of a pair of tapered movable tongues connected by a cross bar, and for locomotive power lines are connected by rods with a lever in the signal box. This lever is worked by the signalman, the signals and points acting together, so that the points cannot be set for the train to pass in a certain direction without the signals corresponding. Special arrangements of points are made on self-acting inclines, which are moved by the trucks in their descent. At the colliery sidings worked by horses, the points are sometimes balanced by a counterpoise which ensures their always remaining in one position except at the passing outward of a truck, the flange of the wheel then pressing them over and the weight directly afterwards bringing them back.

The rails used for main underground roads are either the double-headed (D, Fig. 270) or the Vignoles rail (C, Fig. 270), the same as those used for surface railways and already described, but lighter sections are used underground. Bridge rails (B, Fig. 270) are also used to some extent on main roads and largely in the workings. They are the lightest obtainable form adapted for permanent roads, and although their section is unsuitable for transverse sleepers, they are so laid when used in the mine where longitudinal sleepers are objectionable. The sleepers are placed from 2 to 3 feet apart, and the rails secured to them by means of tram nails driven through holes in the flanges into the sleepers. A somewhat broader sleeper is placed under the joints than where spaced between, and the abutting ends of rails are secured by nails. The tubs nearly all run on flanged wheels, but in exceptional instances keen-edged tram-wheels are used, the angle rails (E, Fig. 270) being the only means of keeping such wheels on the rails. Where rails have to be laid along the faces or roads in the workings which are only expected to last a short time, sometimes a flat iron edge rail (F, Fig. 270), about 2 inches deep by $\frac{3}{4}$ of an inch wide is used. Notches are cut in the wooden sleepers and the rails are secured in place by means of wooden wedges driven in the notch next the rail. This rail is light to handle, is easily laid, and can be quickly taken up to be relaid elsewhere, but it is only suited for temporary purposes and for light tubs or trams carrying 6 or 7 cwts.

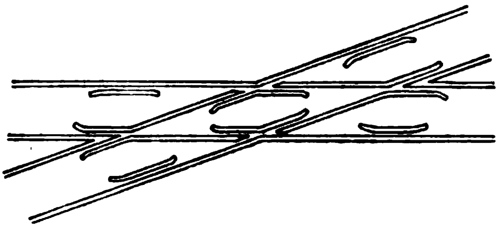


Fig. 278.—Crossing of Rails, not at Right-angles.

Engine-planes, self-acting inclines, and other main roads are laid with heavier and longer rails than the branch roads, which are only used for limited periods in the workings. The heavier rails weigh from 24 to 40 lbs. per yard in from 12 to 18-foot lengths, laid on sleepers, generally of larch from 3 to 4 feet long by from 5 inches × $2\frac{1}{2}$, to 6 inches × 4. Within the last few years steel sleepers have been introduced very largely, especially on main engine roads, and have proved a great advantage. The rails in the workings are usually laid in 3 or 6-foot lengths, and weigh 16 lbs. per yard for small trams, increasing in weight where heavier trams are used.

The gauge of the tramway is determined by the thickness of the seams, the size of the winding shaft, the quantity of coal to be raised, and the kind of hauling it is intended to adopt. In small collieries with thin seams small trams are necessary, but in large ones trams of greater carrying capacity are used, especially where there are long engine-planes with easy gradients. The carrying capacity of trams ranges from about 6 to 40 cwt., and the gauge of the tramroads is usually from 18 to 30 inches. In a few collieries the gauge is 3 feet.

Small trams are much more easily moved about than the heavier ones, and this allows of sheet iron being laid at the junctions of roads in the workings, whereas for the heavier trams there must be turn-tables or else no break in the continuity of the rails. Small trams may be taken from the end of engine-planes or horse-roads to the working faces by putters without horses if the gradients of the roads are suitable. Large trams usually have to be taken to the face by horses, and the hauliers are men. Boy drivers are often employed where the trams are light, as they are able to lift the empties on the rails when necessary. The large trams used in South Wales are adopted to meet the circumstances prevailing, and tend to economy in the haulage. The reduction of the length of the working day by legislation or otherwise tends to the use of larger trams. If a large quantity must be raised in a short time in deep shafts requiring great ventilating capacity, both necessities are most simply met by the use of large shafts and large trams. The deeper the shaft and the shorter the day for winding, the greater is the need for large trams for a given output. There are, however, limits to the size of tram which can be used, owing to the difficulty of maintaining underground roads beyond a certain width, and this difficulty increases with the depth and the inclination of the seams. The larger the tram in use the greater is the necessity for the underground tramroads to be carefully and skilfully laid, approximating, as far as circumstances will allow, to the construction of a surface railway. If the crush is great, the displacement of timbers and rails is increased in proportion to the width of the road. Usually, seams in which roads may be made a width to take a double line of rails have only a slight inclination, and are worked by means of comparatively shallow shafts. A serious drawback to the use of large trams is the greater liability to accidents as compared with that of small ones.

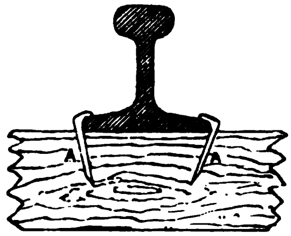


Fig. 279.—Dog Spikes holding the Rail.

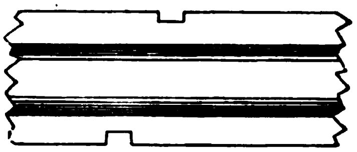


Fig. 280.—Notched Rail for Dog Spikes.

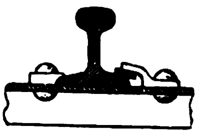


Fig. 281.—Livesey's Clips.

If the surface tramway is to be worked by an ordinary light locomotive, or the underground by an electrical or a compressed air locomotive, the gauge of the way should not be less than 2 feet; if by a stationary engine the gauge should not be less than 18 inches. Any gauge above 3 feet is really a railway and beyond what is ever required in the mine.

The double-headed rail is not much used in mines because it cannot be kept securely in place without the use of chairs and fish-plates. There is the same objection to the introduction underground of the bull-headed type.

The flat-bottomed rail is the one most frequently used on engine-planes and the chief horse-roads underground. It is secured to wooden sleepers placed from 2 to 3 feet apart by dog spikes, or to steel sleepers by wooden keys, bolts or clips. The spikes are not driven vertically into the sleeper but slightly slanting, one on each side of the rail, as shown at A A in Fig. 279, so that their heads may have proper contact with the sloping surface at the base of the rail. The spikes are sometimes made with a projection at the back, or with ears at the side, to facilitate their withdrawal. When driven as described they do not prevent the rail from moving in the direction of its length, to avoid which the rail is sometimes notched as shown in Fig. 280. The spikes are driven at the notches and prevent the rail from moving in the same way as nails driven through nail-holes in the flanges of bridge-rails do. The weight of the rails used will vary according to the capacity of the tubs, and also with the distance the sleepers are placed apart. If the floor is soft and springy in any section of the road, the sleepers are placed closer together, and in extremely bad cases it may be necessary to lay longitudinal timbers under the transverse sleepers to distribute the weight. The rails laid on these main roads are usually in 4-yard, 5-yard, or 6-yard lengths, longer rails being inconvenient to handle in the roadways. Fish-plates should be used with flat-bottomed rails for permanent work.

In countries where wood is abundant, it makes the most economical sleeper, but steel sleepers seem to be coming more and more into use owing to their greater durability and reduced cost of maintenance. Different forms of metallic sleepers have been used in making surface railways in other countries, and to a limited extent in this. Their use is sure to increase as the forest supplies dwindle, and increased attention has of late years been given to their design.

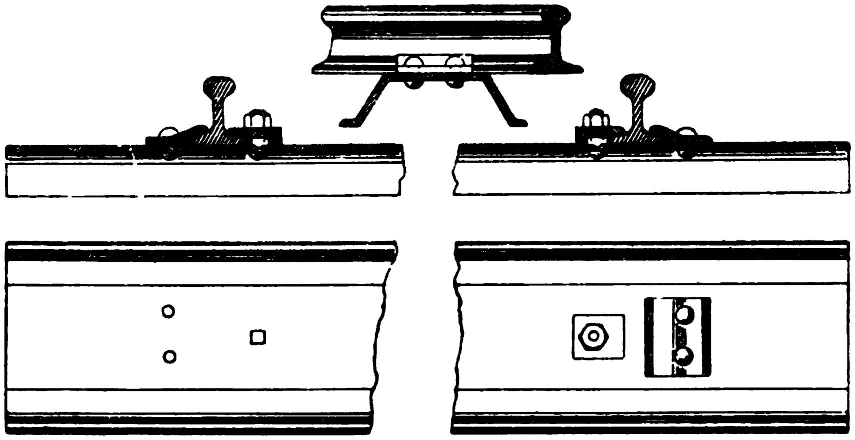


Fig. 282.—Fast and Loose Clips made at Tredegar.

The flat-bottomed rail is usually secured to steel sleepers by some form of clip, and the joints of the rails for permanent roads are usually fished.

Fig. 281 shows Livesey's patent clips, from which it will be seen that there are two clips, the larger one of which is corrugated on the inside to correspond with the corrugations on the loose key. Both clips are riveted to the sleeper at the steel-works, so that there is very little fitting of parts on the railway. The rail is simply canted in or out of place at the fixed clips, after which the key is driven to secure the rail. The corrugations prevent its working loose, but in order to give it greater holding power it is sometimes made longer, with the end turned downward so as to fill the space between the rail and the clip.

Fig. 282 is an illustration of fast and loose clips for a light steel sleeper, which has been largely made at Tredegar for gauges from 2 feet and upwards. The outside clip is riveted on at the works so that the gauge is always maintained by this, the loose clip on the other side being secured by a bolt.

Some clips are independent of bolts and rivets, and where they do not introduce a number of loose parts have much to recommend them. Colquhoun's clips, Fig. 283, are made on this principle. When the clip is passed through the two holes in one side of the sleeper and brought into its proper position with the inner end of the clasp gripping the base of the rail, a wooden key is driven; the rail, sleeper and clip are thus all fastened firmly together. It has been extensively used on engine-planes at collieries.

In Wood's clips the sleeper and rail are gripped together without the use of bolts or rivets. They are formed in one piece, and are so shaped that the driving of an ordinary wooden key is made the means of security.

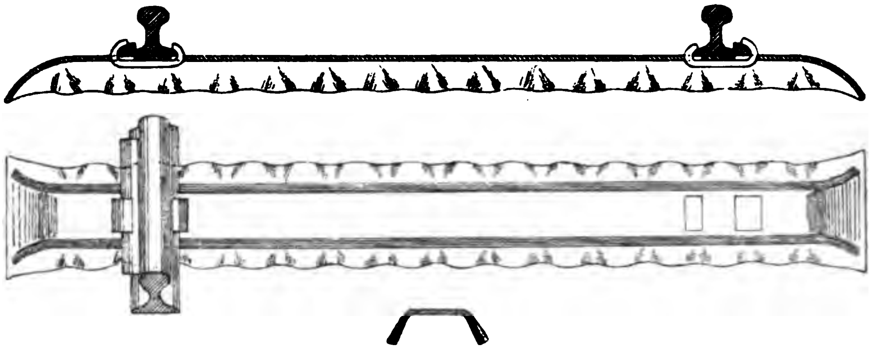


Fig. 283.—Colquhoun's Clips.

In Raffarel's patent sleeper there are two cast-iron or steel brackets, and the driving of a single key ingeniously fastens together the rail, sleeper, and brackets. The brackets are in contact with the rail at the flange and under the head. This sleeper for a 2′ 9″ gauge weighs 16 lbs. or, inclusive of fastenings, 22 lbs.

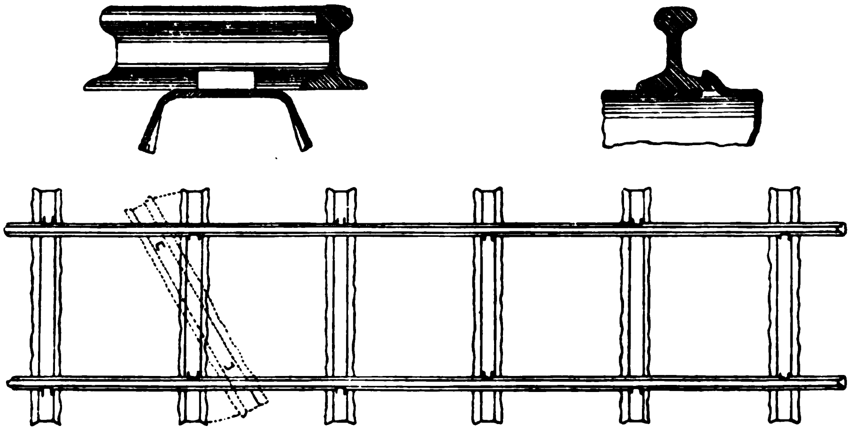


Fig. 284.—Metallic Sleepers for an Underground Roadway.

A sleeper which has been much used in the South Wales collieries is shown in Fig. 284. There are no loose parts, but a clip is formed at one end of each sleeper. The position of the clips is different on adjacent sleepers, but the same on alternate ones. The sleepers formed with inside clips are first laid obliquely across the road in the position of the dotted lines in the illustration, and when turned until they are at right angles with the rails they form locking sleepers, and together with the sleepers having outside clips, hold the rail securely in place.

A sleeper with the chairs or gripping jaws formed out of its own material, and without loose pieces of attachment, is the simplest and most economical. Fig. 285 shows a steel sleeper of this kind which has been used in the collieries of South Wales. The rail may be canted into place between the rigid upturned metal, after which the wooden key is driven to hold it firmly in place. This sleeper allows of great rapidity in laying a tramway.

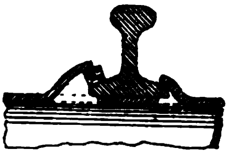


Fig. 285.—Steel Sleeper with Jaws formed on it for an Underground Tramway.

The drawback to the use of steel sleepers having smooth surfaces for horse-roads is the increased liability to horses slipping on them, but a better foot-hold for the horses is obtained if a corrugated form of sleeper is used.

Messrs. J. & F. Howard, of Bedford, who are makers of portable railway plant for mining and other purposes, have adopted the form of sleeper shown in Figs. 286 and 287. The sleeper is made of plate steel corrugated and flanged in order to give the greatest strength with the least weight of material. In constructing the sleeper the chairs for the rails are formed by pressure on the crown of the sleeper, none of the metal being removed or cut away, but, on the contrary, the parts of the chairs upon which the rails are intended to rest are increased in thickness and durability. The chairs are formed on the sleepers by special hydraulic machinery with a precision of position in all sleepers that ensures accuracy of gauge being afterwards obtained. The rails rest upon the whole width of the sleeper, thus having a great bearing surface. No bolts, dogs, rivets, or fish-plates are required, the only fastening being the simple metal key shown in the drawing, and this is serrated on one side to fit into the cheeks of the chair when driven home.

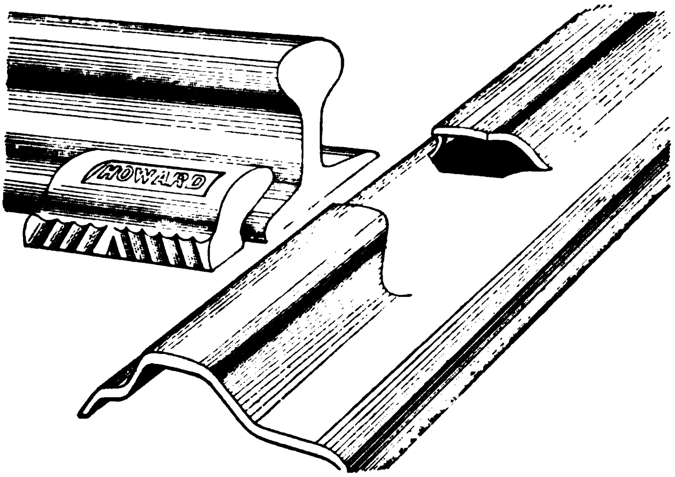


Fig. 286.—Steel Sleeper, Key and Rail.

For underground work a modified form of sleeper can be used, threaded on to the rails, and the use of keys is thus dispensed with.

The method of laying the line is shown in Fig. 287. The jointing sleepers are formed in the same manner as the single sleepers, except that two corrugations are rolled on one plate; the chairs or rail seats are pressed out of each corrugation in such a manner that the rail ends abut and are held in each seating respectively by its own serrated key. The keys are so arranged that either of them may be taken out without disturbing the others. This method of securing and joining the lengths of rails enables the laying down, and also the taking up, of the road for removal and re-layal to be effected with the greatest ease and expedition.

The corrugated form of sleeper allows the ballast to settle down very firmly under it, and the rail seats being below the crown of the sleeper, prevent any tendency for it to shift sideways on curves. For lines worked by horses this is the strongest form of sleeper, as the corrugation prevents any bending likely to be caused by the constant treading of the animals employed, and also gives a good foothold, thus preventing the slipping which so frequently occurs on flat-crowned sleepers.

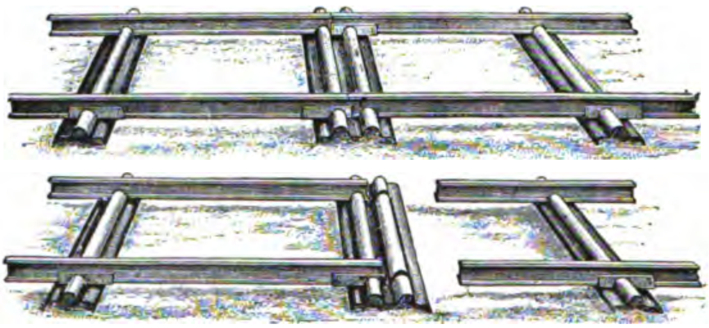


Fig 287.—Portable Railway, showing Jointing Sleepers.

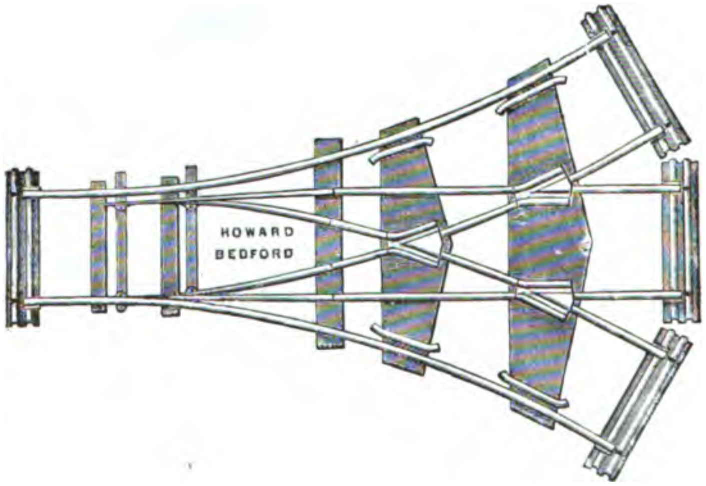


Fig. 288.—Three-way Points.

The advantages of a portable line of rails of this form may be summed up as follows:—

Simplicity, strength, and durability.

Accuracy of gauge, which cannot vary, as the seats for the rails are formed by special machinery.

The gauge does not depend on the skill of the workman, and will be preserved as long as the line lasts.

No bolts, rivets, or spikes, with their many disadvantages, are required to fasten the rails to the sleepers.

A metal *safety key* fastens the rail to the steel sleeper, so that it cannot shake loose.

The rails may be laid down, removed, and relaid with despatch and without the aid of skilled labour.

The rails and sleepers not being riveted together, stow into small space, whereby cost of freight is reduced to a minimum.

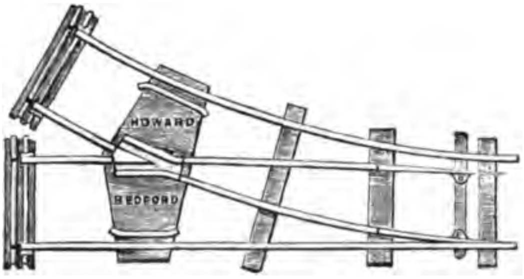


Fig. 289.—Right-hand Points.

In order to prevent corrosion, the patent sleepers, when at a certain temperature, are coated with an anti-oxidation compound, while for handling and shipment there is no fear of any damage to the sleeper or alteration in shape, nor are there any projections to break off. They are shipped ready for laying, a great consideration where skilled labour is scarce. The expansion of the metal of sleepers equally exposed to high temperature is not sufficient to destroy the accuracy of the gauge. The rails are supplied in lengths up to 21 feet, and of various sections according to the nature of the traffic.

If the line is to be used for light locomotives, the jointing sleeper is not used, the ordinary suspended fish-joint being preferable for main lines. The usual gauges for which these sleepers are made are 20, 24, 30 and 36 inches.

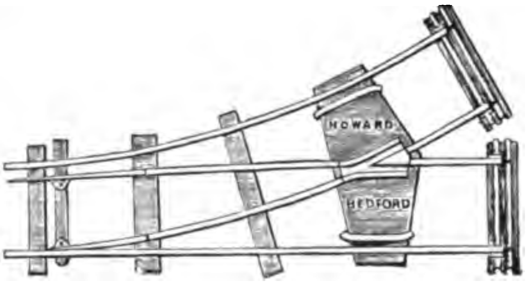


Fig. 290.—Left-hand Points.

The points and crossings for main underground roads or for surface tramways are shown in Figs. 288 to 291. Fig. 288 is a three-way, Fig. 289 a right-hand, and Fig. 290 a left-hand set of points and crossings, while Fig. 291 shows the arrangement of a siding.

The sidings are formed, as shown in the illustration, by a set of points and crossings, either right- or left-hand, a section of curved line, and a number of sections of straight line.

Pass-byes in a straight line are made by a pair of right- and left-hand points and crossings, with two curve pieces and the desired length of line between. On single lines of rail of considerable length these pass-byes are necessary, and if a large number of trams have to pass to and fro they must occur frequently.

For special curves, which must be shaped on the spot, an ordinary rail bender, called a "Jim Crow," is required.

The points are similar to those described for surface railways, but for horse-power lines no switch boxes are required, and frequently the use of levers is dispensed with. For locomotive traffic the points and crossings can be provided with a signal disc or lantern.

The single right-hand or the single left-hand set may be so contrived, that by taking a pair of rails from any part of the straight line, the points and crossings will take the place of the removed rails without cutting or fitting, and this removal can be quickly effected.

Wherever possible single sets of points should be arranged for, the wear being less on two pairs of single points that on one set of double points, beside which single sets are simpler to lay down and keep in order. Three-way points and crossings are avoided underground by turning right- and left-hand roads off a straight one in positions which are not quite opposite to each other.

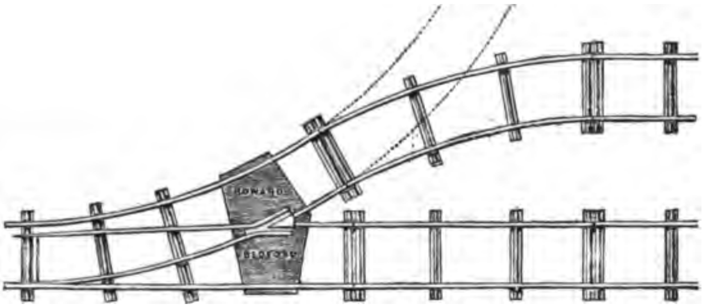


Fig. 291.—Arrangement for Siding.

The points and crossings are made of standard sizes with curves of 20, 25 and 30 feet radius, and where locomotives are used they are made with curves of 100 feet radius, or to any special radius required to suit the gauge of the rails and the wheel base of the engines. For locomotives on a main tramway the curves should not, as a rule, have a radius of less than from 150 to 200 feet.

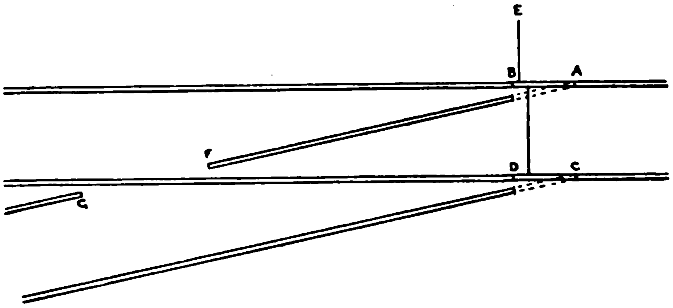


Fig. 292.—Points, &c., used Underground.

A form of points used on some engine-planes in South Wales is shown in Fig. 292. Here, instead of a pair of tapered movable tongues, are two ordinary square-ended rails, AB and CD, connected by a cross bar. The inner end of these rails is moved instead of the outer, as is the case with tapered tongues, by means of a lever at E, and the connecting rod EB. The joints of the rails at A and C are of the ordinary fish-plate type, with the two outer bolts omitted, or shorter fish-plates can be substituted. The space between the rails then allows of the ends at B being moved into the position shown by the dotted lines. In both positions of the points they are prevented from going too far by studs placed outside the rails, which ensure their proper alignment with the rails forming the continuation of the road. If the branch road leading from a main haulage road is only required for occasional use, the necessity for a check rail and a V crossing is sometimes obviated by the arrangement shown at F G in the same Fig. Here the inside rail of the branch road is not continuous, but is laid with a space reaching from F to G, where it crosses the main road rail. A striding rail of the right length is placed over the main line rail to bridge this space when required. A portion of the metal of an ordinary rail is cut away at the intersection of the two rails, and the striding piece fits into grooves at F G. This method raises the branch road two or three inches above the level of the main road. Sometimes, instead of a striding length, a short rail reaching from G to F, secured to the sleeper by an upright pin with counter-sunk head, is used. This rail may be moved on its central pivot so that its ends form a continuation of the rails laid for either the branch or main road.

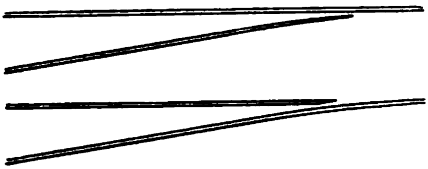


Fig. 293.—Fixed Points at a Pass-bye for Single Tubs.

Wherever lever-worked points occur on main roads an attendant must be placed to set the points in the required direction, and to receive and give signals and perhaps to change the ropes.

The turn-outs or pass-byes for horse traction are sometimes laid with fixed points (see Fig. 293). These are not suitable for journeys of more than one or two tubs, as each empty tub must be pushed over towards the road it is to go on as it approaches the points, or otherwise it is apt to take the wrong road. The points here do not interfere with the outgoing tubs, but at the other end of the turnout, the same pushing over of the full tubs towards their road is necessary. Movable points are much better for long trains of tubs. If desired they may be counterbalanced.

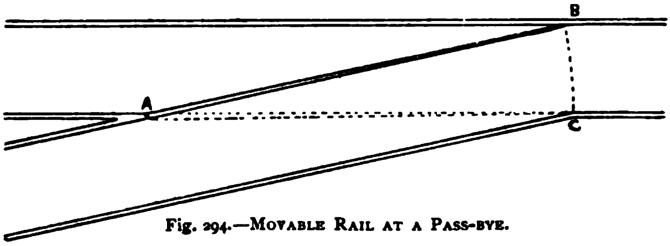


Fig. 294.—Movable Rail at a Pass-bye.

A rough means of diverting the tubs at a pass-bye laid for a temporary purpose is shown in Fig. 294. It consists of a movable rail, A B, pivoted at A. The end at B can be moved by hand to the point C. The trams take one of two roads in accordance with the position this rail is moved into.

The points, rods, and levers used underground are usually made and fitted by the colliery blacksmith, the plate-layer assisting in the fitting. The crossings are cast, patterns for a right-hand V and diamond crossing, and of a left-hand V and diamond crossing, being kept for the purpose.

It is sometimes impossible to avoid the intersection at right angles of roads in the workings. Here it is impossible, without removing coal or the corners of buildings, to lay a three-way set of points and crossings which require room for the curves to be formed. For small trams the difficulty is met as shown in Fig. 295. The rails for the four roads terminate as shown in the drawing, the space about them having first been laid with wrought- or cast-iron plates. Before the plates are laid the floor is prepared with an even surface, over which planking is carefully laid to which to fix the plates. At the termination of the rails they are opened out a little, and raised guides are fixed curved in the contrary direction. By these means the entrance of the tub is facilitated. The iron plates allow of small tubs being guided over them in any direction, but the flanges of the wheels wear grooves in them sooner or later, and where these plates are laid at the pit bottom or pit top or at a self-acting incline, as they frequently are, the large amount of traffic in these positions soon results in grooves in the plates. To prevent the necessity of removing a whole plate, loose wearing pieces may be introduced which are quickly changed.

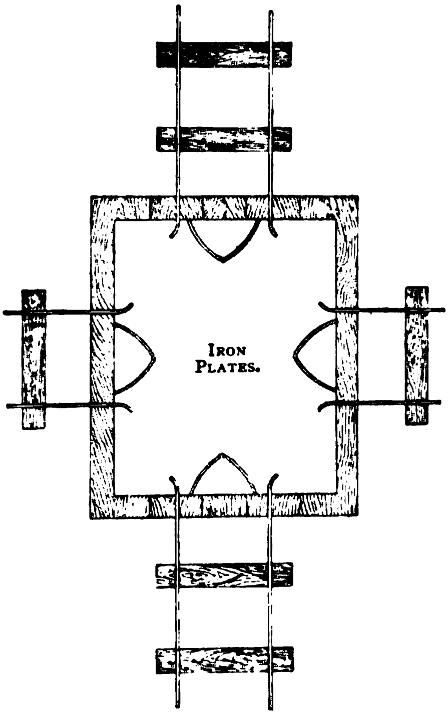


Fig. 295.—Iron Plates at Junction of Roadways.

Heavy tubs are not so easily pushed about on the iron plates, and it may be necessary to use a turn-table. This can be made of any gauge, and when the tub is placed on the line of rail formed on the table top it may be turned on the table which revolves to the position desired. It is in confined positions that turn-tables are necessary. They are only permissible in positions where there is little traffic. A train of trams will pass a curve in less time than a single one can be turned on a table. For that reason on main roads it is necessary to take out sufficient coal, pack-walls or timbers at the sides, and round off corners to allow of the formation of proper curves, rather than to use turn-tables or iron plates, even for horse traction.

It is not practicable for the underground tramways to be laid with the same care and attention a surface railway receives. The rails must be laid in the roads, without the possibility of greatly changing the gradients. The most that can be done is to improve within reasonable limits of cost after making careful surveys and levellings of the roads intended for the main arteries of the colliery. The levellings will show swamps, hillocks, and minor undulations, which when removed will certainly slightly improve the gradients, which, however, cannot be materially altered in this way. The plans aid in showing how to improve or to form the curves necessary for branch roads and for the main road itself, but for the most part the windings of the road have to be followed. The curves on a surface railway are always arcs of circles of a certain radius, except at the extremities, where they join straight lines. The underground sinuosities are mostly made up of a series of short straight lines, at the most pronounced angles of which the rails are roughly curved with a "Jim Crow." Even when it is foreseen that a certain road will be required for an engine-plane, and it is in consequence kept straight for a time by using marks put up by a surveying instrument, for some reason or other, such as the crossing of faults or to alter the gradient, its direction is changed. The turns and undulations nearly always found in underground roads do not allow of the laying of tramways in them on correct principles, and consequently the speed of the trains is reduced to suit such lines as are possible. The sleepers are laid on the floor, which is made even to receive them, and the spaces between are filled with stones obtained in the rippings of roads, &c., and broken up for the purpose. If care is taken that the sleepers rest evenly on the bottom, the whole tramway may be securely carried thus, or with only such displacement as results from the squeezing of the roadway, the effect of which can only be guarded against by constant attention.

The size of tub used will depend upon the height of the seam to be worked, and it may be made of wood, iron, or steel. A very common form is to have oak framing below, the bottom and sides being made of ¾ of an inch or an inch oak or elm, the sides being strengthened by straps of iron. A drawbar of iron or steel passes the length from end to end, secured to the framing, and this bar has a hook at one extremity and a coupling chain at the other. If the tub has vertical sides, the wheels, which are flanged, are placed below, and are from 8 to 12 inches in diameter; but when the tub is narrower at the bottom than the top, the wheels may be set outside, and be from 15 to 18 inches in diameter. They may be made to turn on the axle or with it, the latter being the better plan if the roads are straight and the tubs are not run at a high speed, but where there are curves in the road, the loose wheels work better, particularly if a high speed round the curves has to be maintained. The axles are usually about 1¼ or 1½ inches in diameter.

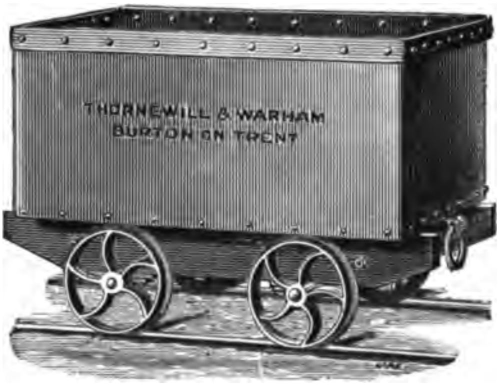


Fig. 296.—COLLIERY TUB.

Fig. 296 shows an ordinary type of colliery tub with iron or steel body, timber frames, and cast-steel wheels and axles as made by Messrs. Thornewill and Warham, Engineers, of Burton-on-Trent. The tubs are strongly yet lightly made, and of a carrying capacity varying from 5 to 20 cwts. The sides and bottom are connected by an internal angle-iron frame and the top edges are stiffened by a flat iron frame, both being riveted to the body sheets. The wood frame is bolted to the body and provided with cast-iron open bottom chocks to receive the axles secured by bolts. A draw-bar and links for coupling tubs together are fitted to the frame. The wheels are of a special quality of cast-steel and secured to the axles.

Tubs are made by Messrs. Thornewill and Warham of various sizes and types to suit the requirements of the different colliery districts, some being wholly of wood, with iron bands and fittings, others are wholly of iron; some have closed ends, as that in Fig. 296; others have one end open with two iron bars across.

In some districts tail-boards are used on the tubs in order to admit of filling and emptying them easily. Where tubs are subjected to heavy work arising from steep inclinations and undulations, or from passing through ill-kept, muddy, and water-logged roads, the wear and tear are excessive, and necessitate frequent repairs. At collieries where these circumstances prevail, it is found more economical to have tubs made wholly of wood, with iron bands and fittings, being easier to repair than those having iron or steel bodies, which are bulged out of shape by getting off the rails and striking the rough and uneven sides of the roadway when in rapid motion on inclines, or elsewhere.

For the removal of hot ashes from a ventilating furnace or boiler-fires underground, an iron or steel tub must be used.

To make the resistance equal in both directions along the main levels, assuming the empty tub to be 3 cwt., the loaded one 12 cwt., and the friction to be $\frac{1}{80}$ of the weight, the inclination would have to be 1 in 133. This rise would give the most advantageous effect to horse power, but it is usual to give a less inclination to the main levels, viz., about $\frac{3}{16}$ of an inch per yard, or 1 in 192, in order to win as much coal as possible to the rise of the levels.

The lowest gradient at which a self-acting incline will work depends on the state of the road and the load, but even under the most favourable conditions the gradient requires to be about $1\frac{1}{4}$ inches per yard, or 1 in 28. Tubs to carry from 6 to 9 cwt. of coal weigh about 3 or 4 cwt., and the larger tubs in proportion; they may be taken to weigh from $\frac{1}{2}$ to $\frac{1}{3}$ of the weight of coal they carry.

Where the output is large, and horses are employed, it is necessary to have two lines of rails on the main roads—at least for some distance from the shafts. Where there is only one line of rails, there should be good sidings at frequent intervals, which greatly facilitate the inward and outward tram transport.

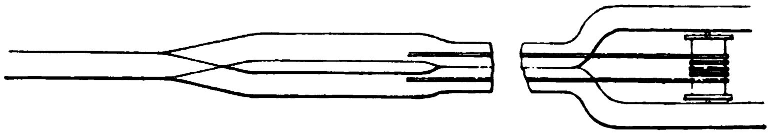


Fig. 297.—Self-acting Incline.

Self-acting inclines may either have a double line of rails laid, or a single line from the bottom to the lower ends of the "meetings," from which suitable points are laid, leading to the double road, and at the top end of the meetings 3 rails are laid to the top of the bank, as shown in sketch, Fig. 297.

The advantage of the incline, as shown in the sketch, is, that a narrower road will do for it throughout its course, except the "meetings," than where a double line of rails is used. This is sometimes important where the sides and roof are bad, or the incline is driven as a stone drift. A rope or chain may be used on the incline, but a rope is preferable, and a drum or a sheave may be used for it to run on. A drum is more suitable on steep inclines, unless the sheave has some gripping arrangement in the groove, such as Fowler's clip pulley. A brake for regulating the speed of the trains is applied to the ring on the drum or sheave, and may be a piece of wood fixed in a lever, or an iron strap gripping a half or the whole of the periphery. The rope may be in one, having 3 or 4 coils round the drum, and one end attached to the tubs at the bottom of the incline, and the other to the loaded ones at the top, or there may be two ropes on the drum, coiling in different directions. Friction rollers, to carry the rope, should be laid at intervals of 8 or 10 yards throughout the incline. In laying the rails, long sleepers are used above "meetings," to fasten the three rails to. At the lower end of meetings the points are of peculiar construction, being movable on a pivot, and are self-acting, the full train in descending placing them in proper position for the ascent of the next empty train. The full trains of tubs are thus placed alternately on each side of the incline. A common means of communication between the bottom and top of the incline is by a rapper wire, having attached a hammer at the top and a lever at the bottom of the incline; when the rope is attached to the empty tubs at the bottom, the attendant there "raps" or signals to the attendant at the top, who, when the full train is ready, secures the rope at the end of the set, and lets it down.

Blocks or stops are placed at the bank-head as a safeguard against tubs running down before all is ready. Fig. 298 shows one form of these blocks. They consist of two pieces of wood about 9 inches long and 4 inches square, each moving on an upright pivot. When in place, as shown by the solid lines on the sketch, Fig. 298, they form a stop to the loaded tubs coming out, and when the attendant wishes to let the tubs down the incline, he knocks the stops aside with a hammer, as shown in dotted lines, thus leaving the rails clear of the obstruction previously existing. For large trams stronger "chocks" are required.

More elaborate safety catches for inclined planes have been devised than the form of chock just described.

Mr. Reumaux's catch consists of a bent lever, having two unequal arms, placed flat on the ground and turning round a vertical axis situate about its middle. It is placed parallel to the tramway and at such a distance that at least one of the ends always bars it, and the short arm is next the side of the inclined plane. As the empty tub reaches the top of the incline and passes forwards, its front wheel touches the long arm after the hind wheel has passed the short arm. The apparatus then blocks the way. When desired to have another run on the incline, the attendant, in pushing the tub upon the same tramway, is stopped by the bent arm, and before he can pass on with the tub, he must push the arm back with his foot. Each road must have a safety catch placed on it. It is stated to be cheap, simple, and effective.

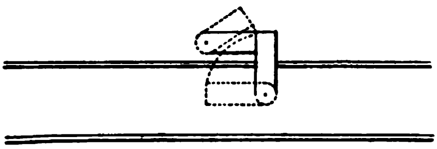


Fig. 298.—BANK-HEAD STOPS.

Another safety catch is that of Mr. Mortier. It consists of an axle with attached levers, placed in the axis of the tramway and supported upon the sleepers. The axle is movable, and in its two extreme positions, one of the two levers clears the tub axles, thus allowing the tub to pass, and in the other the same lever stops the tub axles. In the middle position assumed by the axle, both levers bar the passage of the tub. The empty tubs, in passing up the incline, work this catch automatically. This is done by sloping off the external edge of each lever, and the levers are thrown over by the axles of the tub. When the attendant wishes to pass a full tub downwards he removes the lever obstructing the way by means of a pedal provided with the apparatus. It is said that the catch cannot be put out of order either wilfully or by neglect.

The drum fixed at the top of the incline for the rope to coil on is supported in its position by timber. Two uprights are securely set in the floor and roof, and the drum turns upon bearings fixed in these uprights.

Besides the safety catches described, which are fixed at the top of the incline, an additional safeguard is a "Seizer," shown in Figs. 299 and 300, and this is placed at the foot of the incline. It consists of an iron arm bent in the shape of the letter L. This arm is carried by means of bearings from the point where it is bent. When in position to prevent a run on the incline, a bar of wood, which may be protected with strips of thin iron, is moved round horizontally at one end, the other being secured by an iron pivot round which it moves, until it assumes a position under the L-piece, which may be held back with one hand during the operation. The horizontal portion of the L, on being let go, rests on the bar A large link is placed in the middle of the short-linked chain at the end of th incline rope, and this link is now dropped over the upright portion of the L. This is done each time immediately after the descent of the full tubs, and the link remains on "Seizer" until the empty tram is attached. Should the attendant at the top, through any mistake, push the full tubs over the top of the incline with the rope attached before the empty trams at the bottom are ready, their further descent is arrested so long as the link remains on Seizer, as the bar under the L-piece holds it and the chain at the end of the rope securely in position.

When the attendant at the bank foot has attached the rope to the empty trams, he gives the bar under the L a blow with a hammer, causing it to resume its position clear of the L shown by the dotted lines, Fig. 300. The L then drops into the position shown on the drawing in dotted lines in Fig. 299, and in so doing the large link of the chain falls free from the L-piece. The attendant at the foot of the incline then signals to the top. As shown on the drawings, a slight hole is prepared under the floor level in which the L swings, and the bearings are carried on a slight timber frame prepared for it.

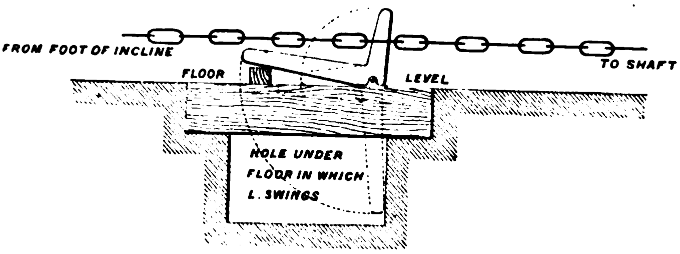


Fig. 299.—Seizer in Elevation.

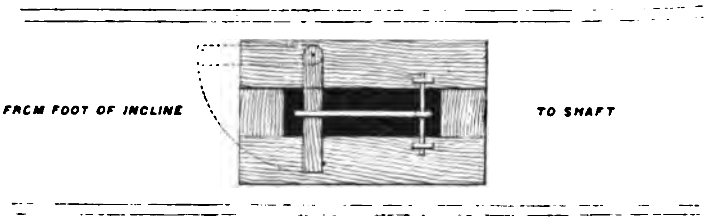


Fig. 300.—Ground View of Seizer.

Many accidents occur at collieries where stops of the description shown in Fig. 298 are placed at the top of self-acting inclines, and these alone are relied on without the further safeguard of a "Seizer" at the bottom. The danger of this arrangement is increased where no regular attendant is placed in charge at the bank top, and where, consequently, each putter or twinboy (after reaching the bank top with the loaded tub he has brought from the workings) lets it down the incline. In some instances the loaded tub reaches the top of the incline by a road the inclination of which is not sufficient for the tubs to be spragged, and yet is too great for the putter to reach the bank-head quietly without using considerable force in holding back the tub. In Somersetshire, the practice is to wear a rope band round the waist, from which hangs a short chain at the front of the wearer. A "crook" or short rod of iron with a hook at each end, bent in opposite directions like the letter S, is used to attach the end of the chain to the tub. On approaching the bank-head, as over other steep parts, if the road traversed is irregular in gradient, the putter hitches his crook and then holds the tub back, placing his feet (frequently naked) firmly against the sleepers to give him a foothold and considerable holding-back power. Frequently the stops, through forgetfulness, are not placed in position at the bank-head, and if then the putter reaches it as now described (having full confidence that the stops are rightly placed), the tub may pass these before he is aware of it, and directly afterwards continue over the brow of the incline, which perhaps has an inclination of 1 in 3, and anyhow is much too steep for the putter to longer control the tub he is attached to. Overpowered by, and unable to disengage himself from it, as the tub increases its speed, he is thrown from his feet and dragged along till the tub reaches the foot of the incline or is more quickly stopped by getting off the rails in the incline. Many fatal accidents from this cause have been recorded. Others occur from runaway tubs down inclines where the attendant omits to attach the rope at the top before knocking out the stops, and so causes an accident. Accidents on inclines may also be produced in other ways.

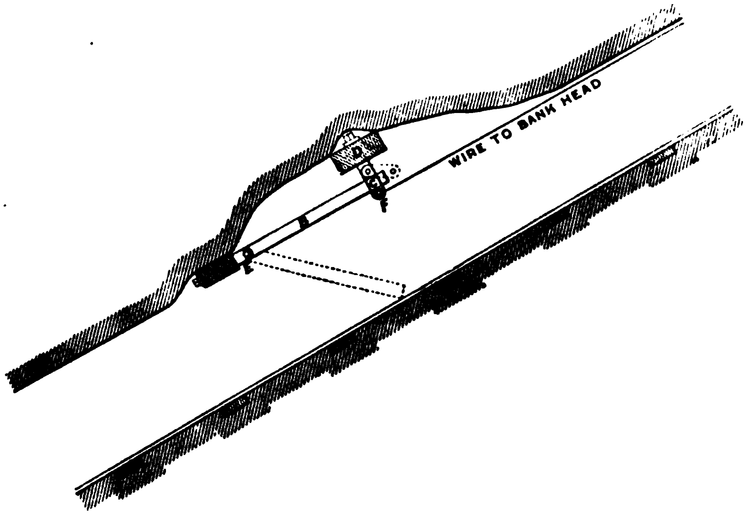


Fig. 301.

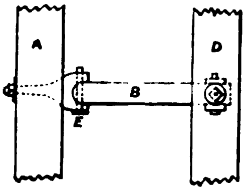


Fig. 302.

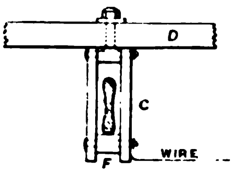


Fig. 303.

Arrangement to Stop Runaway Tubs on Self-acting Inclines.

Figs. 301—303 show an arrangement which may under some circumstances be beneficially applied for guarding against accidents from runaway tubs down inclines, although no appliances of this kind are intended to supplant the care and vigilance which are essential at inclines, as, indeed, at all other underground positions.

To an ordinary cross-bar, A, Figs. 301 and 302, are attached a strong pair of jaws, E. A piece of railway metal is then taken, the length of which should be about 1½ times the height of the incline at the place it is to be fixed. A hole is drilled through the jaws, E, and also through one end of this piece of metal, a pin being afterwards passed through these holes. Through another cross-bar, D, is passed an eye-bolt carrying two swinging plates, C, and near the bottom of these plates is placed a cross-piece from one to the other, on which the piece of railway plate, B, rests when up out of the way ready for use, see Fig. 303. A wire is attached to the bottom of the plates, C, which is carried to the bank-head a suitable distance from it. A lever is attached to the wire at the bank-head, by means of which, if the incline man should happen to let any tubs run down the incline without having attached the rope, he may by pulling it give motion to the plates, C, which then take the position shown by the dotted lines; the consequence is that the railway plate drops into the position shown by the dotted lines on to the middle rail of the incline, if the appliance is placed above meetings, or between the two rails if placed below meetings, thus effectually stopping the tubs from running any further.

The apparatus is simple, and may be placed in one or more positions on the incline, according to the length of it, and the same wire may communicate with all from the bank-head.

In Fifeshire a very curious arrangement of self-acting inclines is in force, called the Fife system of "cut-chain" haulage on inclines. These cut-chain inclines cannot be worked advantageously unless the inclination is from 1 in 3 to 1 in 6, and it is found in practice that the greatest distance which will work effectively on a gradient of 1 in 4 is from 60 to 70 yards.

Where the system is adopted, the winning place which is to be the inclined plane, must be carried wide enough to have a double line of rails laid throughout. It is driven in the direction of full "rise," and level roads are turned out of it on either side every 15 yards. At the point of turning the level roads, a platform consisting of two cast-iron plates each 3 feet square is placed, to turn the tubs upon, and there are four movable rails for joining the lines on the incline when required.

Tubs may be run from the top of the incline or from any of the branch roads on the incline.

The method will be best understood by reference to the accompanying plan, Fig. 304. A main chain reaches from the bottom, F, to the top of the incline, passing round the wheel at B, and so far the system resembles the ordinary self-acting incline. Tubs may be attached to the main chain at B, and run in the ordinary way. In addition to the main chain, short or "cut" chains are used between the top of the incline and the different branch roads. The cut chains are shown on the plan, one from B to C, another from C to D, and a third from D to E.

If desired to run tubs from the top of the incline, the cut chains are not used, but rest on the floor between the rails. At each end of the main chain, and also at the lower end of each cut chain, is a link, see Figs. 305 and 306; and at the upper end of the cut-chains there is a hook shown in Fig. 307. When it is desired to run tubs from the branch, C, the hook of the cut-chain is attached to the link of the main chain at B, and the link upon the lower end of the cut-chain is connected to the draw-bar of the tub to be let down. After this connection, a chain is continuous from F to the top of the incline, round the wheel there and down to C. The movable rails on the incline at C having been lifted, the full tub is put over at C, and runs to the bottom F, the empty tub being taken up from F to the plates at C.

An even number of runs must be made from each branch at a time; 2, 4 or more may be made, but if only 1 or 3 be made the cuts cannot be put together to allow the next branch at D to run, unless it be possible to put the "cut" across to the other side of the incline to be connected. The roadways being wide, frequently require a row of props between the two lines of rails, and where this is so, it prevents the cut-chains from being readily thrown across from one side to the other. By an even number of runs at a time the cuts may be easily adjusted for other branches.

During the time of the runs from C, the cut-chains between C and E, rest on the floor between the rails.

If desired to run from E, the cut-chain from D to E is attached to the cut, CD above it, which again is attached to the one, BC, above it, and the letting down is similar to that described at C. The same operation is required when running from the branch D.

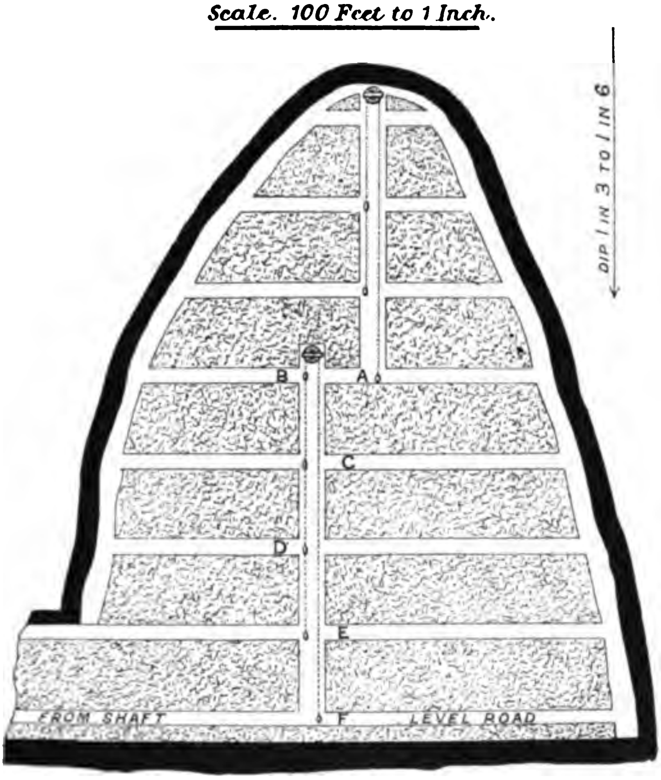


Fig. 304.—Fifeshire System of Cut-chain Haulage on Inclines.

When the heading or winning place has advanced 60 or 70 yards, it becomes necessary to set off either to the right or to the left, whichever may be most convenient for siding room, and drive another incline at A. When there is a second incline, as at A, above the other, two tubs at a time may be run from F to B, to supply the incline A.

In order to regulate the speed of the tubs, a friction strap is fitted upon the wheel, and a wire attached to the handle is carried down the incline to the lowest branch, E, so that after putting over a tub at any branch road, the person in charge of it can stop it, or regulate its speed by pulling the wire which brings the friction strap into action.

On the Continent, where the coal seams are highly inclined, a somewhat curious arrangement of self-acting inclined plane prevails. It will be understood by reference to Fig. 308. The road is laid with four rails, the outer pair of which forms the gauge for a sort of tram, upon which is built a horizontal platform. Upon this platform the tub, whether full or empty, is carried.

The inner line of rails forms the gauge for another tram, which is long, narrow, and heavy, and acts as a counterbalance. This counterbalance tram is made so low that it passes underneath the other at meetings. The two trams are attached to ropes which are wound on a drum placed at the top of the incline.

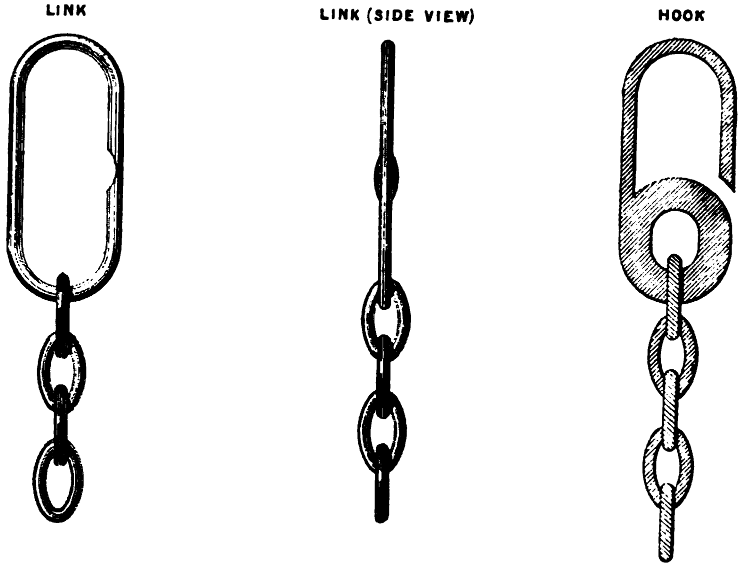


Fig. 305. Fig. 306. Fig. 307.

Hook and Link used on Cut-chain Inclines.

The weight of the full tub in descending is sufficient to raise the counterbalance; the latter, in turn, being heavy enough to pull up the empty tub.

At large collieries for the most part, the underground conveyance is by means of engine planes. Although hand and horse putting may be very useful over short distances, and self-acting inclines work very economically, the circumstances of all collieries having long flat or dipping roadways leading from the shafts, or long roadways of varying gradients, demand some system of haulage by machinery. Few collieries are so favourably circumstanced as to require no haulage by machinery. In the improbable case of a pit sunk in the bottom of a basin, the haulage would be the cheapest possible. In most cases the coal has to be hauled a considerable distance, and in some for two or three miles. In many collieries a portion of the field lies below the level of the pit bottom, but may not admit of a road dipping uniformly all the way to reach it. Steam engines for underground conveyance may be placed underground, and the steam be either generated on the surface, and conveyed to them, or the boilers may be placed underground close to an upcast shaft. It is probably the better plan to have the boilers all together on the surface. The smoke passing up the upcast deteriorates the shaft lining, and it is obnoxious in a pit used for the ascent and descent of workmen. Moreover, it is highly objectionable to have large underground fires in a fiery mine, and a source of great anxiety in any. Whatever system of rope haulage is adopted, the engines may be placed either on the surface or underground, but the engine for driving an endless chain should be placed underground. A chain is about three times as heavy as a rope of equal strength, and it adds very much to the power of an engine to have the additional weight in a deep shaft. Another objection to chains being in the shaft is that they are more liable to break than ropes.

Again, where compressed air is used as the motive power, this may be generated above ground and conveyed to the hauling engines underground. The system of haulage may be *direct* with one rope, which the weight of the empty tubs is sufficient to run back, or it may have a double line of rails, the empty train of tubs in that case slightly assisting the full train, or it may be by *tail rope*, which necessitates the use of two ropes, one to pull the loaded tubs out and the other to draw the empty tubs in. Then there is the *endless rope* system, which is capable of two applications, first with the haulage rope below the tubs, and secondly with the haulage rope over or by the side of the tubs. There is also the *endless chain* system.

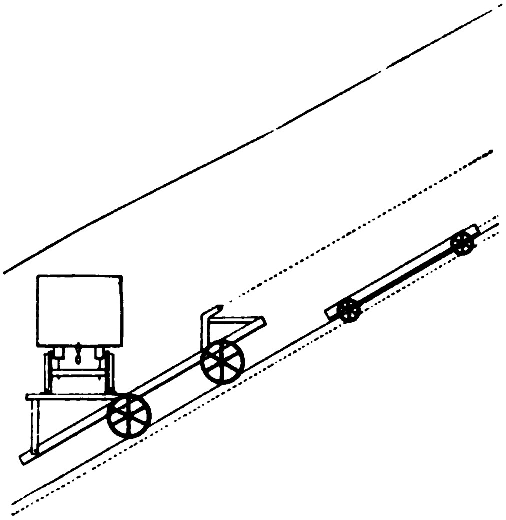


Fig. 308.—Counterbalance Arrangement of Self-acting Inclines in the Steep Measures of Northern France and Belgium.

Direct Haulage.—To apply the direct system it is necessary that the fall from the shaft be more than 1 in 30 to overcome gravitation; it need not be a uniform dip, but no part of the road should have a less inclination than 1 in 28. The engines for hauling the loaded tubs up this incline may be placed vertically or horizontally: single or double engines may do the work: the drum or drums may be placed on the crank shaft, or may be on the second motion with spur wheels. The best form is undoubtedly a pair of horizontal engines, and if the speed required is great, the engine should work direct as in the ordinary coupled winding engines; but if the speed desired is low, the advantage of spur gearing is that it enables a smaller engine to do the work. A decision on this point will be governed somewhat by the speed of piston in the engine. Mr. Percy's rule is to allow a piston speed of 350 or 400 feet per minute for an intermittent haulage as this would be, but for one running continuously it should not exceed 250 feet per minute. A speed of six miles an hour on the average may be safely taken, and much higher, up to 10 or 15 miles an hour, if the roads and tubs are in good repair and properly maintained.

A single road only is necessary for direct haulage, though a double road may be used if desired, and friction rollers should be laid at intervals of 8 or 10 yards throughout its length to carry the rope. The timber on which the rollers rest should reach between the sleepers as per sketch, Fig. 309, and the height of the floor rollers must depend upon the clearance under the tub, which again will depend on the size of wheel used. Besides the floor rollers it will be necessary in going round curves to place either long rollers with their spindles set vertically or bevelled pulleys. If the dip " in bye " is not regular, but rather decreases, it will be necessary from the point at which the dip changes to place one or two roof rollers. On steep inclines worked by direct haulage, there is necessarily a great strain on the coupling chains and drawbars of the tubs and the capping of the rope. These should be made of the very best material and subjected to a careful and frequent inspection. The truest economy is to have ropes of the best steel, and all other materials equally good.

It is customary to place a " trigger " or bar-hook, sometimes called a " devil " and a " bull," behind the last tub of the full train. This is a kind of fork with one end firmly fixed on the back end of the last tub and the other end, which is pointed, trailing along the ground so long as the train is ascending, but if a detached or broken coupling chain or rope causes the train (or the latter part of it) to run backwards, the "trigger" is forced into the ground, thus arresting the further backward movement of the tubs. This arrangement applies only to the ascent of the full train, and is not applicable to the empty going down. A bridle chain may also be used, and this is a means of safety to both the upward and downward set. The chain is made fast behind the last tub going up (which will be the first going down) and is passed under the train of tubs to be securely connected with the hauling rope at the other end of the train. This chain hangs loosely and does not come into operation unless a coupling chain becomes detached or breaks, or a drawbar breaks, thus severing the train, which in that case is almost immediately joined again by the bridle chain. The bridle chain is, of course, no protection in the event of a broken rope whether the train be full or empty, but there is far more liability to a breakage in the former, in which case there is the protection of the "trigger." A number of accidents occur every year on these inclines, and other travelling roads for the workmen should if possible be provided. In any case, besides the refuge holes required by the Act of Parliament, a distinct light such as a red one, which in a mine worked by safety lamps may require to be a safety lamp, should be placed on the front end of the first tub of the full and of the empty train, as well as an alarm bell to sound continuously whilst the train is in motion, so that fair warning may be given of the approaching train both to the eye and ear of all on the incline. By direct haulage, coal may be brought from more than one part of the mine. Branch roads may lead off the main road at any number of places, but each one will require an attendant, if the work is great, to set the points and attend to the signals.

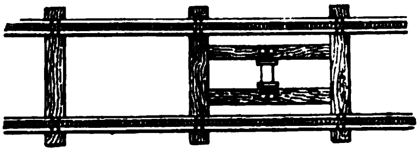


Fig. 309.—FRICTION ROLLER ON ENGINE-PLANE.

THE TAIL-ROPE SYSTEM.—This has the great advantage of being applicable to almost any condition of road, a single line of rails only being required. Althougth crookedness and irregularity of the gradient need not preclude the adoption of this mode, it is much better applied where roads are straight and have regular gradients. It gives great facilities for working branches.

The engine for working the "tail-rope" should be double and placed horizontally. It must have two drums, one for the main rope and the other for the "tail," and the engine is worked on the second motion, each drum being on a separate shaft and connected by spur gearing to the engine shaft. Each drum may be put in and out of gear by means of clutches worked by levers. The length of rope required to work any plane is three times the length of the plane, the "tail-rope" being long enough to reach from the front end of a train of empty tubs standing at the out-bye end of the plane to the "in-bye" end of the plane, where it is passed round a large pulley and reaches back along the plane to the drum of the engine. As the train of empties is drawn in, the tail-rope is wound on to its drum, and the main rope (which is usually stronger in the proportion of 3 to 2 as compared with the tail-rope, and must be the length of the plane) is run off the other drum, which is at the time out of gear. After changing the full for the empty train, the tail-rope drum is thrown out of gear and the main rope drum into gear, the engine is reversed, and the full train drawn out, to which is attached the tail-rope ready for the next empty train. The brake is usually applied to the loose drum gently, whether the train is going in or out. This prevents the rope from getting slack. The main and tail-ropes are carried by rollers and pulleys at intervals of about 8 or 10 yards apart. The branches worked off the main road (as

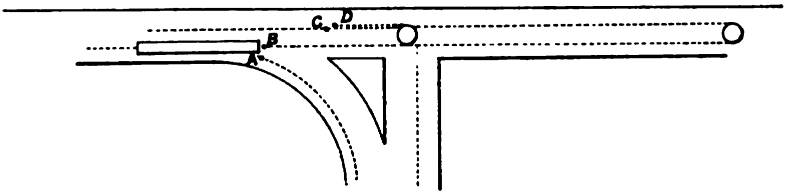


Fig. 310.

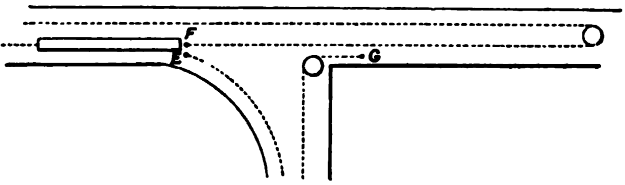


Fig. 311.

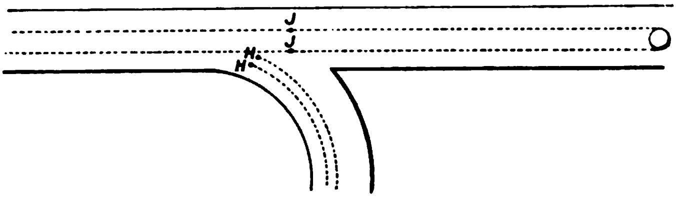


Fig. 312.

Connection for Ropes between Main and Branch Roads.

well as the main road itself) have pulleys placed at the end of them round which a rope passes, its two ends reaching out to the main road; and on the train reaching the branch line, if desired the train may stop by signal, then be disconnected from the tail-rope at the front end of the train and a connection made with the branch line rope. There are different ways of making this connection, as seen by the three sketches, Figs. 310, 311, 312.

In Fig. 310, a wheel is fixed near the roof or under the rails, and one end of the branch rope is brought round this wheel, shown at D. When the train is to be sent into the branch the rope end A replaces B at the front end of the train, and the end D replaces C on the tail-rope. This method requires a winch to pull the branch rope properly in place. In Fig. 311 the end E replaces F, the tail rope is left entire, but being disconnected from the train at F, is brought a little further by the engine and connected to G. This method does not require a winch. The plan shown in Fig. 312 is perhaps the simplest, and in it the position of the ropes is so arranged that when the full train reaches the shaft the shackles on the main and tail ropes are just opposite each other at the branch end, when the two ends J J are changed to H H by the attendant at the branch end.

The change is quickly effected by means of shackle joints on both the main and tail ropes. In Fig. 313 the pin P secures the shackle. At its lower end a thread is cut for screwing through the under side of the shackle. The attendant uses a key fitting the head of the pin, which he readily withdraws by turning it a few times. In some instances the heads of the pins are sunk in the shackle when screwed home, and then the heads have central square recesses formed in them about half-an-inch deep. The attendant withdraws the pins by a suitable key which fits the recesses in the heads.

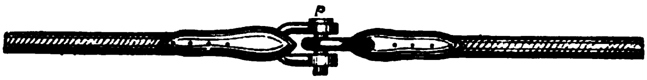


Fig. 313.—Shackle Joint for Connecting and Disconnecting Ropes.

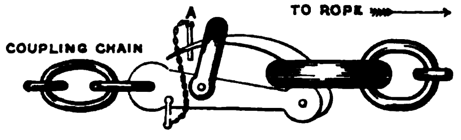


Fig. 314.—Knock-off Link.

Where the plane has no branches, only one set of ropes will be required. On very short sharp curves a single sheave for each rope will be sufficient to prevent their touching the side, but where the curve is of large radius a number of sheaves or rollers must be placed. From 30 to 60 tubs form a train for the tail-rope system. The connection between the train of tubs and the ropes is made by some form of knock-off link, one being shown in Fig. 314. When it is required to disconnect the rope, the cottar A is withdrawn, and the link pushed over by the hand or foot. Where the rise from the shaft is suitable for the loaded train to run out with the rope attached, a tail-rope only need be applied for drawing the empties in, and no main rope would, of course, in that case be required.

The Endless Rope Systems.—The Endless Rope is divided into No. 1 Endless Rope and No. 2 Endless Rope systems.

No. 1 is a modification of the Tail Rope system. The rope has to be kept constantly tight, and this may be done at any convenient point by passing it round a pulley fixed upon a tram, to which a hanging weight is attached (Fig. 315); or the hanging weight may be replaced by a tram working on a short incline, and loaded sufficiently (see Fig. 316). Motion to the rope at the other end is facilitated by a deep-grooved pulley (see Fig. 317), or by a pair of ordinary large pulleys after taking the rope two or three times round each to cause friction. There are several ways of attaching the trains of tubs to the rope

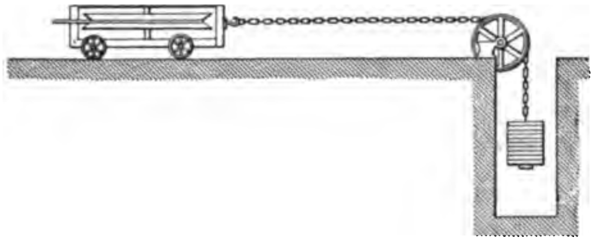


Fig. 315.—Tightening Arrangement for No. 1 Endless Rope Haulage.

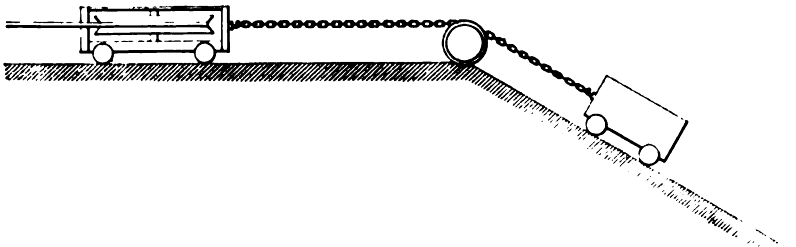


Fig. 316.—Tightening Arrangement for the No. 1 Endless Rope System of Haulage.

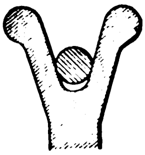


Fig. 317.—Deep-grooved Pulley for Endless Rope Haulage.

(which in the No. 1 system runs under the tubs) by a socket in the rope, and a short attachment chain (Fig. 318), or some form of clamp for gripping the rope.

The mode most commonly employed is that of having a bogie provided with a pair of grips, with which the attendant seizes the rope as required. In this case a train of tubs is taken at once.

A very effective haulage-clip is Hanson's. It is made in two types by Mr. Isaac Hill, George Street Iron and Brass Works, Derby. Figs. 319 and 320 show this clip as adapted for the bogie, whilst 321 and 322 show the form of clip suitable for attaching a single tub or a train of tubs to the rope.

The mode of adapting the clip to bogies, or the present existing bogies, will be seen by Figs. 319 and 320, which is done by very simple means, and at the same time will be noticed that the working of the clip is most effectual for roads of any reasonable gradient.

In the bogies there are fixed two bars of wrought iron, B and D, Fig. 320, 12 ins. apart from each other, and the clip being 9 ins. on the clipping or gripping part, gives 3 ins. of space for the clip to move from one bar, B, to the other bar, D. (Note the distance on Fig. 320 from the bar D to clip.) The clip is suspended on a 1 in. round iron bolt or pin at A A, Fig. 320, and in section of same at A, Fig. 319; and as the gradient of the road alters, the whole clip will slide backwards and forwards upon the bolt or pin, till it is prevented by one of the bars, B or D, Fig. 320.

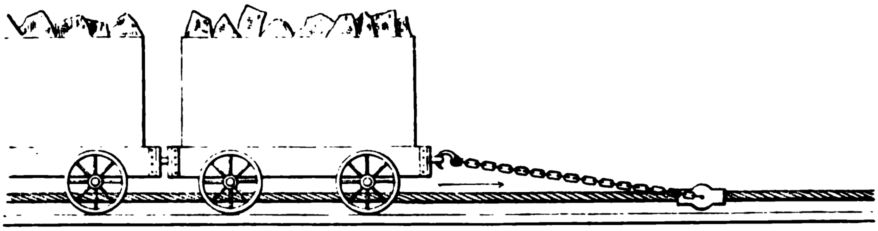


Fig. 318.—Under Method of Tub Attachment to Endless Rope applied at one or both Ends of the Train.

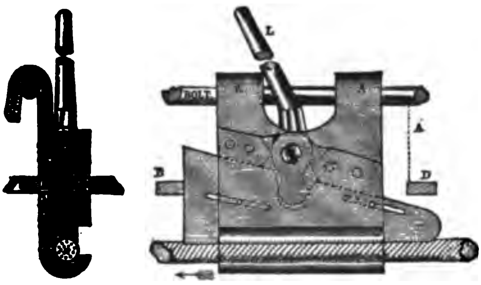


Fig. 319. Fig. 320.

Hanson's Haulage Clip for Bogies.

The clip is represented to be drawing or shunting a load up an incline or on the level in the direction of the arrow in Fig. 320, and it will be noticed that the greater the load or the greater the resistance the tighter the incline block will be forced upon the rope. Should the gradient of the road alter to a descent, the resistance being taken off the bar in the bogie at B, the tendency of the load will be to hasten on at a greater speed, thus causing the clip to slide back upon the bolt or pin at A A, Fig. 320, till it arrives at the bar, D, in the bogie and dotted line A to D, Fig. 320, thus holding back the load and preventing it from obtaining a speed greater than that of the rope. The bar, D, pushing against the body of the clip, tends to force the wedge upon the rope as did the bar, B, when it was on the wedge. The operator lifts the rope into the groove at the lower part of the plate, C, Fig. 319, and at pleasure brings down the incline block by means of lever L, Fig. 320, which block grips the rope as in the hand-clips, the load being at once set in motion.

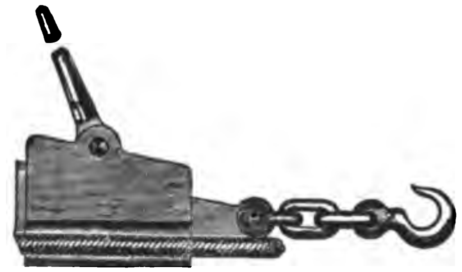


Fig. 321.—Hanson's Hand Haulage Clip.

The bolt or pin acting as a centre for the clip to rotate upon, the latter can be moved at the will of the operator.

This bogie clip is found to be very effectual and sure in work, and can be used by a boy in the pit.

Hanson's hand-clip is shown in Figs. 321 and 322. It will pass round curves very easily in its inward and outward journeys, the lower part being made to suit the groove of pulleys.

The gripping part of the clip being 7 inches long, it holds the rope firmly without exercising a cutting or tearing action on it. The greater the load, the tighter the clip holds the rope. It is easily handled, and can be instantly disengaged by a boy.

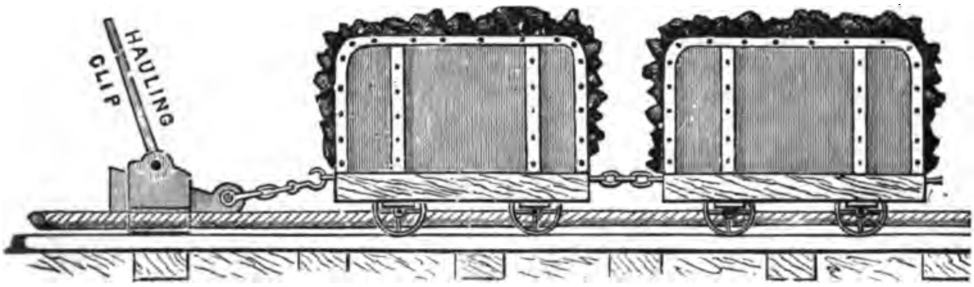


Fig. 322.—Hanson's Hand Haulage Clip attached.

Mr. James W. Smallman's haulage clip is shown in Figs. 323 and 324. In it two side-plates are loosely joined together by a central nut and bolt. The upper ends of the side-plates are formed with inclines, between which the head of the lever works. The ring or hook coupling, by which the clip is connected to the tub or train of tubs, is usually pivoted to the side-plates by a cotter-pin. When the handle of the lever is raised, the side-plates are free, and the clip may be easily dropped on or lifted off the rope. When the handle is depressed, the head of the lever, sliding along the inclines (which should always be kept well greased), forces the tops of the side-plates apart, and thus causes the grooved bottoms of the side-plates to approach one another and grip the rope; the nut serving as the fulcrum for one of the side-plates, the head of the bolt as the fulcrum for the other, and both side-plates as the fulcrum for the lever.

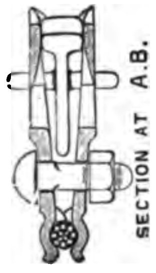


Fig. 323.

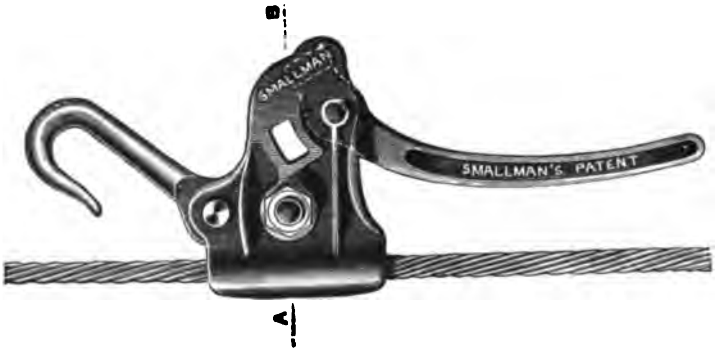


Fig. 324.

Figs. 323 and 324.—Smallman's Haulage Clip.

Each time the operator attaches the clip to the rope, he can tell whether its holding power will be sufficient by the force he exerts in depressing the lever. If the lever go home without the application of much force, the operator should walk along-side the clip, and, while the clip is on the rope, turn the nut with a spanner; the clip being thus readily tightened and perfectly adjusted.

A single or double line of rails may be used, but when single the rope works backwards and forwards, instead of continuously round in one direction; one part of the rope is on the road, and the other runs at the side. With a double line the rope works continuously round in one direction, the empty tubs go in on one line, and the full ones come out along the other. Friction rollers are laid at intervals for the rope in the case of a double line in the middle of the roads, and in the case of a single line along the middle of the road and at the side. The amount of rope required to work this system is, of course, less than by tail rope, and the rope lasts longer. By suitable arrangements at the junctions branch roads can be worked, an attendant disconnecting the tubs while passing round the turn, where the wheel of the branch is fixed. It works on an undulating road, and where there are curves of good radius.

The "Elswick" patent haulage clip, shown in Fig. 324A, was originally designed by Messrs. Anderson and Finney, of the Elswick Colliery, Newcastle-on-Tyne, to overcome difficulties experienced on quickly inclined undulating planes with over tub ropes.

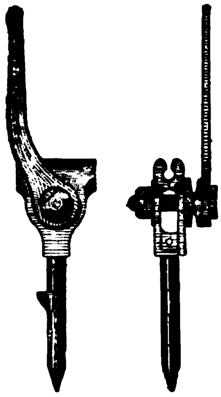


Fig. 324A.—The Elswick Wire Rope Haulage Clip.

Introduced about the year 1895, it is now coming into extensive use for endless rope haulage, both for over and under tubs. A perfectly firm grip of the rope is obtained by the action of a portion of a helical screw plane on the end of the lever acting upon a similar screw plane formed on the hinged jaw of the clip. The rope is harmlessly gripped for a length of about three inches, and is almost enclosed in the rope groove. A slight curve in the rope groove causes the seized portion of rope to act like a spring, and the surfaces of the two screw planes are thereby kept in close contact. This effectually prevents any slipping of the lever which might otherwise be caused by the jolting of the tub. The clips are made to take any specified size of rope and, without special adjustment, preserve an equal grip, even though the rope vary in diameter from constant wear. The clip is locked to the tub by being inserted in a loop or socket on the tub end, and is then turned one-quarter round. The gib projecting on the tapered shank passes under the loop, and the clip cannot be detached from the tub so long as it remains attached to the rope. The erection of sheaves, levers, or other apparatus is not required at the top or bottom of banks, the rope not being detached at any of these points. At the end of the tub's journey the rope is automatically detached from the clip by the lever striking against a bar or roller placed at a suitable height in the road. The clips are made of toughened cast steel, and are exceptionally strong and durable, but the steel is weldable, so that broken clips can be easily repaired. Where tubs are loaded to a considerable height the clips are adapted to attach to the centre bar crook by means of a chain.

The principle of the clips for under tub haulage is precisely the same, except that the mode of attachment to the tub is made suitable to the special conditions of the tub and road.

The No. 2 Endless Rope system, where worked similarly to the Endless Chain system, requires a double line of rails. The rope, instead of being under the tubs as in the No. 1 system, rests upon them, they being attached to the rope either singly or in trains of tubs varying in number from 1 to 12, and they are placed at regular intervals apart.

Some modifications of the No. 2 Endless Rope have been tried, such as carrying the ropes at the sides of the tubs instead of over the centre, and also by using a single line of rails with a suitable arrangement of pass-byes. These modifications do not generally work so well as the system itself, and require special conditions for them to work at all. If the engine plane has curves in opposite directions the rope at the side tends to pull the tubs off the way, and consequently empty tubs at the curves leave the metals. Again, the opposite curves in a roadway give considerable trouble to a single road with pass-byes, although this modification may work fairly well in a straight engine-plane with a uniform gradient, or where the curves are all in one direction, and a regular supply of tubs can be arranged to suit the pass-byes.

Motion is given to the rope, which is fixed much in the same way as No. 1 Endless Rope. The speed of rope in the other systems described is fast, from 6 to 10 miles an hour, but with the No. 2 Endless Rope a slow speed of about 1½ miles or 2 miles an hour is given to the rope. Branch lines, curves and undulations in the road can be overcome, but the fewer curves and undulations the better. There are two usual methods of connecting the tubs with the rope. In either case chains are used. One method consists in fixing one end of the attachment chain to the coupling chain of the tubs, throwing the other end over the rope which is in constant motion; then coiling it 2 or 3 times round the rope, and securing it to the hook at its end. If the road is undulating it will be necessary to put another chain in a similar way at the other end of the tub or tubs (see sketch, Fig. 325).

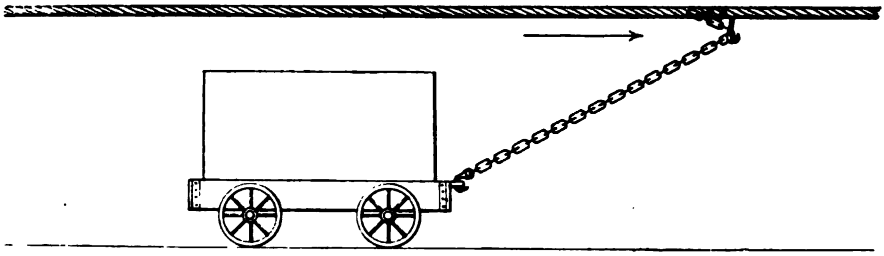


Fig. 325.—Over Method of Tub Attachment to No. 2 Endless Rope applied at either one or both Ends of the Tub or Train.

In the other method, instead of the chain being coiled round the rope, strong loops of hemp 1 inch in diameter are fastened at intervals along the rope by a wrapping of string. One hook of the chain is first attached to the tub, and the hook at the other end is then passed through the loop on the rope. This is not so applicable on an undulating plane where tubs are put on singly, as it would be necessary to put an attachment chain at each end of every tub, which would mean a great number on the rope altogether. Where applicable it is a more expeditious method than the former (see sketch, Fig. 326).

A more expeditious attachment than either of the foregoing methods is obtained by the use of the "Elswick" patent haulage clip, described at Fig. 324A, which is equally applicable in its two forms to either the No. 1 or the No. 2 Endless Rope system.

It is evident that where the coal is loaded up above the top of the tub, unless the roads are high, this system cannot be adopted. In working branch roads by the No. 2 endless rope system, it is necessary to dis-connect and re-connect every chain used in attaching the tubs to the rope, and the arrangement of pulleys to work a branch road is shown in Fig. 327. Each branch must have a separate tightening apparatus. The upper pulley has a clutch-box, and may be thrown out of gear if it is not desired to work the branch line. The rope is passed round the driving pulleys two or three times, the trod being shaped to receive it. If the engine is placed on the surface the rope passes over a pulley at the top of the pit, and goes down the shaft to another pulley at the bottom placed at right angles to the one on top, and if there are a number of branch roads at the pit bottom to be worked the rope is coiled round other pulleys fixed in position to work those branch roads. It is then brought back to the pit bottom, where another pulley guides it up the shaft; it then passes over another pulley at the pit top to the driving pulley at the engine. From each of the main roads leading from the shaft branch lines may, if required, be worked. Two branch lines may be worked opposite each other off the main road by having three pulleys on the vertical shaft instead of two as shown in the sketch, and other branch lines may again be worked off these. In the case of two branch roads working opposite each other off the main road, the tightening apparatus for each branch would be placed at the in-bye end, instead of as shown on sketch. An engine placed on the surface can be made to work an endless rope at another level in the shaft in addition to those at the pit bottom. There is no "runrider" required in this system as in No. 1, and the speed of the ropes tends to their duration.

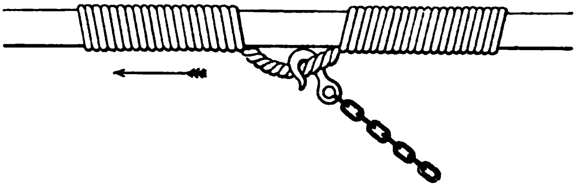


Fig. 326.—Over Arrangement of Attachment to No. 2 Endless Rope applied at either one or both Ends of the Train.

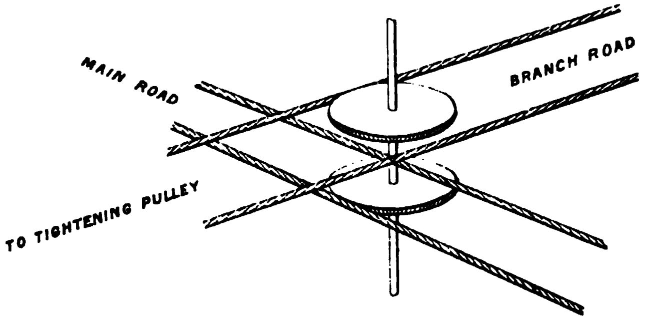


Fig. 327.—No. 2 Endless Rope System of Haulage. Arrangement of Ropes for a Branch Road.

Endless Chain System.—The Endless Chain System is somewhat similar to the No. 2 Endless Rope, and a double line of rails is always requisite in its application, one for the full, and the other for the empty tubs. The chain may be passed 3 or 4 times round the driving-wheel, which consists of an ordinary sheave, round which a piece of boiler plate about 10 inches wide is fixed, and to this are attached about 12 steel or iron "feet" on which the chain rests without touching the plate; or forks a few inches apart may be fixed in the trod, the chain then only passing half-a-turn round the wheel. The driving-wheels are placed horizontally, and motion conveyed from the engine to them by bevel gearing. The return wheels are ordinary sheaves, round which the chain passes half-a-turn. No tightening arrangement is needed, for the chain rests upon a fork in the tubs, which are always placed singly on the chain at distances varying from 10 to 40 yards apart, and it is better not to have them further than 30 yards apart, so that there may be no danger of the chain touching the floor. The speed at which the chain is driven varies from 1 to 4 miles an hour. Owing to the low speed of the chain, the condition of the rails is not of so much importance as with the Tail Rope and No. 1 Endless Rope, where the speed is greater, but care is required at the termini in the formation of the roads. On the "coming off" side the way is made to rise a few yards from the terminus, in order to obtain such a fall as will enable the tub, on leaving the chain, to acquire an impetus which carries it off the road; on the "going on" side the way is made to dip from the terminus, so that the tub may run to the chain without the aid of manual labour. Fig. 328 shows the arrangement of working roads leading in different directions from the pit bottom.

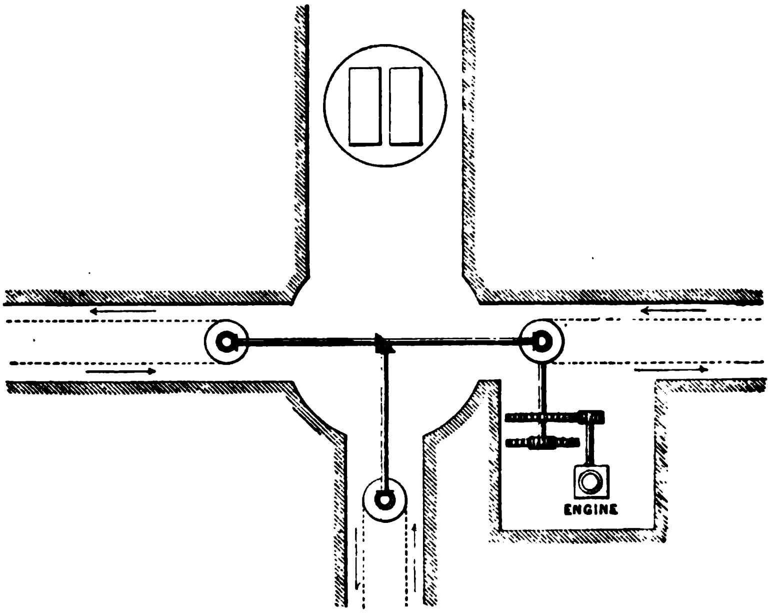


Fig. 328.—Endless Chain Haulage. Arrangement of Bevel Gearing from Hauling Engine to Roads leading from Pit Bottom.

Every principal chain has its separate pulley and upright shaft, with bevel gearing at the top, and catch-box to put in and out of gear. All of them may thus be at work together, or any of them thrown out of gear at will. The handles to work the catch-boxes may be placed at each wheel, or they may be all placed together in the engine-house or elsewhere. Branch lines form no obstacle to the successful working of the endless chain. There are two ways of working these. A double pulley may be placed opposite the branch similar to that shown in Fig. 327 for working No. 2 Endless Rope branch. The main chain passes round one pulley, and the branch chain takes its power from the other, which has a catch-box arrangement for throwing into and out of gear. Another and perhaps better way is to have three chains and three pulleys connected by gearing, as shown in Fig. 329. This leaves the landing-place clear for tubs.

The end of the tub on which the fork is placed is made to go last in starting up-hill, but in going down-hill it is placed first. Where the coal is not loaded up above the level of the top of the tub, the fork for the chain forms part of the tub, the groove for the chain being about an inch wide. One form is shown in Fig. 330, but where the coal stands high above the tub removable forks fitting into pockets in the tubs are used, and the length of the fork used is such as to enable the chain to stand sufficiently above the tub without touching the coal. A fork must be placed on both ends of the tub for this arrangement. In working round curves pulleys are placed to guide the chain, one for the full, and the other for the empty tubs. The road for the tubs, as they approach these pulleys, is made to rise, and just before reaching the pulley a fall commences. This causes the tub to leave the chain and pass round the curve under the pulley by its own momentum, and attaches itself again to the chain on the other side. Fig. 331 shows this.

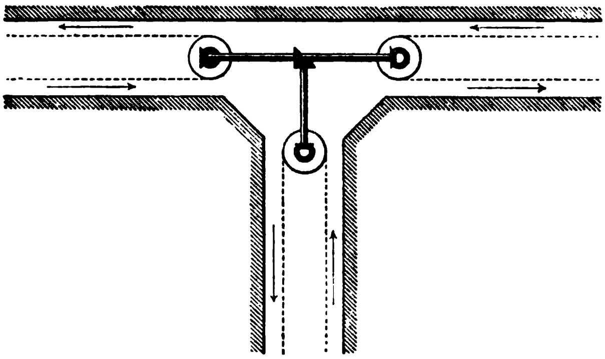


Fig. 329.—Endless Chain Haulage. Spur Gearing working a Branch from a Main Chain.

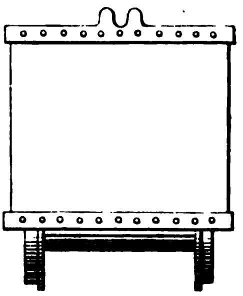


Fig. 330.—Iron Tub with Fork for Endless Chain Haulage.

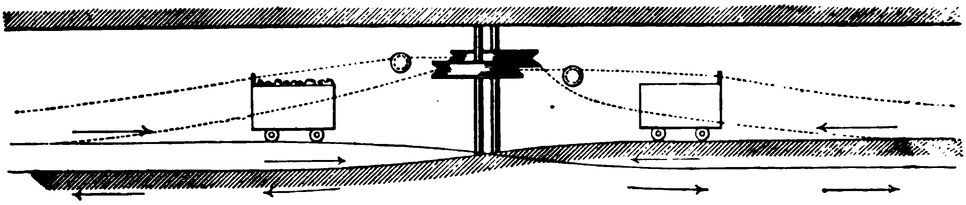


Fig. 331.—Endless Chain Haulage. Arrangement for Self-acting Curve.

Provision has been made to guard against the ill-effect of a broken chain. At the end of the plane, where a small guiding pulley is placed to direct the chain on to the driving-wheel, a movable catch may be placed, which keeps out of action whilst the chain is working properly, but if the chain breaks at that place the catch holds it. Again, claws are placed in the floor between the rails in both full and empty roads; if the chain breaks it falls down, and is held by these claws. An automatic stop-block may be placed at intervals in the full road if the plane rises towards the shaft, and in the empty way if the plane falls towards the shaft. The tubs ascending press the stop-block down, and after the tub has passed over it rises again in position to catch and hold any tubs rushing back owing to a broken chain. This block would be applicable only under certain conditions of road.

Care should be taken that the tubs are placed on the chain at such intervals as to ensure keeping the chain from touching the ground. No rollers are required for the endless chain; a regular gradient is unnecessary; a heavy dip at one part of the plane may be succeeded by a rise.

A plane may have many changes in its rate and direction of inclination, and where this is so, the energy arising from the counterbalancing effect of the undulating road is not all lost. It is quite possible, where there is a continual load upon the heavy part of a plane in favour of the full tubs, for the plane to self-act, even although some portions of the plane may be flat or dip in favour of the empty tubs. The first cost of fitting the road is heavier than for the Tail rope, as the chain costs more than the ropes, and a double line of rails is necessary. There are, no doubt, circumstances in which each of the different methods will recommend themselves.

Signalling.—Until recently the method of signalling on engine planes was the same as on the self-acting inclined planes, by means of signal wire and hammers and levers.

If the engine was underground, a hammer was fixed in the engine-house. Attached to the hammer was a lever, from the end of which the wire was carried along the engine-plane and supported by iron hooks or staples driven at frequent intervals into the road-posts. If the engine was on the surface, the hammer was placed at a point near the bottom of the shaft convenient to the attendant who received the signals, and by means of another signalling arrangement communicated the signals he received to the engineman. At the terminus of the engine-plane and at any points between its ends from which it was desired to signal, levers were attached to this wire. By pulling the levers the hammer was raised and then (by releasing the lever) allowed to fall on to a flat piece of iron, making a distinct signalling sound.

This method has now, for the most part, given way to the electric signal. In the best form of this method, two wires are carried along the engine-plane throughout its course. The wires are about 6 inches apart, and are supported parallel to each other by insulators fixed to the road-posts or to the sides at intervals of about 20 yards; but this varies according to circumstances.

In the engine-house at the pit bottom, or wherever it is desired to receive the signal, an electric bell is fixed and a battery sufficiently powerful to keep the apparatus well supplied with an electric current. Having a return wire, signals may be given from one end to the other, or from any intermediate point, but can only be received at the electric bell. Through any portions of the plane which are subject to falling water, insulated wire must be used.

At any point on the plane, a signal may be given instantaneously by the "run-rider." He has merely to bring the two wires together and rub them with his fingers, or connect the two wires by tapping them with a short iron rod. Contact is at once formed and the bell rings. It is as easy to give a signal at a long as at a short distance from the engine, and less effort is required to ring the bell than in the old wire and hammer system. The first cost for the wire is also less, as a light wire answers the purpose, while in cost of maintenance it compares favourably with the old system. Besides these advantages it is impossible to calculate the value of the convenience in signalling instantaneously from any point.

For convenience in repairing the road timbers, the wires should not be in one piece, but in short lengths of, say, 50 yards. The lengths are joined for ordinary working occasions, but may be quickly disconnected at any point of the engine-plane at night or whenever it may be necessary to renew timber in that part of the road.

A few questions and answers, bearing upon the subject of this chapter, will now be given, showing how to find the size of engines necessary to work engine-planes, &c.

> *Question* 50.—What sized hauling engines would be required to haul 100 tons per hour by direct haulage up an incline 1,000 yards long rising 1 in 6, steam pressure being (in the boilers) 50 lbs. per square inch?

Here assume that the average speed is six miles an hour and the drum 5 feet in diameter, and that the engines are to be a pair working on the first motion. Assume that the engines have a 30-inch stroke, and if this works out disproportionately to the size of the cylinders it can be adjusted afterwards. Six miles = 1,760 × 3 × 6 = 31,680 feet. 1,000 yards = 3,000 feet. To find the time occupied in making the journey one way, as 31,680 : 3,000 : : 60 min. $\frac{3{,}000 \times 60}{31{,}680} =$ 5·6 say 6 minutes, as the time occupied by the train in travelling one way = 12 minutes and say 4 minutes interval = 16 minutes for each journey = $\frac{60}{16}$ or $3\frac{3}{4}$ trains per hour $\frac{100 \text{ tons}}{3{\cdot}75} = 26\frac{2}{3}$ tons, or say 27 tons per train. Suppose each tub to carry 10 cwt. this means 54 tubs, and if each tub weighs 4 cwt. there are 27 tons of coal and 10 tons 16 cwt. or say 11 tons of tubs = 38 tons as the gross load. But the inclination is 1 in 6, therefore $\frac{38}{6} = 6\frac{1}{3}$ tons as the net load. To consider its weight, assume a steel rope weighing 7 lbs. per yard is used; 1,000 × 7 = 7,000 lbs. as the weight of the rope, but the net load on an incline of 1 in 6 would be $\frac{7{,}000}{6} = 1{,}167$ lbs. and the net load exclusive of friction becomes $6\frac{1}{3}$ tons = 14,187 lbs. representing the tubs and coal + 1,167 lbs. representing the rope = 15,354 lbs. acting at the circumference of a drum whose circumference is 15·7 feet. Add to this the friction. Experiments made to fix the co-efficient of friction for tubs at different times have given different results, varying from $\frac{1}{38}$th to $\frac{1}{28}$th of the gross load. Considering the friction of the rope on the rollers besides the wheels of the tubs, it will perhaps be prudent to adopt the highest co-efficient of friction, viz. $\frac{1}{28}$th of the gross load, and by adopting this it will be taking a proportion to cover friction of the rollers. The rope is 7,000 lbs., the tubs 11 tons or 24,640 lbs., the coal 27 tons or 60,480 lbs. = 7,000 + 24,640 + 60,480 = 92,120 lbs. total gross load: $\frac{92{,}120}{28} = 3{,}290$, the allowance for friction, which added to 15,354 = 18,644 lbs., the actual load upon a circumference of 15·7 feet. 18,644 × 15·7 = 292,710 the moment of load. The power is the pressure upon the piston, and its leverage is the double stroke of the piston 5 feet. Assuming $\frac{2}{3}$rds of the boiler pressure of 50 lbs. to be effective = 33 lbs. effective pressure. $\frac{292{,}710}{5 \times 33} = 1{,}774$ which gives theoretically the required piston area in square inches. To this add $\frac{1}{2}$ for general resistance and surplus power, making the required piston area 2,661, or say 2,670 square inches for one cylinder and $\frac{2,670}{2} = 1,335$ square inches for each cylinder of a pair. $\sqrt{\frac{1,335}{\cdot 7854}} = 41{\cdot}22$ as the diameter of the piston, so that a pair of engines will be required on the first motion with 41¼-inch cylinders and 30-inch stroke working a 5-foot diameter drum. But the cylinders and stroke are not proportioned to one another, and to find another proportion in which the area of cylinders × stroke will give the same power, proceed as follows:—Take the area of the cylinder at 1,335, and multiply this by the assumed stroke of 30 inches; 1,335 × 30 = 40,050. Now let the proportion of the stroke in the size, we shall find, be twice the diameter of piston.

Let x = the diameter in inches we wish to find.

Then $2\,x$ = the length of stroke in inches we wish to find.

Therefore $x^2 \times \cdot 7854 \times 2\,x = 40,050$ in order that the power may be equal to that of the size first ascertained.

$$x^2 \times 2\,x = \frac{40,050}{\cdot 7854}$$

$$2\,x^3 = \frac{40,050}{\cdot 7854}$$

$$x^3 = \frac{20,025}{\cdot 7854}$$

$x = 29{\cdot}45$ or, say, 30 inches, as the diameter of the piston, and $30 \times 2 = 60 = 5$ feet as the length of stroke. To find the piston speed, 6 minutes is the time occupied by the train in travelling 3,000 feet, therefore the speed of the train on the average is $\frac{3,000}{6} = 500$ feet per minute. The drum has a circumference of 15·7 feet, therefore $\frac{500}{15{\cdot}7} = 31\tfrac{3}{4}$ revolutions per minute; the piston travels 10 feet for one revolution, therefore $31\tfrac{3}{4} \times 10 = 317{\cdot}5$ feet per minute of piston speed, which is under a fair allowable speed. Now to look at the strength of the rope taken, viz., 7 lbs. per yard. The net load on the rope is 15,354 lbs. or 6·85 tons. By the rule already given to find the circumference of a steel rope, whose working load is 6·85 tons, $C = \sqrt{2{\cdot}4 \times 6{\cdot}85}$ = a little over 4 inches, and the weight of a 4-inch circumference steel rope is about 7 lbs. per yard.

> *Question* 51.—A pair of hauling engines, 24-inch cylinders and 4-foot stroke, with a boiler pressure of 45 lbs. per square inch, and working on the first motion, by direct haulage, a plane 2,000 yards long, dipping from the shaft 1 in 9. What quantity of coal can be dealt with by such an engine, and what size drum would you use?

Assuming the maximum piston speed of 400 feet per minute, and that the trains run at an average rate of 8 miles an hour on a single line of rails, and that ⅔ of the boiler pressure is effective, the power on the pistons would be $24^2 \times \cdot 7854 \times 30$ (the effective steam pressure) × 2 (for the two cylinders) = 27,143, and deducting ⅓ for frictional allowances = 27,143 − 9,048 = 18,095, travelling at 400 feet per minute. The tubs have to travel 8 miles an hour $= \frac{1,760 \times 3 \times 8}{60} =$ 704 feet per minute. While the piston travels $4 \times 2 = 8$ feet the crank travels $4 \times 3{\cdot}14159 = 12{\cdot}566$, say 12½ feet, and while the circumference of the drum travels 704 feet, the piston travels 400 feet, and the crank travels $\frac{400}{8} \times 12{\cdot}566$ = 628 feet. Therefore, as 628 : 704 :: 12·566 : 14·08 circumference of the drum

$= \frac{14{\cdot}08}{3{\cdot}14159} = 4{\cdot}48$ or $4\frac{1}{2}$ feet as the diameter. The load × 14·08 must equal the power. $\frac{18{,}095 \times 8}{14} = 10{,}340$ lbs., the load with which these engines are capable of dealing. To proportion this load so as to allow for weight of coals, weight of tubs, weight of rope and friction, proceed thus:—The steel rope will probably weigh about 5 lbs. per yard; $2{,}000 \times 5 = 10{,}000$ lbs., and this divided by the inclination $9 = 1{,}111$ lbs., which, deducted from the load, 10,340 lbs., leaves a balance of 9,229 lbs. This × 9, the inclination, = 83,061. Next allow $\frac{1}{28}$ of the gross load, viz., $83{,}061 + 10{,}000$ lbs. the rope $= 93{,}061$ lbs. for friction, $\frac{93{,}061}{28} = 3{,}324$ lbs. Deducting this from the balance of $9{,}229 = 5{,}905$. Multiply this by the inclination $9 = 53{,}145$. The tubs when empty will weigh from $\frac{1}{2}$ to $\frac{1}{3}$ of the load they carry, say $\frac{1}{3}$, then $\frac{53{,}145}{4} = 13{,}286$ lbs. as the weight of the tubs, and $53{,}145 - 13{,}286 = 39{,}859$ lbs. as the weight of coal. The speed of the rope is 704 feet per minute, and the distance being 2,000 yards = 6,000 feet, $\frac{6{,}000}{704} = 8\frac{1}{2}$ minutes each single journey will occupy, and $8\frac{1}{2} \times 2 = 17$ minutes each double journey, and allowing intervals amounting to 8 minutes, that means $\frac{60}{17 + 8} = 2{\cdot}4$ journeys per hour. The load of coal is 39,859 lbs. $\times\ 2{\cdot}4 = 95{,}661$ lbs. $\frac{95{,}661}{2{,}240} = 42{\cdot}7$ tons per hour. Showing that, with a pair of engines, having 24-inch cylinders, 4-foot stroke, running 400 feet per minute, and drum $4\frac{1}{2}$ feet in diameter, with a boiler pressure of 45 lbs. per square inch, 42·7 tons per hour could be dealt with from a distance of 2,000 yards, the inclination being 1 in 9, but as $4\frac{1}{2}$ feet is rather small, it would be better to use a 6-foot drum.

> *Question* 52.—What size and description of hauling engine would you erect to haul 370 tons of coal per day of $8\frac{1}{2}$ hours by the endless chain, up a plane 1,250 yards long, the average inclination of which is 1 in 27, being a rise for the full tubs, and tubs holding 6 cwt. of coal being used?

Supposing the tubs to be placed at intervals of 15 yards apart, $\frac{1{,}250}{15} = 83$ full and 83 empty tubs on the road. The weight of coal would be $83 \times 6 = 498$ cwt. = 24 tons 18 cwt., and as there are the same number of tubs on each line of rails they will balance each other. The load on the engine will, therefore, be 24 tons 18 cwt., and the friction of the tubs and chain and gearing.

On an inclination of 1 in 27, the actual work done by the hauling engine $= \frac{498}{27}$ = say 18 cwt. The friction of the tubs, assuming they weigh 3 cwt. each, and that the co-efficient of friction is $\frac{1}{28}$, will be $\frac{83 \times 2 \times 3}{28} = 18$ cwt. There would be $1{,}250 \times 2 = 2{,}500$ yards of iron chain, say of $\frac{5}{8}$ inch diameter, weighing 11 lbs. per yard = about 246 cwt. and $\frac{246}{28}$ = nearly 9 cwt. The friction for the coal is $\frac{498}{28} = 18$ cwt. This gives a total load upon the chain of 18 cwt. coal + 18 cwt. friction of coal + 18 cwt. friction of tubs + 9 cwt. friction of chain = 63 cwt., besides the resistance from the driving pulleys round which the chain has to pass. As these cannot well be calculated, allow 2 cwt., making 65 cwt. as the total load.

Now to arrive at the speed at which the chain must be worked. It is desired to draw 370 tons in 8½ hours to the shaft $= \frac{370 \times 2{,}240}{8\frac{1}{2} \times 60} = 1{,}625$ lbs. per minute. 1 tub holds 6 cwt. = 672 lbs. and $\frac{1{,}625}{672} = 2 \cdot 4$ tubs per minute, say 2½, as the rate of delivery from the chain to the shaft, and as they are placed 15 yards apart the speed of the chain must be $15 \times 2\frac{1}{2} = 37\frac{1}{2}$ yards, or say 113 feet per minute; $113 \times 60 = 6{,}780$ feet per hour, equivalent to $\frac{6{,}780}{5{,}280} = 1 \cdot 3$ mile an hour nearly. Now to fix on the size of engine. Assume that the driving-wheel is 3 feet in diameter, and that the stroke of the engine is 2 feet, and arrive at the proportion of gearing as follows; the speed of chain is 113 feet per minute; the driving-wheel has a circumference of $3 \times 3 \cdot 1416 = 9 \cdot 4248$. $\frac{113}{9 \cdot 4248} = 12$ revolutions per minute. The piston speed for an engine working continuously like this should not exceed 250 feet per minute, but take it at 200 feet per minute. With a stroke of 2 feet this would mean $\frac{200}{4} = 50$ revolutions per minute, so that the proportion of gearing would be as 50 is to 12, or about 4 to 1. The work done would be 65 cwt., at the rate of 1·3 mile an hour. $\frac{65 \times 112 \times 1 \cdot 3 \times 5{,}280}{60 \times 33{,}000} = 25$ horse-power, and assuming the engine yielded 50 per cent. of useful effect, the indicated horse-power would be 50 or 25 for each engine. This would be equal, supposing the effective steam pressure to be 30 lbs., to $\frac{25 \times 33{,}000}{30 \times 200} = 137 \cdot 5$, and $\sqrt{\frac{137 \cdot 5}{\cdot 7854}}$ $= 13 \cdot 23$ as the diameter in inches of the cylinders, and 2 feet as the length of stroke.

Question 53.—What size hauling engines would you place underground to work an incline by tail rope whose rates of dip from the shaft are as follow: —For the first 200 yards, the dip is 1 in 28, for the next 400 yards it is 1 in 30, and for the next 500 yards which is the end of the plane it is 1 in 31. The coal to be brought out to be 1,000 tons in 8 hours.

$$\begin{array}{rrlrl}
 & 200 \text{ yards} & 1 \text{ in } 28 = \frac{200}{28} = & 7\frac{1}{4} & \\
 & 400 \quad ,, & 1 \text{ in } 30 = \frac{400}{30} = & 13\frac{1}{3} & \\
 & 500 \quad ,, & 1 \text{ in } 31 = \frac{500}{31} = & 16\frac{1}{8} & \\
\hline
\text{Total} & 1{,}100 & & 37 & \text{and } \frac{1{,}100}{37} = 30;
\end{array}$$

making the average gradient, therefore, 1 in 30.

Assume that the average speed is 8 miles an hour, and that the engines are to be a pair with 6-foot drums. Assume also that the engines have a 4-foot stroke, and that the gearing is 1 to 1. 8 miles $= 1{,}760 \times 3 \times 8 = 42{,}240$ feet. 1,100 yards = 3,300 feet. To find the time occupied in making the journey one way; as 42,240 : 3,300 : : 60 minutes : 4·7 say 5 minutes as the time occupied by the train in travelling one way = 10 minutes, and say 5 minutes interval = 15 minutes for each trip $= \frac{60}{15} = 4$ trains per hour; $\frac{1{,}000}{8 \times 4} = 31\frac{1}{4}$ tons per train.

Suppose each tub to hold 10 cwt., this means 63 tubs, and if each tub weighs 4 cwt. it means $31\frac{1}{4}$ tons of coal, and $63 \times 4 = 252$ cwts. or 12 tons 12 cwts. of tubs = say 44 tons as the gross load. But the average inclination being 1 in 30, $\frac{44}{30} = 1\frac{1}{2}$ tons nearly as the net load. The ropes to some extent balance each other, but as it is usual to make the main rope stronger than the tail rope take this into account. Supposing a $2\frac{1}{4}$ lb. per yard steel rope to be used for the main and a $1\frac{1}{2}$ lb. per yard for the tail rope, then $2\frac{1}{4} - 1\frac{1}{2} = \frac{3}{4} \times 1{,}100 = 825$ lbs. as the weight of the rope, the net load of which would be $\frac{825}{30} = 28$ lbs. nearly. The net load exclusive of friction is $1\frac{1}{2}$ tons = 3,360 lbs. representing the tubs and coal + 28 lbs. representing the rope = 3,388 lbs. acting at the circumference of a drum whose circumference is $6 \times 3{\cdot}1416 = 18{\cdot}849$ feet. But add to this the friction. The ropes are $1{,}100 \times 2\frac{1}{4} + 1{,}100 \times 1\frac{1}{2} = 1{,}100 \times 3\frac{3}{4} = 4{,}125$ lbs., the tubs 252 cwts. or 28,224 lbs., the coal $31\frac{1}{4}$ tons or 70,000 lbs. $4{,}125 + 28{,}224 + 70{,}000 = 102{,}349$ lbs. total gross load. $\frac{102{,}349}{28} = 3{,}655$ lbs., the allowance for friction which added to 3,388 = 7,043 lbs. the actual load upon a circumference of 18·849. $7{,}043 \times 18{\cdot}849 = 132{,}754$ the moment of load. A 4-foot stroke and an effective steam pressure of 30 lbs. = $\frac{132{,}754}{4 \times 2 \times 30} = 553$, the theoretical required piston area in square inches. Add $\frac{1}{2}$ for general resistance, making the required piston area 830 square inches for one cylinder. $\frac{830}{2} = 415$ square inches for each cylinder of a pair. $\sqrt{\frac{415}{{\cdot}7854}} = 22{\cdot}98$, or say 23 inches as the size of the pair of engines required, with a 4-foot stroke which is pretty well in proportion and 6-foot drums. Now to test the piston speed. The train travels 3,300 feet in 5 minutes = $\frac{3{,}300}{5} = 660$ feet per minute. The drums have a circumference of 18·849 feet, therefore $\frac{660}{18{\cdot}849} = 35$ revolutions per minute, and $35 \times 8 = 280$ feet per minute of piston speed, which is well under a fair allowable speed. The net load on the rope is 3,388 lbs. or 1·6 ton $\sqrt{2{\cdot}4 \times 1{\cdot}6} = 2$ inches nearly circumference, and the weight of a 2-inch circumference steel wire-rope is $4\frac{1}{4}$ lbs. per fathom or $2\frac{1}{8}$ lbs. per yard, so that the rope chosen weighing $2\frac{1}{4}$ lbs. per yard is quite strong enough for the purpose.

> *Question* 54.—A train of 10 tubs, ascend an incline, each tub (with coal) weighs 1 ton with a rise of $4\frac{1}{2}$ inches per yard, what is the power required and the strain on the rope?

Assume that the friction of the tubs is $\frac{1}{28}$th of their weight. A rise of $4\frac{1}{2}$ inches per yard $= \frac{36}{4\frac{1}{2}} = 1$ in 8, 10 tubs $\times 2{,}240 = 22{,}400$. $\frac{22{,}400}{28} = 800$ lbs. necessary to overcome friction, and $\frac{22{,}400}{8} = 2{,}800$ lbs. necessary to overcome gravity, which would be the power without considering friction; the amount of strain on the rope would be therefore $800 + 2{,}800 = 3{,}600$ lbs. = 1 ton 12 cwts. 0 qr. 16 lbs. It often happens that $\frac{1}{70}$th of the weight is taken for the friction, if that were adopted it would mean $\frac{22{,}400}{70} = 320$ lbs. for friction. 2,800 lbs. for gravity as before = 3,120 lbs. total = 1 ton 7 cwts. 3 qrs. 12 lbs., but the former allowance is preferable as being safer.

Question 55.—Are hauling engines sometimes worked with compressed air as a motive power?

Yes, and it has great advantages over steam under certain conditions, as the latter is not a convenient motive power to convey far into a mine. An engine, whether for hauling, working rock-drills or coal-cutting machinery, worked by compressed air can be placed at any point in the workings, near the shaft or far away from it. It has also been successfully applied as a motive power to supply locomotives running underground, in which case the engine carries compressed air in a reservoir. The exhaust, although hardly appreciable in well ventilated pits, is of some use in improving the ventilation in the workings, and the pipes along which the compressed air is conveyed are cool and last long.

The disadvantages in using compressed air are many. First, it requires extensive plant on the surface, a steam-engine being necessary in the first instance to produce the compressed air. A very low proportion of the work done in the steam engines is given by the engines using the compressed air, not more than from 25 to 30 per cent., so that 2 or 3 times as much fuel is consumed as would be necessary if steam were used direct. The reasons for this low useful effect are, first, the fact of having a second engine to employ with its friction to overcome, and loss of heat in compressing the air, which passes through the compressing cylinders; secondly, the leakage of the valves and pistons, and the friction of the air in being carried from the receiver along the pipes; thirdly, little expansion is obtained out of the compressed air; and fourthly, during compression the temperature of the air is increased, and therefore its bulk; on leaving the cylinder its bulk is reduced again, so that the work of compression is performed on a larger bulk of air than work is obtained from.

The compressed air on being liberated absorbs the heat from the surrounding air, causing a great degree of cold, and, unless dry air be used, ice is formed in the cylinder and the exhaust pipe. The means of producing the compressed air and applying it are as follow:—A horizontal steam-engine is so placed that the continuation of its piston-rod through the back cylinder cover forms the piston-rod for the cylinder of the compresser, the piston of the compresser being rather larger than the piston of the steam cylinder. By an arrangement of valves in the air cylinder (which is usually surrounded by water to keep it cool; the water absorbing the heat generated during compression and conducting it through the metal), the air enters the cylinder behind the piston, and is forced out of the cylinder in front of the piston through pipes into a receiver of large dimensions loaded to the required pressure, from which pipes convey it down the pit and along the workings to the engine to be supplied with compressed air.

There are circumstances in which no other motive force can be so well applied, but on economical grounds, wherever practicable, steam should be employed direct.

Question 56.—What are compound hauling engines?

Compound engines have two or more cylinders, and the same steam works in all the cylinders successively and expansively. Usually only two cylinders are used, one having the smaller diameter being the high-pressure cylinder into which the steam as it leaves the boiler passes, and after completing the stroke, then into the other, which is the larger or low-pressure cylinder. The steam is used expansively in both cylinders, and leaves the low-pressure cylinder only a little above the atmospheric pressure. A receiver is placed between the high and low-pressure cylinders, to equalise the back pressure on the high-pressure piston and the initial pressure on the low-pressure piston. The low-pressure cylinder may have a valve applied to it so that high-pressure steam can be admitted to it at the first few revolutions when the engines have the heaviest lift.

Compound engines have been applied more to work pumps than for hauling and winding.

> *Question* 57.—It is intended to erect a pair of coupled engines, working direct without gearing to haul 8 full coal tubs each weighing 11 cwt. (load and tub) up an inclined plane 560 yards long and rising 1 in 7. The desired rate of speed is 10 miles an hour. What size and stroke should the engines be and how many revolutions per minute ? The drum of the engine is 3 feet diameter, and the rope not allowed to overlap itself on it. The boiler pressure of steam is 35 lbs.

A rope of steel wire having a circumference of 1½ inches would weigh 1 lb. per yard ; the total weight would be 560 lbs. The whole weight on the engine due to gravitation would be 8 tubs @ 11 cwt. $= 8 \times 11 \times 112 = 9{,}856$ lbs. + 560 lbs. $= 10{,}416$ and $\frac{10{,}416}{7} = 1{,}488$ lbs. ; that due to friction inclusive of rollers, &c. would be $\frac{10{,}416}{28} = 372$ lbs. Total load $= 1{,}488 + 372 = 1{,}860$ lbs. The diameter of drum being 3 feet the circumference is 9·4248 feet. $1{,}860 \times 9{\cdot}4248 = 17{,}529$ the moment of load. An 18-inch stroke and a boiler pressure of 35 lbs. $= \frac{35 \times 2}{3} =$ say 23 lbs. effective pressure ; $\frac{17{,}529}{1\frac{1}{2} \times 2 \times 23} = 254$ the theoretical piston area, to which add $\frac{1}{2} = 254 + 127 = 381$ for one cylinder and $\frac{381}{2}$ say 191 square inches for each of a pair $\sqrt{\frac{191}{{\cdot}7854}} = 15{\cdot}59$ as the diameter of a pair, say 15½ inches. The rate at which the trams are hauled is 10 miles an hour : $\therefore \frac{10 \times 1{,}760 \times 3}{60 \times 9{\cdot}4248} = 93{\cdot}37$ say 93⅓ revolutions of a drum per minute, and the drum makes one revolution for three feet of piston travel $= 93\frac{1}{3} \times 3 = 280$ feet as the piston speed per minute—quite an allowable speed. To find the horse-power of these engines $\frac{15\frac{1}{2}^2 \times {\cdot}7854 \times 2 \times 280 \times 23}{33{,}000} = 74$ horse-power.

If as stated the rope is not to overlap itself on the drum which in hauling up a plane 560 yards long makes $\frac{560 \times 3}{9{\cdot}4248} = 178$ revolutions, and the rope is about ½-inch diameter, therefore $178 \times \frac{1}{2}'' = 89$ inches or 7 ft. 5 in., so that the drum would have to be 7 feet 6 inches in length, if 3 feet in diameter, which is rather an unusually small size.

> *Question* 58.—The weight of the tubs, coal, rope, &c., which is pulled up a slant whose angle is 26 degrees, amounts to 70 cwt. What is the working load of the rope, friction $\frac{1}{70}$?

As there are 57·3° in an arc whose length = radius, approximately for this purpose the value of a rise of 26° in inches per yard may be expressed by taking a circle whose radius is one yard or 36 inches. Multiply this radius by 26° (the number of degrees) and divide by 57·3° and get approximately the rise per yard, thus $\frac{36 \times 26}{57{\cdot}3} = 16$ inches rise per yard $= \frac{36}{16} = 1$ in 2·25. Therefore the force of gravitation on a weight of 70 cwt. is $\frac{70}{2{\cdot}25} = 31{\cdot}1$ cwt.

The force due to friction would be $\frac{70}{70} = 1$ cwt., hence the working load of the rope would be $31{\cdot}1 + 1 = 32{\cdot}1$ cwt. or $32\frac{1}{9}$ cwt.

Question 59.—At what gradient would the full tubs hold the empties in suspension, the set to consist of 8 full tubs weighing 11 cwt. each and 8 empties weighing 3 cwt. each, friction $\frac{1}{70}$th?

Equilibrium will ensue when the sum of the friction of the two loads is equal to the difference of their gravity on the incline. Supposing in the question asked that the weight of each load respectively includes the weight and friction of rope, friction of rollers, &c., connected with it; then—

Let 1 in x denote the gradient;

$$\text{then } \frac{11 \times 8}{x} - \frac{3 \times 8}{x} = \frac{(11 \times 8) + (3 \times 8)}{70};$$

$$\therefore \frac{88}{x} - \frac{24}{x} = \frac{112}{70}$$

$$\text{and } \frac{64}{x} = 1{\cdot}6$$

$$64 = 1{\cdot}6\,x$$

$$x = \frac{64}{1{\cdot}6} = 40.$$

Therefore the gradient would be 1 in 40. Put as a formula for the above rule, $(F - E)\,R = (F + E)\,G$, where F stands for the full loads, E for the empty ones. R the denominator of the fraction of the friction, the numerator of which is always 1, G the denominator of the fraction of the gradient, the numerator of which is always 1.

Question 60.—What ought to be the gradient of a horse road when the loaded train consists of six tubs 18 cwt. each, and six empties of $4\frac{1}{2}$ cwt. each, friction $\frac{1}{70}$th, so that the resistance may be equal both ways?

In this case resistance is equal when the friction of the full load, less its gravitation, is equal to the friction of the empty load, plus its gravitation.

Let 1 in x denote the gradient;

$$\text{then } \frac{108}{70} - \frac{108}{x} = \frac{27}{70} + \frac{27}{x}$$

$$\therefore \frac{27}{x} + \frac{108}{x} = \frac{108}{70} - \frac{27}{70}$$

$$\frac{135}{x} = \frac{81}{70};$$

$$x = 116\tfrac{2}{3},$$

the gradient therefore is 1 in $116\frac{2}{3}$. Express this rule by a formula, using the same expressions as in the last example, as follows—

$$\frac{F}{R} - \frac{F}{G} = \frac{E}{R} + \frac{E}{G} \therefore (F + E)\,R = (F - E)\,G.$$

Question 61.—What are the usual methods of determining the size of the high and low-pressure cylinders of compound engines, and how is the horse-power of such engines calculated?

It is necessary, in the first place, to determine the rate of expansion in compound engines, then if A = area of low-pressure cylinder, a = area of high-pressure cylinder, E = rate of expansion $= \frac{A\,L}{a\,l}$, L = length of stroke in feet,

& l = distance travelled by piston before steam is cut off in the high-pressure cylinder. The best proportion for the two cylinders is when $a = A \div \sqrt{E}$, so if the steam is to be expanded 4 times $a = \cdot 5\ A$, and, as the formula assumes the steam being also cut off at ·5 of the stroke in the small cylinder, then the initial volume × 4 = the final volume $a \times \cdot 5\ L \times 4 = A \times L$. To find the horse-power, each cylinder should be indicated to find the mean effective pressure in each, then treat each as a separate engine and add the results together.

> *Question* 62.—What is the horse-power of a pair of hauling-engines, the diameter of cylinders being 16 inches, stroke 30 inches, number of strokes per minute 25, pressure of steam 35 lbs.? The engine is geared as follows: The crank is connected to a shaft containing a pinion with 15 teeth which drives a spur-wheel having 70 teeth. On the same shaft containing this spur-wheel is a pinion with 20 teeth which drives a spur-wheel on the drum shaft with 80 teeth.

The effective horse-power of an engine is only slightly reduced owing to the additional friction which has to be overcome. Of course a greater weight would be lifted but for the gearing, at a sacrifice of speed; for instance, whilst the crank-shaft made one revolution the 70-toothed spur-wheel would only make $\frac{15}{70}$ths of a revolution, so that whilst the 70-toothed spur-wheel makes one revolution the crank would make $\frac{70}{15} = 4\frac{2}{3}$ revolutions, and whilst the pinion on the spur-wheel shaft makes one revolution the spur-wheel on the drum shaft only makes $\frac{20}{80}$ or $\frac{1}{4}$th of a revolution. Therefore whilst the drum revolves once the crank revolves $4\frac{2}{3} \times 4 = 18\frac{2}{3}$ times.

The horse-power of the engines would be

$$\frac{16^2 \times \cdot 7854 \times 2 \times 2\frac{1}{2} \times 2 \times 25 \times 35}{33{,}000} = 53{\cdot}312 \text{ H.-P.}$$

> *Question* 63.—What sized hauling-engines or horse-power would be required to draw 120 tons per hour up an incline 1,500 yards long, gradient 1 in 8, at a speed of 4 miles per hour? Could the engines be geared to the hauling drum by spur-wheels in the proportion of 5 to 1?

This is a large quantity to bring up the incline stated, and if direct haulage be adopted, necessitating a double line of rails throughout, or the laying of the rails may be by a three-plate way above meetings and a single one below after the manner of a self-acting incline; but this would be possible only if there are no branch roads and the incline is straight throughout its course. A pair of engines would of course be necessary, and assume the drums to be 6 feet. A speed of 4 miles an hour = 1,760 × 3 × 4 = 21,120 feet. Length of plane 1,500 × 3 = 4,500 feet, $\frac{4{,}500 \times 60}{21{,}120} = 13$ minutes nearly, as the time occupied by the trains travelling, and as no time must be wasted in order to get up this quantity, allow an interval of 2 minutes to change the trains at both ends, &c., and one train would arrive at the shaft every 15 minutes, $\therefore \frac{60}{15} = 4$ trains per hour, and $\frac{120}{4} = 30$ tons to be sent per train. Suppose each tub to carry 10 cwt., then 60 tubs are required, and say each tub weighs 4 cwt., there are 30 tons of coal, but as the two trains of tubs will balance each other, they need not be considered, except for friction. Then a steel rope weighing about 8 lbs. per yard would be necessary, 1,500 × 8 = 12,000 lbs. 30 tons = 67,200 lbs., which, added to 12,000 = 79,200 and $\frac{79{,}200}{8} = 9{,}900$ lbs. as the load due to gravity. For the friction we have 120 tubs at 4 cwt. each = 480 cwt. = 53,760 lbs., and

$\frac{53,760 + 79,200}{28} = 4,749$ lbs. the allowance for friction. $9,900 + 4,749 = 14,649$ lbs. as the actual load upon a circumference of 18·849. $14,649 \times 18{\cdot}849 = 276,119$ the moment of load. Assuming a 5-foot stroke and 30 lbs. effective steam pressure $\frac{276,119}{10 \times 30} = 920{\cdot}4$ theoretical piston area, to which add $\frac{1}{2}$, $920{\cdot}4 + 460{\cdot}2 =$ say, $1,381 = 691$ for one cylinder and $\sqrt{\frac{691}{{\cdot}7854}} = 29{\cdot}66$, or say 30 inch cylinders. The size of the engines is not affected by the gearing, this would only affect the piston speed maintained in doing the work. To consider the possibility of gearing the engines, the drums are to be 6 feet in diameter and have a circumference of 18·849 feet. The speed of the ropes is $\frac{4,500}{13} = 346$ feet per minute, and therefore the drums revolve $\frac{346}{18{\cdot}849} = 18{\cdot}4$ times per minute. The piston speed must not exceed 400 feet per minute, and as the stroke is 5 feet, $\frac{400}{5 \times 2} = 40$ revolutions of the engine. The proportion of gearing must therefore not exceed 18·4 to 40 or say 2 to 1, and it would be better not to use gearing at all with 6-foot drums, in which case the piston speed would be $5 \times 2 \times 18{\cdot}4 = 184$, thus allowing a margin for occasional quicker speed following delays through accident.

Question 64.—How do you proceed to calculate the friction of tubs?

By allowing the tubs to run down an incline with a regular gradient, taking the length of the incline and noting the time occupied by the tubs in traversing it. The gradient must, of course, be ascertained if not known, and the experiment may be tried on either full or empty tubs.

Resistance due to friction $= \frac{H}{D} - \frac{D}{T^2 \times 16\frac{1}{12}}$ where H = height of plane in feet, D = length of plane in feet, T = time in seconds taken by tub to traverse the distance D.

Supposing a tub traversed a plane 150 feet long and having a fall between the two ends of 5·74 feet in 20 seconds, the resistance due to friction is found thus:—
$\frac{5{\cdot}74}{150} - \frac{150}{20^2 \times 16\frac{1}{12}} = {\cdot}01495$, so that if the loaded tub weighed 1,804 lbs., $1,804 \times {\cdot}01495 = 26{\cdot}97$ lbs. and $\frac{26{\cdot}97}{1,804} = \frac{1}{66{\cdot}9}$ say $\frac{1}{70}$th as the co-efficient of friction.

If an empty tub traversed the same plane in 23 seconds and the tub weighed 688 lbs., then $\frac{5{\cdot}74}{150} - \frac{150}{23^2 \times 16\frac{1}{12}} = {\cdot}02063$ and $\cdot02063 \times 688 = 14{\cdot}2$ lbs. and $\frac{14.2}{688} = \frac{1}{48{\cdot}45}$ say $\frac{1}{48}$th as the co-efficient of friction.

Another way of finding the co-efficient of friction is by placing the tub on a perfectly level piece of the road laid in the ordinary way, and attaching one end of a cord to the tub, the other with a weight at the end is passed over a pulley, and whatever weight is found sufficient to keep the tub in motion without accelerating its speed is taken as the amount of friction, and this weight divided by the weight of the tub operated on gives the co-efficient of friction.

Another method consists in starting a tub down a piece of road which becomes gradually flatter and observing where the tub comes to rest. The average gradient between the starting and resting points shows the ratio of the friction to the weight. If a tub continues to run on $\frac{1}{2}$ an inch per yard dip or 1 in 72, the friction is under $\frac{1}{72}$ of its weight; on 1 inch to the yard or 1 in 36 the friction is under $\frac{1}{36}$th of its weight.

> *Question* 65.—What gradient would be necessary for a self-acting incline whose length is 300 yards, six tubs being sent on either side, the empty tubs weighing 5 cwt. each, and 9 cwt. of coal being sent in each, a steel rope of $1\frac{1}{2}$-ins. circumference being used, the friction of the loads being taken at $\frac{1}{50}$th of the weight, and the friction of the rollers at $\frac{1}{25}$th of their weight.

Ascertain by the rule already given the gradient at which the full and the empty trains just balance each other, then it is evident that a slight increase in the gradient will be sufficient to give them motion.

The full train is $(9 + 5) \times 6 \times 112 = 9{,}408$ lbs. The empty train is $5 \times 6 \times 112 = 3{,}360$ lbs. A $1\frac{1}{2}$-inches circumference steel rope weighs nearly 1 lb. per yard. On 300 yards = 300 lbs. Rollers, say 30 @ 12 lbs. = 360 lbs. The friction then on the loads and tubs will be $\frac{9{,}408 + 3{,}360}{50} = 255{\cdot}3$ lbs., that on the rope and rollers will be $\frac{300 + 360}{25} = 26{\cdot}4$. $255{\cdot}3 + 26{\cdot}4 =$ say 282 lbs. The full load is 9,408 lbs. and the empty load 3,360 + the 300 lbs. of rope on it = 3,660. The difference in gravity therefore is $9{,}408 - 3{,}660 = 5{,}748$ lbs., and since equilibrium will ensue when the sum of the friction of the two loads is equal to the difference of their gravity, $5{,}748 \text{ lbs.} \times \frac{1}{G} = 282 \text{ lbs.} \therefore \frac{1}{G} = \frac{282}{5{,}748} = \frac{1}{20{\cdot}4}$, therefore the gradient would be 1 in 20·4, for the full and empty trains to balance each other, and it therefore should be 1 in 18 or 2 inches per yard to overcome the resistance of the drum and give motion to the trains. If the gradient is not quite uniform the heaviest part should be at the top.

> *Question* 66.—An engine draws 10 tubs of coal up an incline 100 yards long whose gradient is 1 in 9, at an average speed of 5 miles an hour, what will the same engine draw on a level road of the same length? A steel-wire rope weighing 7 lbs. per fathom is used.

Suppose the loaded tubs to weigh 10 cwt. each, $10 \times 10 \times 112 = 11{,}200$ lbs. For the rope $100 \times 3\frac{1}{2} = 350$ lbs. $\frac{11{,}200 + 350}{9} = 1{,}284$ lbs. due to gravity. Take $\frac{1}{28}$th the gross load for friction, $\frac{11{,}200 + 350}{28} = 413$ lbs. due to friction. The load would be $1{,}284 + 413 = 1{,}697$ lbs. travelling 5 miles an hour, $\frac{1{,}697 \times 5 \times 5{,}280}{60 \times 33{,}000} = 22{\cdot}63$ horse-power. Of course the engine would require to have double this horse-power, but for the purpose of the comparison made, that need not be considered. To ascertain how many tubs an engine of the same size will draw along a level road 100 yards long at the rate of 5 miles an hour, the work of the engine per minute would be $22{\cdot}63 \times 33{,}000 = 746{,}790$ lbs., but a good deal of the work would be devoted to overcoming friction. A tail rope would be necessary on the level road, which would make double the amount of friction for rope. Per minute it would be

$\frac{700 \times 5 \times 5{,}280}{28 \times 60} = 11{,}000$. $746{,}790 - 11{,}000 = 735{,}790$ lbs. The work due to friction on each tub of coal per minute would be $\frac{1{,}120 \times 5 \times 5{,}280}{28 \times 60} = 17{,}600$, $\therefore \frac{735{,}790}{17{,}600}$ = nearly 42 tubs of coal the engine under these circumstances would take out, but if the co-efficient of friction be taken at $\frac{1}{70}$th of the gross load, as is frequently done, the difference between what would be done on an incline of 1 in 9 and a level road of equal length would be more marked.

Question 67.—What should be the diameter of the low-pressure cylinder of a compound hauling-engine, if that of the high-pressure cylinder is 18 inches, the initial pressure of the steam being 147 lbs. above that of the atmosphere and exhausting at 3 lbs. above the atmosphere, the length of stroke being 27 inches? At what point in the stroke would the steam be cut off?

Here the absolute initial pressure of the steam is $147 + 15 = 162$ lbs., and the pressure on leaving the low-pressure cylinder is $3 + 15 = 18$ lbs. The rate of expansion therefore is as 18 : 162, or 9 times.

By the formula $a = A \div \sqrt{E}$, therefore $18^2 \times {}^{\cdot}7854 = A \div \sqrt{9}$ $\therefore$ $254{\cdot}469 = \frac{A}{3}$ $\therefore$ $A = 254{\cdot}469 \times 3 = 763{\cdot}407$; and therefore the diameter of the low-pressure cylinder is $\sqrt{\frac{763{\cdot}407}{{}^{\cdot}7854}} = 31{\cdot}17$ inches. The expansion being 9 times, the steam is cut off at $\sqrt{9} = \frac{1}{3}$rd of the stroke.

CHAPTER X.

DRAINAGE.

Winding Water up Shafts—Lifting and Forcing Pumps—Making Pipe Joints Watertight—Balance Bobs—The Windbore—Clack-piece—Fish-piece—The Working Barrel—Action of the Pumps—Water Speed in Pipes—Construction and Method of securing Pumps—Preserving Pipes from the Action of Mineral Water—Pump Rods: Their Material; Method of Joining; Steadying; Safety Catches—Attachment of Bucket to Spears—Hanging Clack and Bucket Doors—General Arrangement of Lifting and Forcing Sets of Pumps—Determination of Weight Necessary for Balance Bobs—Use and Action of the Air Vessel—Bunton and Plank Brattices to form compartment in Shaft for Pumps—Arrangement for a Sinking Set of Pumps—Sliding Suction for Sinking Set—Messrs. Thornewill and Warham's Details of Pump Work—The "Deane" Sinking Pumps—The Cornish Pumping Engine—The Cornish Double-beat Valve—Davey's Differential Valve Gear for Cornish Engine—Davey's Compound Differential Pumping Engine—Relative Advantages of placing Pumping Engines Above and Underground—Steam Pumps—The Compound Differential Engine as arranged underground—The Worthington Pumping Engine—Compressed Air for underground Pumping Engines—Hydraulic Pumps—Moore's Hydraulic Mine Pump—Wire Rope Systems of Pumping—Electrical Pumping Plant at the Trafalgar Colliery—Syphons for Drainage of Underground Roadways—Memoranda—Powers of Engines for given work—The Pulsometer, its Action and Use—Pulsometer arrangement for Draining Underground Workings—Calculation of Contents of Water Barrels.

THE water which finds its way into the underground workings must be removed. In the case of an adit driven at a proper rise and having no workings to the dip, the water will run out without trouble; but in the case of a pit with workings far enough beneath the surface, the water is removed either by pumps placed in the shaft, or by tanks placed in the cage, to be used when the pit has done drawing coals for the day. Unless the quantity is very small indeed, it is found an advantage to have pumps, for with only 20 galls. per minute to be removed it would take more than 6 hours winding, lifting one ton at a time at the rate of 20 tons per hour, giving 3 minutes to fill, wind the cage, and empty. There are two kinds of pumps used—the *lifting* and the *forcing*. In the lifting pumps, a bucket works in the working barrel, the part of the pump below the bucket is called the "suction," and above it the lift. The suction should not exceed from 20 to 24 feet in height. The height of the lift, limited by the strength of the material and the weight of the spears, does not usually exceed 50 fathoms. Spears or pump-rods connect the bucket working in the barrel with the beam of the engine, and as they are inside the pipes conveying the water up the shaft, they reduce the area of these for the water, and so increase the friction. Lifting pumps are not suitable to great depths on account of the wear and tear to the leather rings forming the packing of the bucket, necessitating frequent change, and costing a considerable amount for repairs. In the forcing or plunger pumps the plunger pole, or ram, works through a stuffing box into a plunger case of bored cast-iron, and at every down stroke the water is forced upwards through an upper clack into the column of pipes above, and these pipes in the plunger pumps are quite clear for the ascent of the water.

The advantage of the forcing method is that there is less wear and tear, the hemp packing of the stuffing box is preferable to the leathers of the bucket, as giving less friction and being more durable. It is not necessary to remove the ram itself to do the repairs, but in the lifting pumps the bucket must always be withdrawn from the working barrel when it is necessary to repair it. The spears of the plunger pumps are exposed to view in the shaft, and in their descent their weight is sufficient to force the water before them. In deep mines, where the large moving parts are heavier than the column of water raised, it is necessary to counterbalance the extra weight by balance bobs placed at the surface or at intervals in the pit, or both. The joints between the various lengths of pipes and that between the door and clack-piece are made tight by thin layers of some soft material, such as tarred flannel, which is wrapped round a ring of lead or wrought-iron, screw bolts and nuts being employed for tightening them at the flanges.

In both kinds of pumps the bottom piece is called the windbore, or suction piece, and has a number of holes in the bottom, covered by the water in the sump of the pit or cistern in which it may be placed, and these holes are made of such a size as to prevent pieces of wood or stone from entering the windbore with the water. As a further precaution the windbore is sometimes surrounded by a gauze cage. The next is the clack-piece, provided with a door, and contains the clack, which is a kind of valve resting in its seat. The clack may be taken out for repairs through the clack-door, and if there is any risk of this being at times impracticable, it is made with a bow on the top, so that a hook, called a fish-piece, may be attached to the end of the spears in place of the bucket and passed down the column and secured to the bow of the clack, which is thus lifted out of its seat and up the column of pumps. The fish-piece, shown in Fig. 332, is frequently used during sinking operations. When the fish-piece passes through the clack bow, the protruding pieces at the sides are forced in, and after it has gone further downwards they are again forced outwards by the springs shown on the drawing. On the fish-piece now being raised these protruding pieces project beyond the bow and lift it with them.

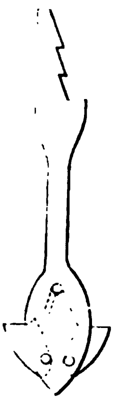


Fig. 332.—FISH-PIECE.

The next piece above the clack-piece is the working barrel, which, in the case of a lifting set, has the bucket working in it, and attached to the bucket are the spears extending upwards inside the column of pumps to the beam of the engine, or to a set-off or "apron" attached to the main spears working other sets of pumps in the same shaft. At the top of the working barrel is the bucket door-piece, by removing the door of which the bucket may be taken out for repairs. As the rods descend the bucket moves down the working barrel, the clack-valves fall and prevent the water from descending again to the windbore, and at the same time the bucket-valves open and the water, except that displaced by the bucket and rod, remains in the working barrel.

At the upstroke of the engine the bucket-valves close, and the water which stood in the working barrel at the down stroke is lifted on top of the bucket up the pumps, whilst, owing to the pressure of the atmosphere on the water in the sump, the water follows the course of the bucket in its ascent, and fills the working barrel. If the holes of the windbore are not covered by a foot or two of water, air will pass through them. This is unavoidable during sinking operations, for the pump must keep the water down to the lowest point, and it therefore often works partly on air. In all permanent pumps, however, care must be taken in working the engine that this does not occur.

In the sketch of a lifting and forcing set of pumps shown in Fig. 333, the necessity of two clack-pieces, with their clacks and doors in the plunger set of pumps, is evident. As the ram descends the bottom clack closes, the water passing through the upper clack to the column of pipes above. As it ascends the bottom clack opens, and the water from the cistern follows its course, the water in the column of pipes by its weight at the same time closing the upper valve. The diameter of pipes which form the rising main and the thickness of metal in the pipes depend upon the quantity of water to be raised, and the height it has to be lifted. These must be calculated. In order to reduce the shocks to which all pumps are liable the velocity of the water in the pipes should not exceed 240 feet per minute, and it will be better to limit it to 200 feet per minute. The pipes should be cast of a uniform thickness, and have brackets under the flanges, as well as a belt round the pipe at the socket end so as to better resist the strain in screwing up the joints. All the pipes should be faced, and have just sufficient spigot to keep the ring in position. They are usually made 9 feet long. To preserve the pipes from the action of the mineral water (if such is being raised), they are lined with a thin casing of wood. The pumps are kept steady by collars placed across from the buntons to the side of the shaft at each alternate pipe, so that these collars would be 18 feet apart. The pump rods or spears are usually made of Memel or pitch pine, square in section, and must be as sound and free from knots and faults as possible. They should be of uniform lengths, so that a spare rod will fit anywhere, and may be from 30 to 45 feet long. The lengths are put together by scarfed joints (Fig. 334), and secured by stout wrought-iron plates and bolts. Sometimes these plates are placed on two sides only, but often a plate is placed on each of the four sides. Where the plates are single there should be clinch-bolts a little above the other

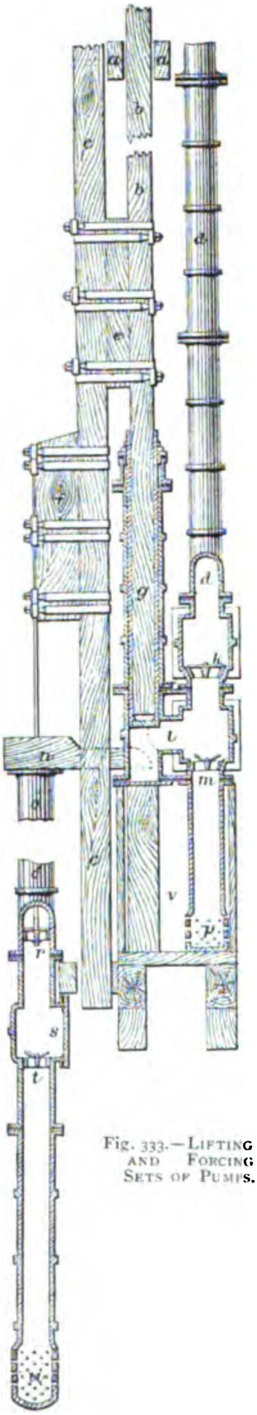


Fig. 333.—Lifting and Forcing Sets of Pumps.

Reference.

- *t*, is the clack in its seat in the lifting set, and *s* is the clack-door.
- *r*, is the bucket working in the working barrel *o*′.
- *o*, is the rising main of the lifting set, and *u* the windbore.
- *n*, is a wooden trough by means of which the water is delivered into the cistern *v*, from which the forcing set takes its water.
- *f*, is the set-off and iron straps from the main rods *c*.

In the lifting set that part of the pipe between the bottom of the working barrel and the top of the clack-piece is usually slightly reduced in size, and this allows of a drop valve with bow being dropped down into position after the bucket has been drawn up through the rising main, if it should ever become necessary.

- *g*, is the ram of the forcing set, and *e* its set-off and iron straps by means of which it is secured to the main rods *c*.
- *b*, are spears working through guides *a a*, to ensure the plunger working truly in its case; the spears *b* need only be long enough for the engine to make its stroke.
- *l*, is the H piece between the plunger and clack-pieces.
- *m* and *k* are the two clacks in their seats, *p* is the windbore receiving water from the cistern *v*, and *d* shows the rising main of the forcing set.

bolts, and at right angles to them. The bolts are made square, and tightly fit the holes prepared for them. The rods work through fixed guides to steady and keep them straight. Where the rod passes through the guide it is cased with hard wood about 1½ inches thick, which may be replaced when worn, and these parts are well greased to lessen the friction. At intervals strong safety-catches are provided to prevent the fall of the rods downwards in case of accident. These consist of short upright balks bolted to the pump-rods, and, at a point just below that reached when the rods are at the bottom of the stroke, strong cross-beams are fixed in the shaft. The method of attaching the spears to the bucket in a lifting set is by means of an iron rod fitting to the bottom of the spears, and secured by bolts as at the joints of the spears, but at the other end of the iron rod alluded to is half a joint which fits on to the corresponding half-joint above the bucket-sword shown in Fig. 335.

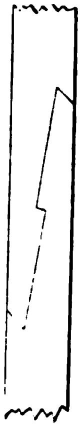


Fig. 334.—
Scarfed Joint.

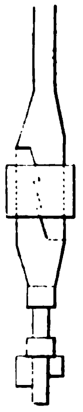


Fig. 335.—Clasp-joint for Attaching Bucket Rod to Foot Rod.

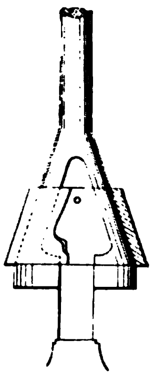
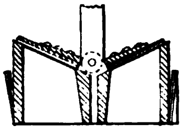


Fig. 336.—Heart Joint for Attaching Bucket Rod to Foot Rod.

This mode of joining is sometimes called a "side-joint" or "clasp-joint," to distinguish it from another form called a "heart-joint," shown in Fig. 336. In both a tapering ring of iron, somewhat similar to that used in capping pit ropes, which has been described, is dropped over the joint to prevent its parting.

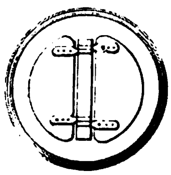
Large balks of oak, single, or two placed one above the other, are fixed across the shaft to carry each forcing set of pumps. For the same purpose, in heavy pump-work, wrought-iron box-girders are often used. They have to bear a great weight in some cases,—the weight of a long column of pumps with the contained water. Where circumstances admit of it the bottom forcing set may rest upon a good foundation made of ashlar. The diameter of the ram-case is usually rather larger than that of the suction or discharge-pipes, and where the water is corrosive the plunger should be encased in brass. The bucket, as stated, works in the working barrel, which is accurately bored out so as to ensure the bucket working truly in it, and to prevent unnecessary friction. Sometimes the working barrels are lined with brass or gun-metal, which greatly adds to their efficiency. The bucket (see Figs. 337 and 338) is packed with rings of leather to prevent leakage, and, in the common butterfly form, is furnished with two falls on the top made of iron faced with leather, the falls working on a hinge in the centre of the bucket on the top (see sketches). The hole in the bucket-rod through which the hinge-pin passes is slightly oblong, to allow the falls to lift a little in opening. Sometimes the buckets are made of brass. The clacks are usually made of brass, and have two wrought-iron falls faced with leather, similar to those of the bucket. The clacks must fit accurately in their seats, and as they remain at rest, the falls only opening and closing with the motion or weight of the water, no leather packing is required.

In other respects a clack resembles a bucket, a sketch of which is given. In the larger fittings, valves, single or double beat, are substituted for the clack. In the single-beat valve the lift is about $\frac{1}{8}$th of the diameter of the valve, and in the double-beat, from $\frac{1}{10}$th to $\frac{1}{12}$th. The valve should fit easily on the spindle. The great object, of course, to be attained by the valves is an uninterrupted passage for the water in its ascent, and the prevention of leakage of the valve when closed. It is important that all clacks or valves should be large in diameter, so as to give ample water-way, and with as small a lift as possible to prevent shocks. Platforms may be erected in the shaft for the examination and renewing of the clacks, buckets, and for packing the plunger. Where the clack and bucket doors are very heavy they may be suspended by chains with tightening screws for convenience in taking them off and putting them on again, or a winch from a recess in the shaft may be applied instead of tightening screws.



BUCKET SECTION

Fig. 337.



BUCKET PLAN

Fig. 338.

The pipes of a lifting set should be at least an inch larger in diameter than the bucket to allow of the bucket being lifted up through the pipes. Where plunger pumps are used it is customary to make the lowest set of pumps a short lifting set, so that in the event of the bottom lift being drowned by a rise of water through accident, the bucket can be drawn up through the rising main. If a forcing set were placed at the bottom, the rise of water a few feet would cover all its valves, and if these failed at such a time, the whole might be lost, or at least necessitate the employment of divers to get at them. The lifting set lifts the water from the sump to a cistern fixed or built on the next level. In some cases the delivery-piece is bell-mouthed at the top with an opening at one side delivering into the cistern along a wooden trough or landing box placed for the purpose; in others the delivery box is fitted accurately to the top pipe, which is an ordinary one. From this cistern branch-pipes lead to the forcing set (unless the windbore of the forcing set is placed directly in a cistern in the shaft which requires no branch pipes) and by it the water is raised a stage to another cistern, and so on from stage to stage. All the sets of pumps except the bottom one are forcing sets. The size of the spears is regulated by the work required of them, and for large sets of pumps, from 20 to 24 inches in diameter, they are made from 12 to 15 inches square. From the main spears in the shaft the rams at the several levels are connected by means of a set-off secured by strong iron straps. A balance bob consists of a beam of iron or timber 20 or 30 feet long turning on an axis at its centre. One end of the beam is attached by means of radius rods to the spears, and the other to a box containing old iron, or other material of the necessary weight. As the spears descend they have the weight at one end of the balance bobs against them. In the up-stroke this weight is an assistance. To ascertain the weight with which to load the balance bobs, it is necessary to take into consideration the relative weight of water in the rising mains, and the total weight of the rods. If the latter much exceed the former, the weight placed to counterbalance by means of the balance bobs must reduce the excessive weight of rods, which, however, must always exceed that of the water in the rising mains sufficiently to overcome friction and cause the down-stroke in the pit or the up-stroke in the engine-house, with a Cornish beam pumping engine. With a double-acting engine, where the force of the steam is used equally on both sides of the piston, the more nearly the weight of the water in the rising mains corresponds to the weight of the rods the better, so that the engine in that case may have an equal amount of work to do during both portions of the stroke.

It is a good plan to fix an air-cock on the top of the working barrel. Air vessels are sometimes placed in connection with the forcing sets of pumps. An air vessel should be placed vertically on the delivery side of the top clack and should have at least five times the cubical contents of the stroke of the plunger, and to be efficient must be charged at a pressure equal to the column of the rising main. If charged with air at its ordinary density, the air vessel is of very little use, and it is rather difficult from its position in the shaft to keep the air of a proper density. The action of the air vessel is this:—As the plunger descends, a part of the water, after passing through the clacks, enters the air vessel instead of flowing up the rising main. In the up-stroke of the plunger, the upper clack shuts and the flow of water for some time is maintained in consequence of the relief from the pressure which forced it into the air vessel. The advantage of the air vessel then is that it forms a cushion to resist the blow of the plunger in its down-stroke, and ensures the more equal flow of water in the delivery pipes. The water in the delivery pipes where no air vessel is used is only in motion during half the time the engine is working, but with an air vessel the water is in motion nearly the whole time, which permits of smaller-sized delivery pipes being used.

To economise space in the pit, the lifts may be fixed in one perpendicular line instead of as shown in Fig. 333. In this case the rods immediately above the plunger and those immediately below are connected by side rods of wood or iron. Fig. 339 shows this method where thick round iron side-rods are secured by screws and nuts to two crossheads strongly attached to the plunger. By this arrangement the rods are in the direct line of their work from the main beam downwards, a one-sided strain is avoided, and the shaft is left more free for other purposes than where a number of sets are taken off in various directions from the main rods.

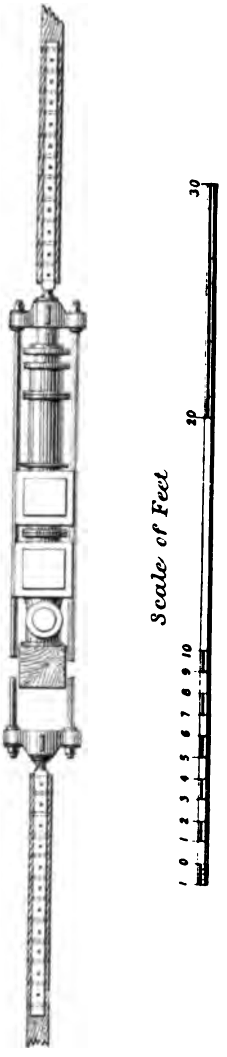
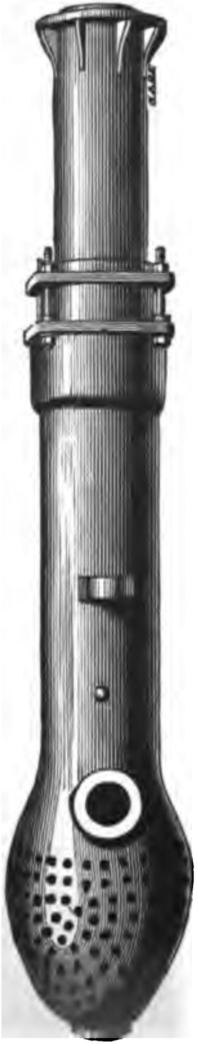


Fig. 339.—Arrangement of Forcing Sets of Pumps in a Shaft.

If the shaft in which the pumps are placed is to be used for a winding shaft also, the pump space should be bratticed off. This divides the shaft into two unequal compartments, the smaller of which serves for the pumps. There are two methods of constructing the brattice in the shaft, viz. the bunton system and the plank system. In the bunton brattice, stringing planks which are deal battens 7 inches × 3 inches in section are fixed to the sides of the shaft, one on opposite sides, from the top to near the bottom. To fix these securely, holes must be drilled in the walling at least 12 inches deep, and these holes are afterwards plugged with wood to receive the spikes which secure the stringing planks to the sides of the shaft. At points of the shaft that have metal tubbing, the stringing planks are nailed to the joints of the tubbing.



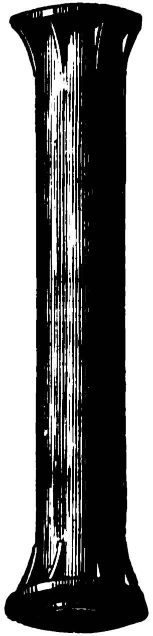


Fig. 341.

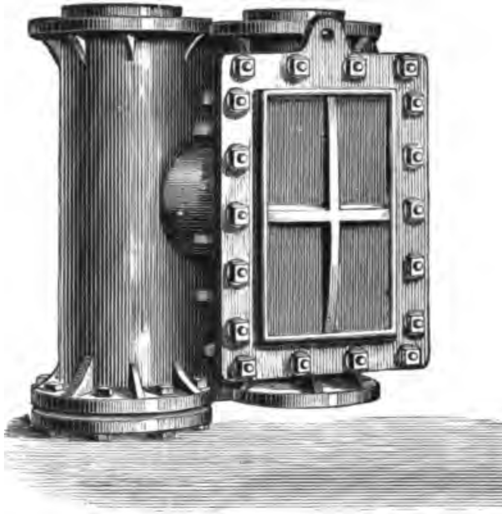


Fig. 342.

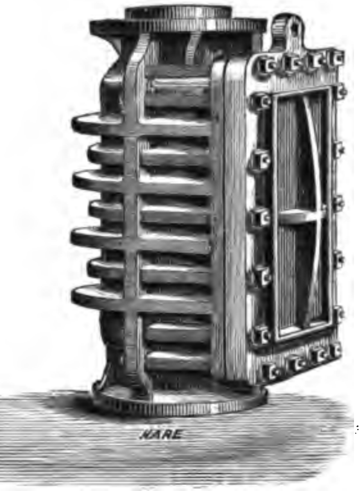


Fig. 343.

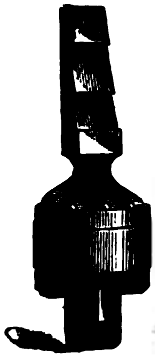


Fig. 344.

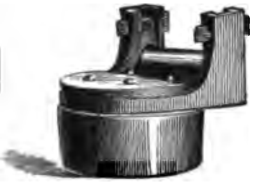


Fig. 346.

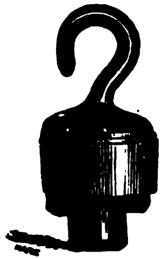


Fig. 347.

Messrs. Thornewill and Warham's Details of Pump-Work

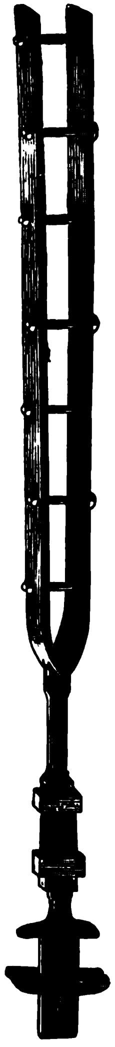


Fig. 348.

Notches are cut out of the stringing planks every 3 feet apart for the purpose of receiving other buntons, which are placed horizontally from the one stringing plank to the other across the shaft. These horizontal buntons are nailed to the stringing planks, and a cleading of fir boards from 1 to 2 inches thick is nailed vertically to the horizontal buntons. The fir boards are either planed smoothly at their edges to ensure a close fit, or if one compartment of the shaft is to be used as a downcast and the other as an upcast, to prevent leakage, these boards may have a groove ploughed in their edges, and a slip of wood which fits in the groove is inserted as the brattice is progressing.

In the plank brattice, no horizontal buntons are used, and the stringing planks, instead of being fixed singly, one on each side of the shaft as in the last case, are placed in pairs on either side of the shaft, a space of 3 inches being left between them. Memel planks 3 inches thick are then guided down edgeways through the space formed on the opposite sides of the shaft by the stringing planks to form the brattice. The brattice planks are kept in position by having their edges planed smooth and by using iron dowels, or the joints may be made true as in the manner described for bunton brattice. The plank brattice makes a stronger and more permanent division of the shaft than the bunton brattice.

Pumps for a sinking pit are somewhat different from the permanent arrangement. Either the whole set of pumps or the bottom part must be hung in the pit so as to follow the progress of the sinking. A lifting set is invariably used in the bottom of a sinking pit; and as the sinking progresses over 40 fathoms, a permanent forcing set may be placed at that level, and the lifting set used again for another stage, supplementary spears to work the bucket being attached to the main spears. A method of hanging the pumps in a sinking pit is by two ground spears, fixed one on each side of the set by iron collars. At the top of each of the ground spears is one of a pair of 5- or 7-fold blocks called ground blocks, the other being placed on buntons at the top of the pit. Through these blocks a pair of ropes are rove, the bottom ends being connected to the ground spears and the surface ends of each being taken to a ground crab. The ropes are called ground ropes. The pumps are steadied in their position by means of temporary buntons or collarings. The top pipe or pump is called a "hogger." It is bell-mouthed on the top, and just below at the side is provided with a flexible hose to accommodate itself to the varying height of the column. As the sinking proceeds, the pumps are lowered by the ground crabs, and when the hogger pump is down nearly to the delivery drift, it is taken off by means of the main crab, and lifted over the top of the spears. To allow of this being done readily, instead of the spears being attached to the engine beam, they are clamped to the front of a piece of wood called a Y. A length of pump is then added to the column, a length of spears being added as required, and the hogger pump replaced.

To avoid hanging the whole column, an arrangement, whereby the suction and three following pipes only are suspended, and all other parts are fixed by buntons, is often used. This is called a "sliding suction," that part being made telescopic. When it is necessary to add a pipe, it is put in by breaking the joint above the bucket door and lowering the parts below this to admit of the new pipe. The engine must not be driven faster than is necessary to keep down the water, or too much air will be drawn into the pumps to the injury of the working parts. If it is desirable at times to lower the water below the level of the upper holes of the windbore, these should be carefully plugged first with soft wood.

Figs. 340 to 349 show details of pump-work made by Messrs. Thornewill and Warham, Engineers, of Burton-on-Trent.

Fig. 340 is a sliding windbore used in connection with a bucket set when sinking. The snore-piece is made very strong to prevent its being broken by pieces of rock striking it when the sinkers are blasting, and it has a hand-hole fitted with a wooden plug. On withdrawing the plug, the hand may be passed through the hole to the inside so as to clear the snore-holes when necessary. The lower portion slides up or down by means of a stuffing box which is usually packed with spun yarn and luted (sealed) with clay. The lug shown on the drawing above the hand-hole forms a means of attachment for the suspending rods, which serve to raise or lower the bottom portion of the slide-piece as required.

Fig. 341 is a working barrel with bracketed flanges and spigot and socket joints. It is bored to gauge and bell-mouthed at each end so that the bucket may be easily led into the barrel bore.

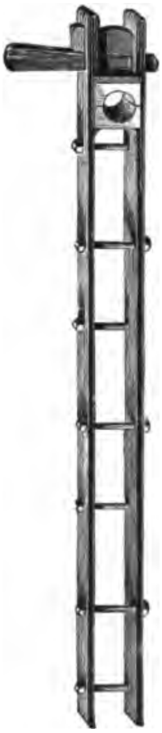


Fig. 348.

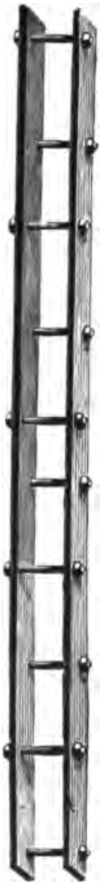


Fig. 349.

Fig. 342 is an H piece for a plunger set, with a clack-box and a portion of the plunger or pole case attached by a branch.

Fig. 343 is a clack-piece for either a bucket or plunger set, and is well ribbed for deep lifts.

Fig. 344 is a bucket with sword and half of clasp joint for attaching to corresponding half on upper rod, as shown in Fig. 345. The buckets have leather falls shod with iron, and bucket packing of leather held in place by a wrought-iron hoop with crossbar.

Fig. 346 is a clack for a plunger-lift, and may be used for a bucket-lift when it is not necessary to draw the clack up through the working barrel, in which latter case a clack as in Fig. 347 is fitted. Both clacks have leathers shod with iron plates securely riveted thereto.

Fig. 348 is a pair of strapping plates for bolting to the wood spears and having fitted a pair of gun-metal boxes for attachment to the gudgeon of a T bob.

Fig. 349 shows the ordinary strapping plates and bolts for connecting the various lengths of wooden spears.

The Pulsometer Engineering Company make a special form of sinking pump called the "Deane" double plunger and shown in Fig. 350. It is designed to raise water from depths beyond the range of the pulsometer, and as it takes up but little room, will be found very serviceable in small-sized shafts of limited depth. It will work either when hanging from tackle as shown in the figure or when attached to the timbering by means of the "dogs" with which it is provided.

The pump is driven by a vertical steam cylinder with its piston, the strokes of which are perfectly regulated by controlling valves. This piston imparts a reciprocating motion to the barrel beneath and at the same time to the lower plunger to which it is attached. This plunger is hollow, but has a valve opening upwards at its upper extremity. The barrel beneath the piston of the steam-engine works over an upper plunger, which is hollow and of smaller size than the lower. A valve is placed below the lower plunger, The operation of the pump is as follows:—On the up-stroke the water runs from the suction-pipe through valves and occupies the displacement of the lower plunger while the contents of the upper barrel are forced up through the upper plunger and discharge-pipe. On the down-stroke the water in the lower barrel is forced through the valve at the upper extremity of the lower plunger, and as the displacement of the lower plunger is double that of the upper, one half is discharged at the delivery pipe while the other half is accommodated in the upper barrel. Thus water equal to the displacement of the upper plunger is discharged at each single stroke of the machine. By the construction of the pump the water is forced up and through its several parts almost in a straight line, doing away with all friction incident to circuitous and intricate passages.

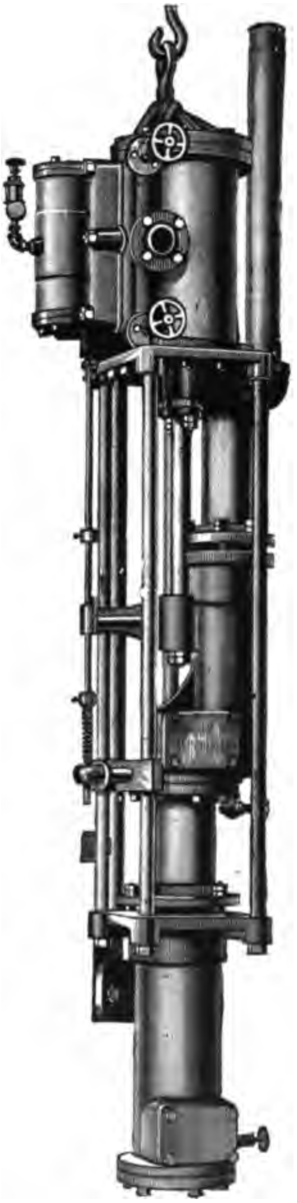


Fig. 350.—The Deane Sinking Pump.

Various kinds of engines are used to work the pumps. Fig. 351 shows a side elevation of the Cornish Pumping Engine. The following description is chiefly taken from André's treatise on *Coal Mining*, as also the drawing, Fig. 333, usually associated with that of the Cornish pumping engine. The cylinder F is 70 inches in diameter with a 10-foot piston stroke. It is provided with a cast-iron steam-jacket connected to the boilers by means of the pipe H. The boilers are situated below the pipe H, so that the water of condensation returns by it. Where the boilers from unavoidable circumstances are above the cylinder, the supply-pipe to the steam-jacket cannot serve as a return condensed water pipe, and provision must be made for freeing the jacket from condensed water. This will be best done by means of a steam trap. The steam-jacket is encased in wood, the annular space between the two being filled by some non-conductor of heat such as sawdust, and the whole is enclosed by brickwork either in contact with the wood casing or separated from it by a few inches, and is plastered on the outside and covered with wood panelling. The cylinder cover is fitted with a false lid or cap which encloses a thick layer of sawdust, and is thus protected from the cooling influence of the air. The space under the cylinder bottom is protected from the same influence by steam filling it from a branch of the pipe H. C is the main beam, cast in two plates which are bolted together, with distance blocks between to keep them truly parallel.

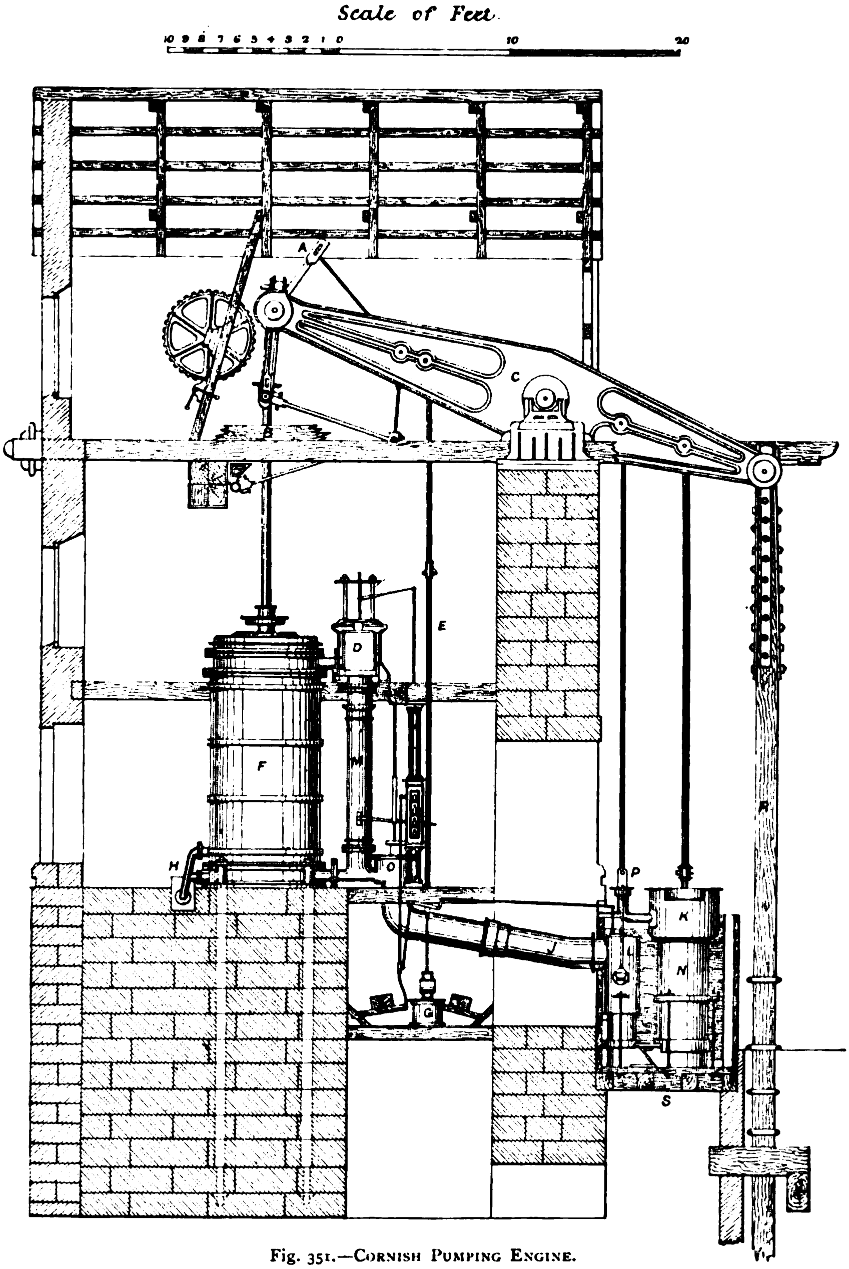


Fig. 351.—Cornish Pumping Engine.

A catch-piece, A, is fixed to the upper part of the beam by means of brackets. On the piston reaching the bottom of its stroke, the catch-piece touches the blocks B, fixed on the spring beams, arresting the piston and thus preventing injury to the cylinder by the engine making too long an in-door stroke. E is the plug or tappet-rod for working the valves and cataract, and D the top nozzle shown also in section in Fig. 352. The nozzle contains three valves. G (Fig. 352) is the governor or regulating valve, for regulating the admission of steam into the chamber E E of the nozzle, whence

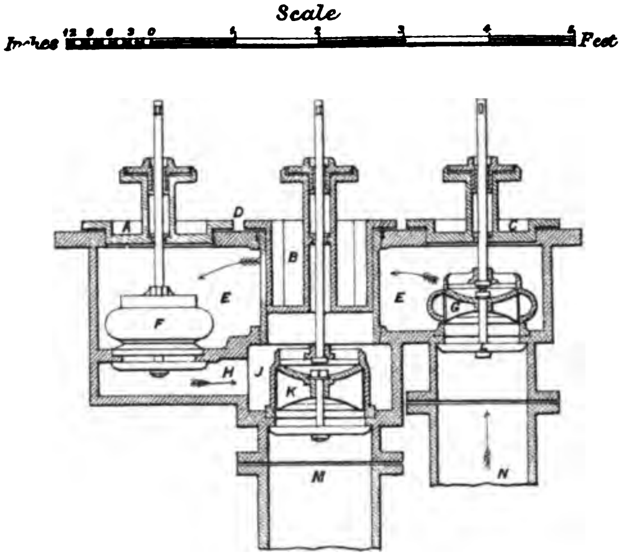


Fig. 352.

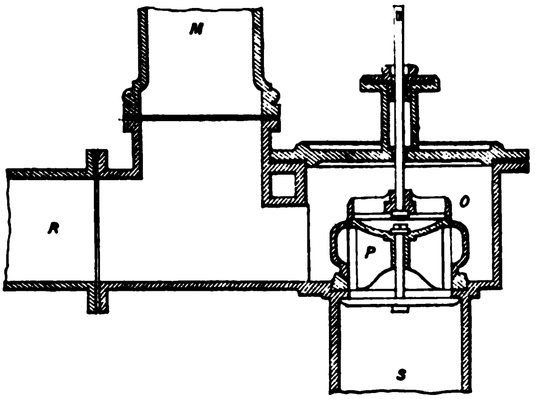


Fig. 353.

CORNISH PUMPING ENGINE VALVES.

it afterwards passes through the steam valve F into the cylinder. The governor-valve is not moved by the engine during its working, but is regulated occasionally by hand. In proportion to the raising of the governor-valve more or less, the steam is less or more wiredrawn or reduced in pressure in its passage from the steam-pipe into the cylinder. By this means, although the boiler pressure may vary, the mean effective pressure in the cylinder will be kept more constant.

The motion of the governor-valve is under the control of the engine-man, and a handle is placed within his reach which is connected by a rod and a lever with the valve-spindle.

F (Fig. 352) is the steam-valve which admits steam into the cylinder. The chamber E E is not separated by the cover B, as may be thought from the drawing, but an uninterrupted passage is afforded for the steam through the chamber E E, as shown by the arrows. On the steam-valve F being lifted (if the governor-valve is also open), the steam is free to pass through it from the chamber E E to the passage H, and thence by the steam-port J into the upper part of the cylinder. K is the equilibrium-valve, and is situated in the middle of the nozzle. On being lifted the steam above the piston is free to find its way along the equilibrium-pipe M (Figs. 351, 352, and 353), and by the lower port into the lower portion of the cylinder under the piston. At the moment the pressure of the steam above and below the piston becomes equal, the rods R (on account of their weight) hung at the other end of the beam descend, and the piston is drawn upwards, the steam above it acting as a cushion in its ascent.

The upper nozzle D has an external casing of thin iron, a space being between it and the nozzle which is filled up with sawdust or some other non-conducting material to retain the heat.

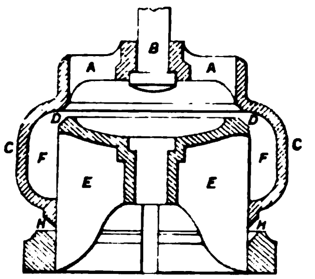


Fig. 354.—Cornish Pumping Engine. Enlarged sketch showing Vertical Section of Equilibrium Valve.

A reference to Fig. 352 shows that the three covers C, A, B, which are bolted to the nozzle over the governor, steam and equilibrium valves respectively, are of such a size as to allow of the valves being examined or repaired on removing the covers. O is the bottom nozzle (Fig. 351), and shown in section in Fig. 353. In it is placed the exhaust-valve P for opening or closing the communication between the lower part of the cylinder and the condenser. Above the valve the nozzle-chamber O is in communication with the cylinder by the lower port, and under the valve the bottom of the nozzle communicates with the eduction-pipe J in Fig. 351, S in Fig. 353. When the exhaust-valve is raised the steam in the lower part of the cylinder is exhausted into the condenser L.

That the action of the Cornish valve may be rendered plainer to the student, an enlarged section of one is given in Fig. 354. The valve is designed to give a large extent of opening for the passage of steam with little traverse, a small amount of power being necessary to work it. E is the fixed seat, made of cast-iron or brass, which forms part of, or is secured to, the valve-chamber. C is a bell-shaped valve-piece, also of brass, actuated by a rod, B. The valve is in contact with its seat at two places, D and H, which are formed into accurate conical surfaces, the one, D, being internal, the other, H, external. When the valve is closed these surfaces accurately fit similar ones on the seat, and when lifted, as shown in Fig. 354, two annular openings are formed at the same time, thus giving a double passage to the steam through the valve. The spindle, B, of the valve, C, is fixed to a centre eye, cast in one with the valve-piece, and connected to it by four arms, A A being two, the others being at right angles to them. The seat is also formed similarly with four arms, having an annular ring at the base, the top edge of which is bevelled and ground to fit the lower edge of the valve, C, and when lifted forms the opening, as shown at H.

Referring again to Fig. 351 it will be observed that the piston-rod is connected with the end of the beam by a parallel motion which ensures the ascent and descent of the piston-rod in a vertical position whilst the end of the beam moves through the arc of a circle. A very striking feature of the Cornish engine is the cataract governor, G (Fig. 351), shown also in section in Figs. 355 and 356. This and the parallel motion were the inventions of Watt. It consists of a pump placed in a circular tank of water below the level of the cylinder. G is a barrel, and F the plunger working in it as in an ordinary small forcing pump. The water reaches the barrel through the inlet-valve, H, which opens freely upwards, but the passage for the outlet is contracted as desired by a moveable plug, L. The plunger is connected by a joint to the lever, A. This lever is loaded with a heavy weight, C, on the same side of the fulcrum as the plunger, and the lever projects on the other side of the fulcrum, terminating in the handle shown on the drawing. When the tappet or plug rod, D (which is worked off the main beam), has descended nearly to the end of its stroke, a projecting block on the lower part of it, E, called a tappet, strikes the end of the lever, A, and in consequence the plunger, F, is raised, the water flowing freely through the valve, H, and following the plunger in its ascent. After the completion of the down stroke the piston of the engine begins to ascend (due to the opening of the equilibrium-valve by the tappet-rod), the tappet-rod also ascends, and leaves the lever, A, and the heavy weight, C, which was raised at the same time as the plunger, becomes the motive power in driving the water from the pump by forcing down the plunger. The inlet-valve, H, having closed before the descent of the plunger, the only passage for the water is by the aperture left round the regulating plug, L. It will be seen, therefore, that the time occupied by the pump-plunger in its descent depends upon the size of this aperture, which the attendant regulates by means of the plug fitting it. Near the fulcrum of the lever, A, but on the tappet-rod side of it, a rod, B, is jointed to the lever, and ascends vertically from it. In its ascent this rod opens first the exhaust, and shortly afterwards the steam valve, thus causing the engine to commence the next down-stroke. The rod, B, acts upon a catch that releases weights which, by their fall, open the valves, causing a suddenness of action for which considerable advantage is claimed, especially as regards the admission of steam into the cylinder by the steam-valve. The interval of time between any two consecutive strokes must, therefore, depend upon the time occupied in the descent of the cataract-plunger, and upon the amount of opening given to the regulating plug, L. By means of a micrometer screw and handle connected with L by a rod and the lever, M, the plug can be regulated to any degree of opening required. The steam-valve can thus be opened any desired

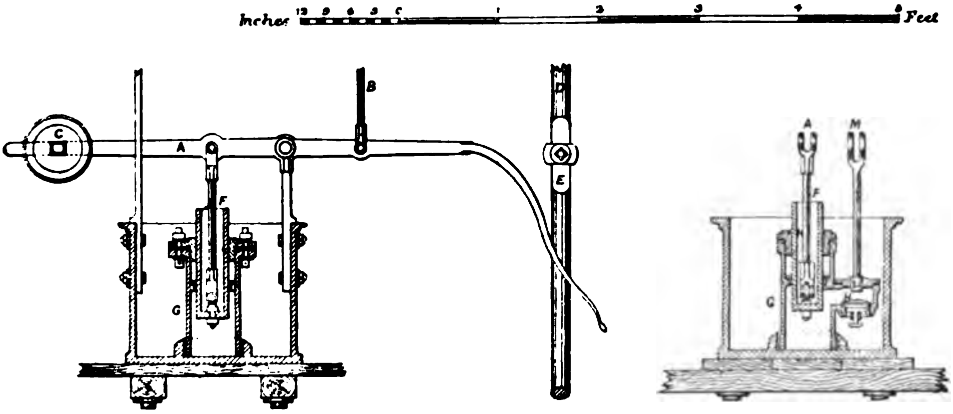


Fig. 355. Fig. 356.

WATT'S CATARACT GOVERNOR FOR REGULATING THE SPEED OF THE CORNISH PUMPING ENGINE.

number of times per minute, and the number of strokes in that interval of time thereby regulated.

J (Fig. 351) is the eduction-pipe, connecting the condenser, L, with the bottom of the exhaust-valve nozzle, O. The size of the air-pump, N, is approximately half that of the steam-cylinder, being 2 feet 9 inches in diameter, with a 5-foot stroke. It is fitted with a foot-valve. P is an ordinary plunger feed pump. The condenser, L, is fitted with an injection cock and valve for regulating the cold-water supply to it. The pump-rods, R, are frequently attached to the main engine-beam, as shown on the drawing, without the intervention of a parallel motion, and with only side guides in the shaft. The consequence is that near the surface the rods deviate considerably from a vertical line in their ascent and descent, but this deviation is less at lower portions of the rods. It is a much better plan for the pump-rods to be attached to the main beam by a parallel motion, so as to ensure the rods maintaining a vertical position. This may be done by a gudgeon on top of the rods carrying two side-blocks working in cast-iron guides for a sufficient length to take the stroke of the engine. The gudgeon may be attached to the beam by two iron radius-rods, the space between them being filled with pitch pine.

Husband's Patent Gearing may be applied to the engine. In the event of breakage this gearing is intended to open the equilibrium-valve, and stop the engine.

The steam in the Cornish Pumping Engine is admitted at high pressure, and may be worked expansively, the steam being cut off at some point of the stroke, which may be from one-sixth to one-third. In consequence, however, of the great shocks to the engine, and especially the pit-work, when a high rate of expansion (with its accompanying high velocity during the stroke), is attempted, which shocks lead to breakage and stoppage of the pumps, it is not found advantageous to work at any but a very moderate rate of expansion, and thus economy of steam is sacrificed for the sake of safety in working. The Cornish engine being single-acting it necessarily requires a larger cylinder for a given amount of work than a double-acting engine. As regards fuel consumed it is economically worked, from 3 to 6 lbs. of coal being used per hour per indicated horse-power.

The Cornish Pumping Engine is a wonderfully elaborate and complicated specimen of ingenuity and skill. Where permanent working can be assured by a continuance of the working conditions and requirements of the mine, it well remunerates the enormous outlay necessary in purchasing and erecting the immense structure. The cost of a first-class Cornish engine is about £8 per H.-P.

A 90-inch or 500 H.-P. engine therefore costs about £4,000; then the three-storied engine-house and pillars for such engine would require about 500 cubic yards of building, the price of which varies in different localities, according to the character and price of materials such as stone, brick, lime, &c. In Cornwall, where suitable stone abounds, it seldom exceeds 6*s*. 6*d*. per yard. Again, the steam case must be covered with some non-conducting material, or else, having a greater surface-area than the steam cylinder, it becomes a large condenser.

Another objection to the Cornish engine is the large number of parts to maintain in an efficient state of repair and cleanliness. It has four double-beat valves with their sets of bright nozzle gear, consisting of pillars, levers and guides, quadrants and catches on the ground-floor, tappet-rod carrying its adjustable blocks to work the four levers of double-beat valves and cataract governor in the basement; air-pump and condenser with foot and delivery valves, injection cock and valve and lever attachments; also a large cast-iron beam, which for a 90-inch engine weighs 30 tons, and the parallel motion with its many joints, all requiring a considerable amount of attention and care.

Davey's Patent Differential Valve Gear may be applied to work the valves of the Cornish as well as to other engines, thus doing away with the tappet gear. A perspective view of it is shown in Fig. 357. A lever (*a*) called the main lever, gives motion to the valves through a rod (*b*). The motion of the engine is given to the outer end of the lever through the rod (*c*), by means of a lever of the first order, the long end of which is attached to the plug rod or any moving part of the engine, where it gets the motion of the piston on a reduced scale; the other end (*d*) deriving its motion from a subsidiary cylinder (*e*), and being controlled by means of the cataract (*f*). The cylinder has a slide valve, which is worked by means of a tappet arm on the rod of the piston of a secondary cylinder; the motion of the secondary piston is also controlled by a secondary cataract. The slide valve is, however, free to move with the motion of a hand lever.

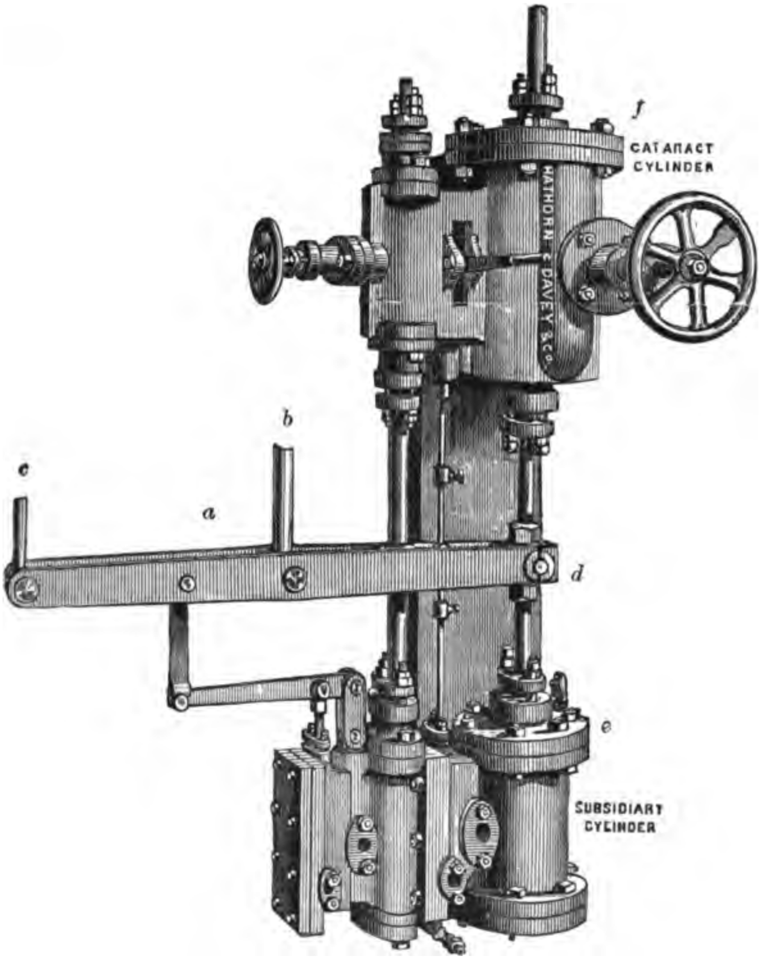


Fig. 357.—Davey's Differential Valve Gear as applied to work the Cornish Engine.

It will be seen that there are two hand wheels and a lever attached to the cataracts. The function of the large wheel is to regulate the speed of the engine during the stroke, the small wheel is for regulating the pause between the strokes, whilst the hand lever enables the engineman to work the engine by hand, if necessary. A rocking shaft, not shown in the engraving, is employed to give motion to the valves of the engine in the usual way.

The action of the gear may thus be described:—Let the engine be "out doors." * The engine end of the main lever will then be in its highest, and the opposite end in its lowest position, the secondary lever being lifted so as to admit steam to the bottom of the secondary cylinder. The engine will pause until the piston of the secondary cylinder has travelled to the end of its stroke, and lifted the valve of the subsidiary cylinder. The pause will be long or short, according to the regulation of the secondary cataract. The piston of the subsidiary cylinder then having steam on its lower surface will travel upwards, actuating the main lever with a speed dependent on the adjustment of the cataract. In doing so, the steam valve of the engine will be opened by means of the rod (*b*), and a rocking shaft, &c., and will be opened quickly, because the engine end of the lever is, for the time being, stationary. Steam now being admitted on the engine piston, it will, after overcoming the inertia of the load, move off at an increasing speed, which is communicated to the engine end of the main lever. The result is that as the opposite end of the lever is moving uniformly in the opposite direction at the same time, the motion of the centre is soon reversed, and the steam valve, which was opened by the motion of the subsidiary piston, is closed by the differential motion brought into action by the motion of the main piston acting on the same point.

Steam engines (such as the Cornish Pumping Engine, just described) in which the exhaust steam is conveyed to a condenser and afterwards removed with the air liberated in the process of condensing by the air-pump have a vacuum gauge fitted to them, as a means of indicating the amount of vacuum obtained.

If it were possible to have a perfect vacuum by using the air-pump we should have registered on the gauge about 30 inches or nearly 15 lbs. pressure per square inch, and the more nearly this reading is approached the better the attempt at producing a vacuum. The air-pump, however, never exhausts all the air from the condenser; what is left is in a highly attenuated state, and this prevents the gauge ever registering so great a pressure. Vacuum gauges are more particularly described in Chap. XVI. of this work.

The Cornish Bull engine is a type of pumping-engine in which the beam is placed below the cylinder and the spears are on the same side of the beam as the cylinder. A weight is placed on the other end of the beam. It is a double-acting engine and has the assistance of the vacuum as well as the pressure of the steam during the up and down stroke of the piston. It does not require the expensive pillar necessary for the single-acting Cornish engine, and being double-acting a cylinder of half the size will do an equal amount of work. The cylinder is placed vertically. A serious objection to the Bull engine is that it stands right over the pit, the piston-rod forming one line with the spears.

Another kind of Cornish engine seems to be a combination of the two described. In it the beam is placed under a vertical cylinder, the spears in the pit being at one end of the beam, the cylinder at the other. This removes the objection of the engine being placed over the shaft, and very expensive pillars are not required. This is a double-acting engine.

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* It is customary to speak of the Cornish engine as being "out-doors" when the piston has reached the top of its stroke, and "in-doors" on its descending to the bottom. In the case of a horizontal pumping engine, the out-door stroke takes place when the connection between the engine and quadrant moves away from the engine-house towards the shaft, and the in-door when the motion is reversed.

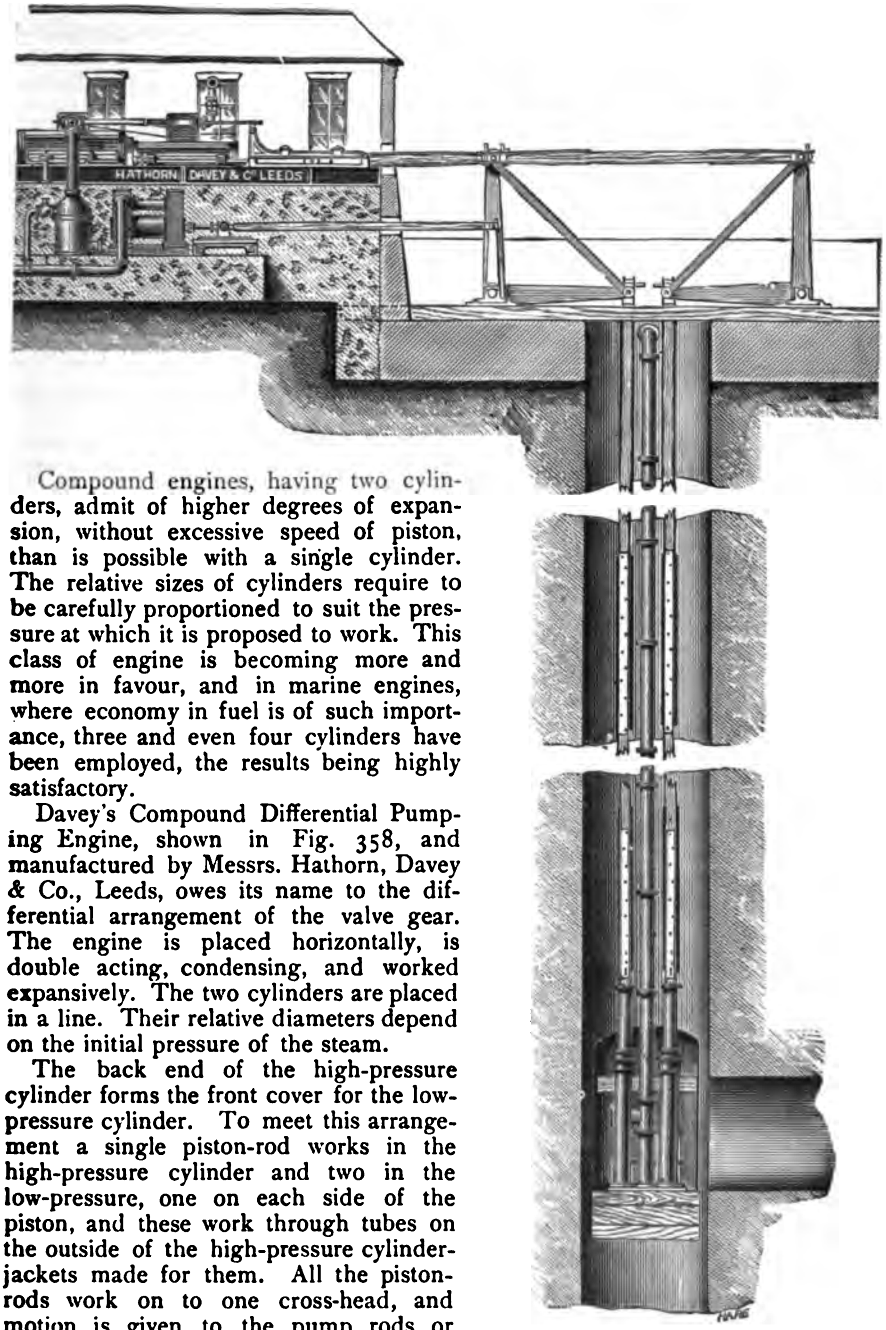

Fig. 358.—Davey's Compound Differential Pumping Engine.

Compound engines, having two cylinders, admit of higher degrees of expansion, without excessive speed of piston, than is possible with a single cylinder. The relative sizes of cylinders require to be carefully proportioned to suit the pressure at which it is proposed to work. This class of engine is becoming more and more in favour, and in marine engines, where economy in fuel is of such importance, three and even four cylinders have been employed, the results being highly satisfactory.

Davey's Compound Differential Pumping Engine, shown in Fig. 358, and manufactured by Messrs. Hathorn, Davey & Co., Leeds, owes its name to the differential arrangement of the valve gear. The engine is placed horizontally, is double acting, condensing, and worked expansively. The two cylinders are placed in a line. Their relative diameters depend on the initial pressure of the steam.

The back end of the high-pressure cylinder forms the front cover for the low-pressure cylinder. To meet this arrangement a single piston-rod works in the high-pressure cylinder and two in the low-pressure, one on each side of the piston, and these work through tubes on the outside of the high-pressure cylinder-jackets made for them. All the piston-rods work on to one cross-head, and motion is given to the pump rods or spears by means of two quadrants or L pieces, working two sets of pumps. The engine does not require expensive foundations. In one arrangement the low-pressure engine piston has a rod passing through its back cylinder cover called a "tail rod," which works the air-pump and condenser, which are placed on a separate bed, behind the low-pressure cylinder. With large engines, the air-pump and condenser are placed by the side of the engine and under the engine-room floor, as illustrated in Fig. 358.

Davey's shutter valve may be applied to pumping engines. By its means the engine is enabled to do more work on one side of the stroke than on the other, and is especially useful when the engine is used for sinking operations, as it enables the engineman to adjust the steam supply whilst the engine is running, and cause it to work uniformly, even though the pumps may be constantly put out of balance by the sinking operations. In the Differential engines intended for sinking purposes one of the quadrants is provided with a balance arm, for the convenience of balancing the engine as the shaft is sunk. To avoid the necessity of altering the balance weight during the progress of the work a shutter apparatus is placed on the back of the main steam valve, enabling the engineman to adjust the relative supply of steam to the two ends of the cylinder, and thereby causing the engine to work uniformly when out of balance.

Davey's safety trip gear is applied to a direct-acting pumping engine in order to check it in the event of the breakage of a spear rod, or to resist the action of a riding column on the pumps. It is applicable to direct-acting engines of all descriptions. By this gear the communication between the high and low pressure cylinders is kept open by a catch which is released by the engine whenever it suddenly increases in speed from loss of load.

The great features of the Differential engine are its differential valve gear and safety trip gear. They admit steam to the engine in proportion to the resistance to be overcome, and in case of a sudden total loss of load, reverse the steam to catch the piston. The distribution of the steam is effected by coupling the motion of the engine with that of a subsidiary piston whose velocity is made uniform by its cataract. The engine is made to cut off steam by its motion, while the uniformly moving subsidiary piston is employed in admitting it. As long as the resistance to the engine is sufficient to prevent its motion from becoming equal to that of the subsidiary piston, steam is admitted up to the fixed point of cut-off; but should a loss of resistance, or an increased pressure of steam cause the engine to acquire a speed relatively greater than the speed of the subsidiary piston, the motion of the steam-valve would be reversed earlier and the supply of steam would be adjusted to the altered conditions. The effect of a sudden loss of load is to reverse the action of the valves and to throw the steam against the motion of the piston, stopping it before the end of the stroke, and this is done by the automatic working of the valve gear. The Cornish engine has no such efficient means of avoiding shocks, and the contrivances for preventing injury to the engine in case of accident are of a rougher description. The Differential pumping-engine gives a uniform speed and prevents all shock; and it does not require so expensive an outlay for the engine-house.

The quadrants connecting the pumps to the engine are constructed of wrought iron, and are so arranged that one plunger makes the up stroke whilst the other is making the down stroke; thereby a continuous stream of water is delivered through the rising main.

The suction and delivery valves and valve boxes are specially constructed to withstand heavy pressures, and are so arranged that they can be readily got at in a chamber by the side of the pit.

The engines are made in all sizes up to 76-inch cylinders, and 10-foot stroke, and have been very largely adopted both for coal and metalliferous mines.

The cost of working a Compound Differential Pumping Engine, supplied to the Yarlside Mining Company, Dalton-in-Furness, including fuel, wages, stores, repairs, and interest on capital at 10 per cent. amounted to less than one farthing per 1,000 gallons raised 100 feet high.

The engine has 33- and 60-inch cylinders with a 10-foot stroke, and it works two 20-inch plunger lifts, each pumping 400 feet high in one lift.

A larger engine of this description has been supplied by Messrs. Hathorn, Davey & Co., to the South Staffordshire Mines Drainage Commissioners. It has a 44-inch high and a 76-inch low pressure cylinder, the stroke being 10 feet, with air-pump and surface condenser, fitted with a large number of 1-inch gun-metal tubes. The condensed water is cleared from grease, and is then used to feed the boilers. The engines work two 19-inch plungers, with 10-foot stroke placed at a depth of 464 feet, forcing the water up an 18-inch column to the surface. The buckets and clacks are of gun-metal and of the Cornish type, with double beats. At each stroke of the plungers 245 gallons of water are raised, and the engine is capable of raising 2,000,000 gallons in twenty-four hours.

The advantages of pumping-engines being placed on the surface are:—a minimum of loss in the steam-supply, the boilers being close to the engines; better supervision and greater facilities for repairs; if the mine should be flooded, and the water rise above the pumps, the engine can work on uninterruptedly; and further, in cases where the heaviest feeders are met with and collected in the shaft, the lifts may be reduced in size from the top lift downwards. The water, perhaps, is pumped out of a shaft by means of a sinking set. The engine obtained for the purpose of working the pumps during sinking may be made to work the pumps permanently.

Pumping-engines are sometimes placed underground. A pit may be sunk comparatively dry, or the water may have been wound out by barrels or "kibbles" during the sinking, and afterwards large quantities of water may be made in the workings. On the other hand, a great advantage of the engine being placed underground is the absence of the cumbersome pump rods or spears, which are both costly in the first instance and in the wear and tear of working. Moreover, the spears limit the length of the column. Another advantage of placing the engine underground is derived from the fact that a much smaller engine will do the work there than on the surface, as the pumps may be always double-acting, and pump-rods being dispensed with, may be worked at a higher speed than engines on the surface. Lodge-rooms and cisterns in the shaft are also not required. In the usual form of pump worked by an engine on the surface the water is only in motion half the time, but with the engine placed underground it is continuously so, and smaller pipes will suffice, although they may require to be made of thicker metal in a long column. A disadvantage in placing the pumping-engine underground is that a disarrangement or accident to the pumps is more difficult to deal with, and also that the engine is liable to be drowned whilst standing. Steam may be conveyed from the surface down the shaft to the engine, or the boilers may be placed underground, but as stated when dealing with steam haulage, it is better to have the boilers on the surface. All large engines for underground pumping and their pumps, and most of the smaller engines or steam pumps, are placed horizontally, an arrangement which takes up little room and gives great compactness. The piston of the engine and the plunger of the pump are generally on one rod. The pumps underground should be double-acting, and it is well to have a pair of engines either of which is capable of working the pumps. The engines may have condensing arrangements or may exhaust into the upcast shaft. If there is no separate condenser, the exhaust steam can be led into the suction water pipe, which will not only get rid of the steam, but assist the engine by forming a partial vacuum. The Compound Differential engines give excellent results when placed underground. They have been made of all sizes of cylinder up to 60 inches in diameter and 8 feet stroke. The engine has frequently been placed, where circumstances have permitted, at a position in the pit above the workings, and the water made in the latter has been lifted to the main engines by means of a hydraulic pumping engine, deriving its supply from the rising main of the main pumping engine. This arrangement has been adopted by Messrs. Hathorn, Davey & Co., in cases where the workings would be flooded before the water could reach the main engines. The hydraulic engines are so constructed as to work under water. In many cases, the power is supplied to them from accumulators on the surface.

The Worthington Compound Expansive Pumping Engine is thus described in the *Practical Engineer* of Feb. 22, 1889.

Direct-acting steam-pumps of the usual type are wasteful in their use of steam, and do not work so economically as the best expansive fly-wheel engines and pumps. They are, however, very convenient for small powers, and have the advantage of low first-cost and inexpensive maintenance, which may in some instances compensate for excessive steam consumption. Where the steam cylinder is in direct line with the ram driven without the intervention of a fly-wheel, it is evident that throughout the stroke a fairly uniform pressure must be maintained on the piston to overcome the resistance of a constant head of water in the pumps. The full boiler pressure must therefore be carried throughout the stroke, or if this is more than is required to overcome the resistance, the steam must be throttled throughout the stroke.

The Worthington pump is an improvement on the ordinary form of direct-acting steam-pumps, and the objections raised to the use of the latter do not apply to the former. Figs. 359 and 360 show a Worthington pumping-engine designed for large powers and high duty. It is a compound engine, and considerable expansion is obtained in the steam cylinder without the intervention of fly-wheels or other heavy moving masses. Two distinct steam-cylinders and pumps are placed side by side and so arranged that the slide-valve of the one steam cylinder is actuated from the piston-rod of the other. By this means the piston-valves common to other direct-acting pumps are dispensed with and only plain slide-valves used. The two engines act in concert, so that one is always in motion whilst the other is stationary: therefore both valves are never at rest at the same time, and there is no tendency to stop in any position without starting again, whatever speed the pump is working at. The plungers are double-acting, and a continuous flow of water is obtained. The pump works without shock or jar, and almost noiselessly, even against the heaviest pressures. The gear designed to equalise the varying pressure in the steam-cylinder due to the expansion is shown in Fig. 360. It consists of two oscillating cylinders working on trunnions placed so as to be opposite to each other at the centre of the piston-stroke. The pistons working in them are single-acting and are connected to the piston-rod, as shown in the drawing. The motive-power for the oscillating cylinders may be air or water supplied from suitable reservoirs, pressing the pistons outwards. At the commencement of its stroke the full boiler pressure is admitted to the steam piston, and the oscillating or compensating cylinders point in the direction shown in position 1, Fig. 360, and resist the forward motion of the steam-piston. The resistance gradually diminishes as the middle of the stroke is approached, and, when reached, the air cylinders press against each other without affecting the steam-piston, as shown in position 2, Fig. 360. As the steam-piston continues its course beyond mid-stroke, the compensating pistons give out more and more power and assist the steam-piston, their force being greatest at the end of the stroke, when the steam pressure is at its lowest. They are then in position 3. The effort of the steam is resisted at the first part of the stroke and strengthened at the latter part. In the engine shown on drawings four steam cylinders are used, two to each pump, high and low pressure; another form of Worthington pump contains only two steam and two

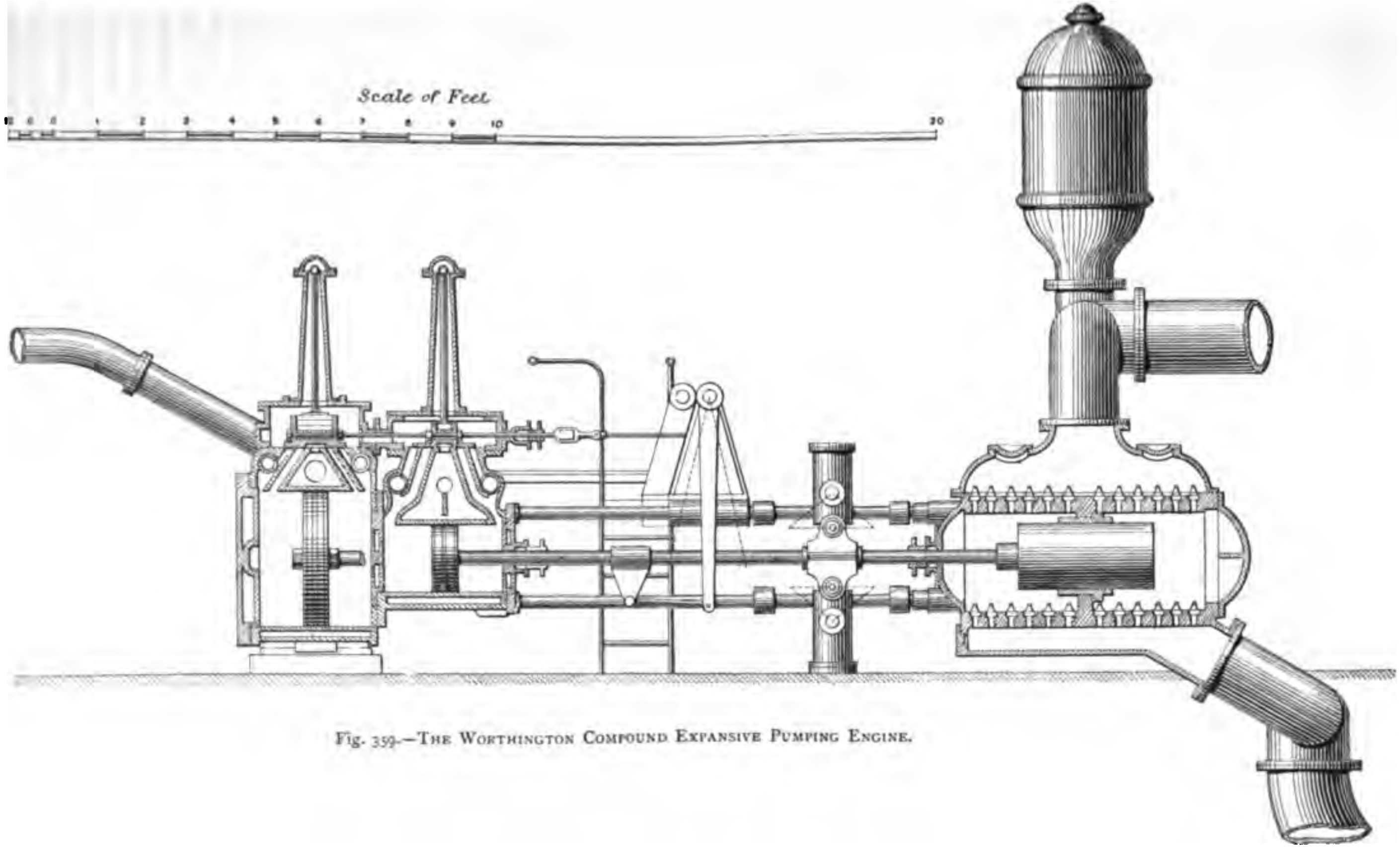

Fig. 359.—The Worthington Compound Expansive Pumping Engine.

water cylinders, but they always consist of two steam-pumps placed alongside each other. In consequence of the slide-valve arrangement, whereby the piston-rod of one cylinder actuates the slide-valve of the other, the pump is always ready to start, in whatever position it may stop, like an engine with two steam cylinders and cranks at right angles.

Double steam-ports are used in the cylinders, the object of which is to cushion the piston and prevent its striking the covers. When the piston passes the first port in its motion towards the cover, it compresses the steam contained and stops. If the second port were not at the end, when the steam-port opened, the piston would not be started. If the first port were at the end, there could be no cushion for the piston.

In a test of the duty obtainable from these pumping-engines, with an engine having a 26-inch stroke, the high-pressure cylinder being 1 foot 6 inches in diameter and the low-pressure 3 feet in diameter, the indicated horse-power of the four cylinders was 130·5, and that in the pumps 120·4, which is equal to an efficiency of 92 per cent. The piston speed per minute per engine was 86·9 feet; boiler pressure 101 lbs. per square inch above atmosphere, and the number of expansions 14·1. The consumption of coal per indicated horse-power was estimated at 1·7 lb.

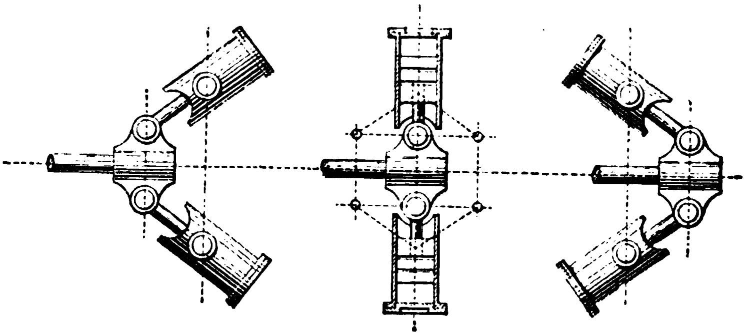
Position 1. Position 2. Position 3.

Fig. 360.—Worthington Compound Expansive Pumping Engine. Oscillating Cylinders to Equalise the Varying Pressure in the Steam Cylinders.

Power is transmitted very easily and safely for even long distances by means of compressed air, although only a small percentage, probably not more than from 20 to 30 of useful effect, is obtained. It is an exceedingly suitable and handy means of working pumping-engines, hauling-engines, or coal-cutting machines in the workings of a colliery.

There are occasions when hydraulic pumping-engines can be beneficially applied. Thus, where water is brought to the shaft at a point above the level of the main pumps and has to fall down to the pumps, its fall represents a power proportionate to the quantity and to the depth it falls. In any case where there is a surplus of power in the main pumping-engine a hydraulic may be used for pumping from the dip, the power-pipe being taken from the rising main, or a lodge-room in the shaft or even from the surface. Whatever the source of supply, it should be situated at least 10 fathoms above the discharge at the sump—the more the better for the working of the hydraulic. In many cases where water is available with a head sufficient to give the necessary pressure, small quantities may be raised from dip-workings to the sump. Where the main pumping-engine is already working at its maximum power the hydraulic system cannot be applied.

The usual arrangement for hydraulic pumps is to have the power-piston and the ram on one rod, one behind the other. The valves are specially made and are generally worked by a tappet arrangement. The surplus pressure on the piston over the ram gives its motion; the area of the piston × lbs. pressure of head is in excess of the area of the ram × lbs. pressure of its head + friction. The speed of ram is from 18 to 30 feet per minute. Friction is an important point in hydraulic pumps. The loss of head in forcing water through long pipes is inversely as the fifth power of their diameters. For instance, with a 2-inch and a 4-inch pipe of the same length, in order to force the same quantity of water through them, the difference of loss of head would be as 1,024 : 32; or, say 165 gallons per minute had to be discharged at a distance of 400 yards, the 2-inch pipe would require 180 feet of head, while the 4-inch pipe would only require 5·625 feet of head to overcome their friction. In proportion to the relative costs of the pipes, the size of the supply pipe, and more especially the discharge-pipe should be large. An objection to hydraulic pumps arises from the fact, that whether the pump has much or little work to do it takes the same quantity of water to do it. The water after doing its work in the hydraulic pump is discharged into the delivery and returned along with that pumped to the sump.

A very ingenious method of raising water from mines is by means of Moore's hydraulic mine-pump. An ordinary horizontal steam-engine placed on the surface works a double-acting water-ram, also on the surface. A strong wrought-iron tube connects the two ends of the water-ram case, and also from either end of this case a power pipe is carried down the shaft to the bottom, or to any point in the workings where it is convenient to fix the hydraulic pump.

The hydraulic pump consists of a double-plunger pump, having connecting rods projecting through glands at each end, and these are made a suitable length to form the plungers of two hydraulic rams, placed at either end of the main pump as shown in Fig. 361. The two power pipes after being carried down the shaft are connected to the outer end of each hydraulic ram. The main pump is double acting, and has two suction valves and connections to a single pipe from the sump or cistern and also two outlet valves and connections to the rising main. Protection against shocks caused by the stoppage of the moving power column, or by irregularities between the motion of the engine rams and the underground hydraulic rams, is obtained by a connection between the two power pipes, and by providing a relief-valve, which, when opened, allows the water to pass from one column to the other. The hydraulic ram gives motion to a small bell crank at the end of each stroke, which works the relief-valve. The reciprocating action of the engine rams on the surface is transmitted direct to the underground hydraulic rams, the water forming a rod which receives motion from the engine rams at one end, and transmits it to the hydraulic rams at the other, there being no valves whatever between. The water in the power pipes should be quite clean when first placed there, and is not afterwards changed. Care must be taken, however, to maintain the column of water solid, so as to completely fill the ram cases and the power pipes; and to provide against leakages, small inlet valves and pipes, connected to a cistern, are placed at a higher level. If the pump is a small one, the steam-engine on the surface may be direct-acting; the piston rod passing out through the back cylinder cover, and there connected to the water-ram. With large pumps, a small engine running at a high speed may be geared to the pump, as the water in the power pipes must not have a high velocity. The engine should have a heavy fly-wheel on the crank shaft. The power columns are worked at a high pressure—about 1,000 lb. to the square inch. The area of the rams and power pipes should be so proportioned that the rams travel at a speed of about 80 ft. per minute, and the speed of the water in the power pipes may be about 300 ft. per minute. The relative sizes of the plunger pump and the hydraulic rams must necessarily vary with the quantity of water and height to which it is to be raised, and also with the pressure at which it may be decided to work the hydraulic rams.

A great advantage from this system of pumping water out of mines, arises from its having no moving rods in—or beam overhanging—the pit, no leaky steam-pipes in the shaft; the only moving parts being at the engine on the surface, and at the hydraulic pump underground; and further, the power pipes take up less room in the shaft than pump rods. It is applicable to pump water in a shaft from any depth, or from dip workings which may be at a considerable distance from the shaft, or it may be used in a sinking pit. Where used to drain a sinking shaft the pump and pipes work in guides, and are suspended by two chains or rods from a hydraulic ram fixed on the surface, and lowered as the sinking proceeds.

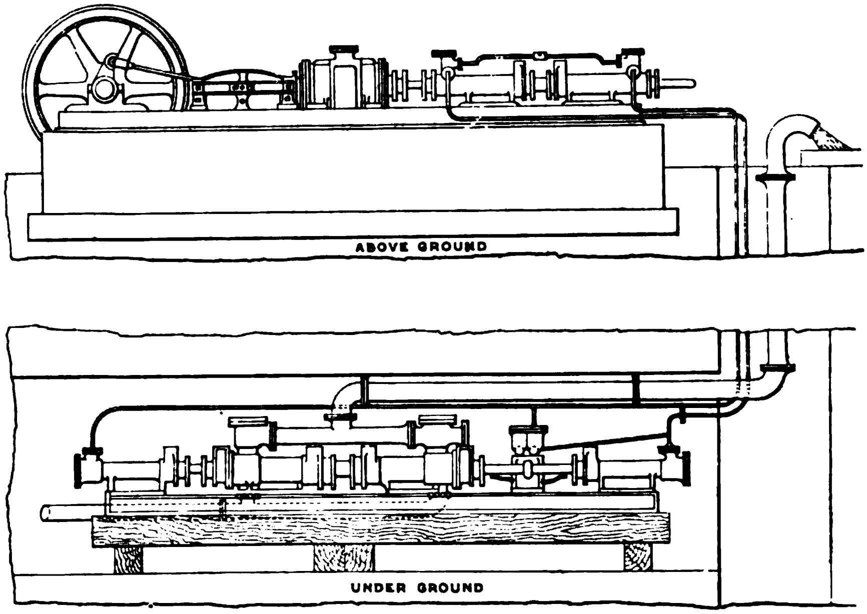

Fig. 361.—Moore's Hydraulic Mine Pump.

When a shot has to be fired in the bottom the whole apparatus is raised a few feet out of the pit bottom. The shot having been fired without injury to the pump, the latter is again lowered into its place in the sump.

A method of pumping water out of dip workings consists in making use of a hauling rope where one is at work, which may be made to give motion to the pump. For the same purpose a steam-engine may be placed at the shaft and communication made to the pump by means of a clip pulley and an endless rope. Wire-rope systems of pumping require considerable attention, are expensive to maintain, more especially where the rope is carried along undulating roads having many curves. For pumping limited quantities up from dip workings, a horse-pump and for still smaller quantities a hand-pump may be used. Pumps worked by horses, however, are not to be recommended. The net result from horses employed in this work is very small, as they can only be worked for a short period at a time.

The application of electricity to underground pumping is practical and in some instances may be economical. At the Trafalgar Colliery, Forest of Dean, Mr. Frank Brain has erected pumping plant which is working satisfactorily and economically. The plant consists of a single 16-inch cylinder engine with a 12-inch stroke working with about 35 lbs. steam pressure on the surface. The power is applied to a generator placed on the surface near the shaft, by means of a belt carried on two pulleys, one being on the crank shaft of the engine, the other on the generator. The pump and electro-motor (the latter being a machine for converting the energy of electric currents into the energy of mechanical motion) are underground at a distance of 1,650 yards from the bottom of the shafts, and the water is raised through a vertical height of 300 feet to the pit bottom, along slant roads.

The pump is a double 9-inch plunger with a 10-inch stroke and fitted with gear running six to one. The spur pinion is keyed on the same shaft as a 64-inch pulley which receives motion from a 14-inch pulley on the motor shaft by means of a leather link belt. When the motor runs 650 revolutions per minute, the pump makes 25 revolutions.

The electric current is conveyed to the motor from the generator on the surface by means of a copper cable 2,000 yards long, wrapped with compounded tape, and taken down the shaft in wooden boxes. It is not afterwards enclosed but carried along the side of the underground roads, being supported on earthenware insulators at intervals of about 10 yards. An old pit rope is used as a return cable. It is about 4 inches in circumference and secured to the road posts by staples. A small insulated copper wire, connected to a battery of eight No. 3 Leclanche cells, connects the engine-house on the surface with the pump-house underground, and through this is registered, upon a bell placed for the purpose in the former, each stroke of the pump. The same wire serves as a telephone line, so that conversation can be carried on between anyone in the engine-house with the man in charge of the pump underground. The water pumped is about 114 gallons per minute.

Syphons may be beneficially applied for conveying water in mines under certain circumstances. The principle of the syphon depends upon the fact that the atmospheric pressure will sustain a column of water 34 feet high. In its simplest form it is merely a tube bent like the letter U inverted, and having one leg longer than the other. If the short leg be inserted in the water to be drawn off and the air extracted from the inside of the tube, the atmospheric pressure acting on the surface of the liquid forces it up the tube. The liquid will consequently flow through the longer leg until the level of the liquid falls below the short end of the tube, or, if the liquid be water, until the surface falls to 34 feet below the highest point of the tube.

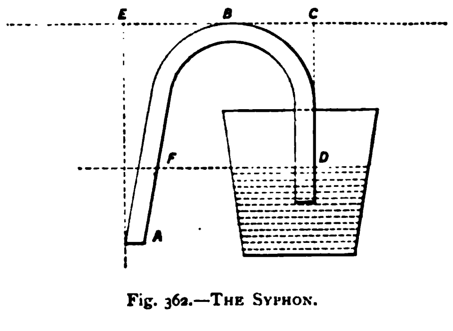

Fig. 362.—The Syphon.

Two circumstances limit the application of the syphon. Its highest point must be less than 34 feet above the surface of the water to be run off, and the delivery end of the pipe must be lower than the plane of the surface of the water to be removed. Suppose, in Fig. 362, A B D to be a syphon filled with water. The force acting on the vessel at D is the pressure of the atmosphere and the water is driver in the direction D B by that force, less the weight of a column of water whose height is D C, and the force acting on the water at the discharge A is the pressure of the atmosphere, less the weight of a column of water whose height is A E. From this we find the pressure tending to maintain the flow of water. Suppose the long leg in the syphon to be 30 feet and the short one to be 20 feet, the pressure maintaining the flow would be as follows:—

	Pounds per square inch.	Pounds per square inch.
Pressure of the atmosphere at D	14·7	
,, ,, the water column D B, 20 ft. × 12 × ·036 =	8·64	6·06
,, ,, the atmosphere at A	14·7	
,, ,, water column A B, 30 ft. × 12 × ·036 =	12·96	1·74
		4·32

The effective pressure acting on the water towards B is 6·06 lbs. per square inch and that at A, tending to uphold the column, is 1·74; so that the real effective pressure maintaining the flow is 4·32 lbs. per square inch. Any other lengths of legs may be calculated, and it will be seen that the less the resistance in the long leg, the more is the effective power increased. Fig. 363 shows the method of laying a syphon in a mine where it is desired to remove the water from the workings at A over the hill B, to C, where proper provision is made for its after disposal. The vertical height D B, should not exceed 28 or at the most 30 feet. Care should be taken in laying the pipes to make the joints perfectly air-tight. A clack must be fitted at A, to prevent a back-flow of water during the filling of the pipes. A tap must be placed at the delivery end C, which is kept shut during the filling. A small tap must be placed at the highest point B, and if the pipes have not a regular rise and fall each way from B (which should be always aimed at), a small tap or plug must be placed at every point in the pipes where they dip both ways. These are necessary for the air to pass out of the pipes as they are being filled. For this purpose these taps are opened, or if plugs they are taken out during the filling, and as the water rises in the pipes to each hole, the taps are closed or the plugs put in. When the pipes are full the tap at C is opened. There are different ways of filling the syphon. Where the syphon is near the shaft or circumstances admit of it, small pipes may lead from a cistern to the syphon, in which case it is merely necessary to turn the tap of the small pipes to fill the syphon. Another plan is to pump the water in from the supply end. A third method is by pouring water into the pipes at the highest point, but this usually entails leading the water there, and would only be adopted under exceptional circumstances.

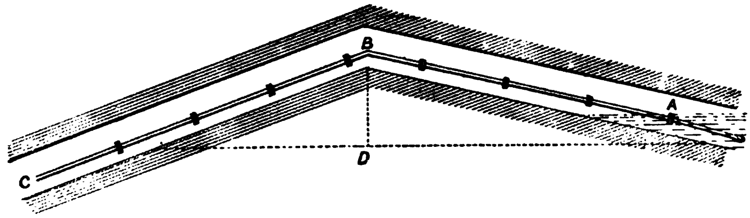

Fig. 363.—Syphon Working in a Mine.

In long syphons with pipes of large diameter, the water to be run off varying in quantity, a good plan is to sink a staple under the short leg and continue the pipe down into it, and it thus becomes self-regulating, because the water will not sink below the level of the pipe, though the feeder may vary. In cases where the feeder is not sufficient to keep the pipes running full, a float may be placed on the top of the water, and connected with a chain passing over a fixed pulley with a weight attached. As the water lowers the weight is drawn up by the float so as to partially stop up the intake-end of the pipes, and as the water rises again the float is raised, and the weight lowered, thereby allowing the syphon to run faster. In fixing syphons it should be borne in mind that the greater the fall in the delivery-leg, and the less the height of the short or suction end, the better it will work, as has been demonstrated by the explanation accompanying Fig. 362 and the example given. Although theoretically 34 feet is the height to which the water will rise, owing to atmospheric pressure, in practice that is never attained, and a syphon must be well laid and managed where the highest point of it is 30 feet above the suction-end of the pipes. Gas oozing from the ground, and rising in bubbles through the water beneath either end of the syphon, interferes with its action in the same way as leakages at the joints, and will sometimes cause the stoppage of the syphon. To prevent this a bell-mouthed piece of metal with a bottom to it may be placed in the water below the end of the pipe where such gas is, which will divert the bubbles, and prevent their entering the pipe. A good plan consists in placing an air-receiver at the highest point in the pipes, because the water absorbs more air at atmospheric pressure than it is able to retain in absorption under the reduced pressure at the summit of the pipes. An air-pump can then be used to extract the air, a tap being placed in the receiver.

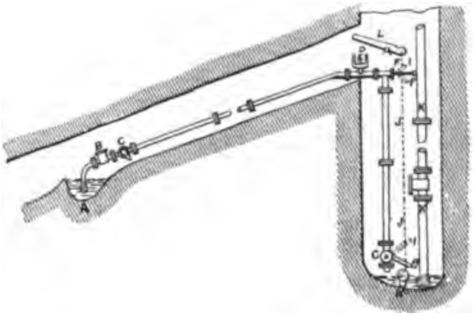

Fig. 364.—Syphon connected with Pit-pumps.

The practice of having pipes of a larger diameter at the delivery-end, as compared with the suction-end, is to be condemned. The consequence of such an arrangement is that the discharge must leave room for air to enter, which will interfere with the working of the syphon. A cure for this would be to let the delivery-pipe be taken to a box, and the discharge always be under water. The water would run over the sides of the box, and the entry of air be prevented. It is much better to lay pipes of a uniform size, or where this is not done, to let the delivery-end be the smaller pipes.

The sketch of syphon (shown in Fig. 364) is well suited for a large quantity of water, the pipes being from 3 to 6 inches diameter, made of cast-iron with flange joints. A is a suction pipe. B is a clack-valve opening outwards. C is a regulating tap which is used only when the water is nearly out or the pipes likely to come on "air." E is a T-pipe, on the top of which, but in the bottom of box, D, is an air-discharging valve; this box should be kept full of water to prevent any air from entering by the valve when the syphon is working. F is an inch pipe, with a tap turned by lever *f*, which is in the position of shut as shown; this pipe connects the syphon with K, the pit-pumps. G is a large tap, with a lever similar to *f*; in this position it is shown open. H is a weight to take down the levers *f* and *g* when they are in the position of I I. J, J, is a rod which connects levers *f* and *g* together.

If the pipes are empty, and are required to work, press down the lever, L, which will raise levers *f* and *g* to the position of I, I. By this action the tap, F, is opened, and the large tap, G, is shut when the water rushes through the tap, F, into the syphon, which fills it in a few minutes, the air blowing off at valve, E. Whenever the air is all expelled from the pipes the water overflows at box, D; then let go the lever, L, when the weight, H, will take down levers *f* and *g*, which shuts the tap, F, and opens the large tap, G; the syphon will now be working, and will do so while the pipes get water at A.

The following useful memoranda relate to water and pumps:—

From Molesworth.

1 Cubic foot of water = 62·4 lbs. = 6·2355 gallons.
1 Cubic inch of water = ·036 lbs.
1 Imperial gallon of water = 10 lbs. = 0·16 cubic foot = 277·274 cubic inches.
1 Cwt. of water = 1·8 cubic foot = 11·2 gallons.
1 Ton of water = 35·9 cubic feet = 224 gallons.

Pressure of Water per square inch at Different Heads.

P = Pressure in lbs. per square inch.
H = Head of water in feet.
$P = H \times {\cdot}4333$.
$H = P \times 2{\cdot}31$.
Pressure per square foot = $H \times 62{\cdot}4$.
Cubic feet of water × ·557 = Cwt. approximately.
" × ·028 = Tons "
1 Cubic foot of sea water = 64·14 lbs.
Weight of sea water = weight of fresh water × 1·028.

Delivery of Water in Pipes.

D = Diameter of pipe in inches.
H = Head of water in feet.
L = Length of pipe in feet.
W = Cubic feet of water discharged per minute.

$$W = 4{\cdot}72 \sqrt{\frac{D^5 H}{L}}$$

$$D = {\cdot}538 \sqrt[5]{\frac{L \times W^2}{H}}$$

Hawkesley's formula for the delivery of water in pipes is,
G = Number of gallons delivered per hour.
L = Length of pipe in yards.
H = Head of water in feet.
D = Diameter of pipe in inches.

$$D = \tfrac{1}{15} \sqrt[5]{\frac{G^2 L}{H}}$$

$$G = \sqrt{\frac{(15\,D)^5 H}{L}}$$

Molesworth's Rule for the Weight of Pipes.

D = Outside diameter of pipe in inches.
d = Inside diameter.

w = Weight of a lineal foot of pipe in lbs.
$w = k(D^2 - d^2)$,

and k =	2·45	for cast iron.
	2·64	,, wrought iron.
	2·82	,, brass.
	3·03	,, copper.
	3·87	,, lead.

The following are also from Molesworth :—

Useful Numbers for Pumps.

D = Diameter of pump in inches.
S = Stroke of pump in inches.
$D^2 \times S \times \cdot 7854 =$ cubic inches.
$D^2 \times S \times \cdot 002833 =$ gallons.
$D^2 \times S \times \cdot 0004545 =$ cubic feet.
$D^2 \times S \times \cdot 02833 =$ lbs. fresh water.

To find the Diameter of a Single-acting Pump.

L = Length of stroke in feet.
G = Number of gallons to be delivered per minute.
N = Number of strokes per minute.
D = Diameter of pump in inches.

$$D = \sqrt{\frac{G}{\cdot 034 \times L \times N}}$$

$G = D^2 \times \cdot 034\ L\ N.$

The reason of ·034 being used in this formula is that a cylinder 1 inch diameter and 1 foot long contains ·034 of a gallon.

NOTE.—This formula gives the net diameter of the pump-plunger; it is usual to increase the area of the plunger $\frac{1}{4}$th to allow for leakage, &c.

Pumping Engines.

G = Number of gallons to be raised in 24 hours.
h = Height in feet to which the water is to be raised.
H. P. = Horse-power required.

H. P. = $\frac{G \times h}{4,752,000}$. 20 per cent. must be added to overcome friction, &c., and 50 or 60 per cent. more is usually allowed for contingencies, making a total of 70 or 80 per cent. additional power.

The number 4,752,000 in this formula is merely derived from 33,000 (the number of pounds raised 1 foot high per minute per horse-power) × 24 × 60.

To find the Quantity of Water an Engine of given Horse-power will Pump.

H = Horse-power of engine.
F = Depth of pit in fathoms.
G = Quantity of water in gallons per minute.

$$G = \frac{H \times 550}{F}$$

$$H = \frac{F \times G}{550}$$

The reason of 550 being used in this formula is that 33,000 lbs. raised a foot high per minute equal 1 horse-power, and there being 10 lbs. of water to one gallon, and 6 feet in a fathom, we have $\frac{33,000}{10 \times 6} = 550$.

The quantity of water in any pipe 3 feet long is its diameter squared and divided by 10, the result being the number of gallons.

André, in his treatise on *Coal Mining*, gives the following rule to find the thickness necessary for cast-iron pipes :—

$t = \cdot 000144\, H\, D$, where H represents the head of water in feet, and D the diameter in inches.

Percy, in his *Mechanical Engineering of Collieries*, gives the following rule for condensing water :—

W = Weight of steam to be condensed.
T = Its temperature.
l = Its latent heat.
W' = Weight of injection-water required.
t = Its temperature.
t' = Temperature of the mixture.

Then $W' = \frac{(T + l - t')\,W}{t' - t}$, the latent heat of steam varying from 966 at atmospheric pressure to 939 at 15 lbs., and 920 at 30 lbs. above the atmosphere.

A few questions will now be given and worked out, in which some of the foregoing formulæ will be applied.

> *Question* 68.—What is the H. P. of an engine capable of raising 560 gallons per minute from a pit 150 fathoms deep ? Give the diameter of the working barrel, and state what velocity the water should travel at.

The diameter of the working barrel must depend upon the length of stroke at which the engine works, and upon the number of strokes per minute at which it is driven. As per the question the quantity of water is large, assume that we adopt a Cornish single-acting beam-engine to raise the water, working with a 10-foot stroke, and being driven at 5 strokes a minute, at which it may comfortably work, the pumps being single acting. Then $\sqrt{\frac{560}{\cdot 034 \times 10 \times 5}} = 18\cdot 15$ inches as the net diameter of pumps, but adding $\frac{1}{4}$th to the area to allow for leakages, we have $18\cdot 15^2 \times \cdot 7854 = 258\cdot 72$, the area, and $258\cdot 72 + \frac{258\cdot 72}{4} = 323\cdot 4$, and a working barrel whose area is 323·4 would have a diameter of $\sqrt{\frac{323\cdot 4}{\cdot 7854}} = 20\cdot 29$, or say, 20 inches as the diameter of the working barrel.

The velocity of the water in the pipes should not exceed 240 feet per minute, and it will be better to limit it to 200 feet per minute. In the above arrangement the flow of water in the pumps would be 50 feet per minute, which is much within the required limits. To find the horse-power of the pumping-engine remember that 1 H. P. is represented by the raising of 33,000 lbs. 1 foot high in a minute. In this case, since there are 10 lbs. to the gallon, and 6 feet to the fathom, there are to raise $560 \times 10 = 5,600$ lbs. through a vertical height of $150 \times 6 = 900$ feet in one minute, and therefore it would require $\frac{5,600 \times 900}{33,000}$ or $\frac{560 \times 10 \times 150 \times 6}{33,000}$ $= 152\cdot 7$ H. P., but to this must be added 80 per cent., 20 per cent. to overcome friction, and 60 per cent. for contingencies. Therefore, $152\cdot 7 + \frac{152\cdot 7 \times 80}{100}$ $= 275$ nearly, which is the horse-power of the engine required.

Question 69.—What size of pumping-engine would you erect underground to raise 30,000 gallons of water per hour from a depth of 300 yards?

Assuming that the piston and pump are double-acting, and that the piston will work at 250 feet per minute, which is a safe speed, and have an effective steam-pressure of 30 lbs. per square inch, $\frac{30,000}{60} = 500$ gallons per minute, and as the pump works at 250 feet per minute, $\frac{500}{250} = 2$ gallons for each foot the pump works. $2 \times 277\cdot274 = 554\cdot548$ cubic inches in each foot of the pump. $\frac{554\cdot548}{12} = 46\cdot212$ square inches area, allow say $\underline{12\cdot788}$ square inches for the pump-rod of about 4 inches diameter $= 59$ and $\sqrt{\frac{59}{\cdot7854}} = 8\cdot667$; say, to allow for leakages 10 inches as the diameter of pumps necessary. The pressure per square inch of water on a 300-yard column will be $300 \times 3 \times \cdot4333 = 390$ lbs. and $46 \times 390 = 17,940$ lbs. total pressure on the pump. $17,940 \div 30$ lbs. the steam pressure $= 598$ square inches, and adding $\frac{1}{2}$ for frictional allowances, $598 + 299 = 897$ square inches area, and $\sqrt{\frac{897}{\cdot7854}} = 33\cdot8$ inches diameter of the cylinder required for the pumping-engine; the stroke might be made 6 feet, and it would then work $\frac{250}{6 \times 2} = 20\frac{5}{6}$ revolutions per minute. The stroke of the pump should always be made as long as practical considerations will admit of.

Question 70.—What quantity of water could you raise and from what depth with an Underground Pumping-Engine, having a 30-inch cylinder and 5-foot stroke, double acting, working a 12-inch pump also double acting, the effective steam pressure on the piston being 30 lbs.?

Assume a piston speed of 250 feet per minute, or $\frac{250}{5 \times 2} = 25$ revolutions per minute. The pump-rod will be the same size as the piston-rod, and the piston-rod would be about ⅛th the diameter of the piston, or $\frac{30}{8}$ = say 4 inches or 12½ square inches area. Deduct this from the area of the pump, thus $(12^2 \times \cdot7854) - 12\frac{1}{2} = 100\cdot5$ as the effective area. The 30-inch cylinder has an area of 706·86, from which deduct the third for frictional allowances $= 706\cdot86 - 235\cdot62 = 471\cdot24$, say 471 square inches area on which the steam pressure operates; $471 \times 30 = 14,130$ available effective power. The effective area of pump being 100·5, $\frac{14,130}{100\cdot5} = 140\cdot5$ lbs. pressure which the pump can support and $\frac{140\cdot5}{\cdot4333} = 324$ feet height of column. And to find the quantity $100\cdot5 \times 12 = 1,206$ cubic inches in each foot $\frac{1,206}{277\cdot274} = 4\cdot4$ gallons per foot. $4\cdot4 \times 250 = 1,100$ gallons per minute. $1,100 \times 60 = 66,000$ gallons per hour, the quantity of water this engine would raise to a height of 324 feet.

In working out these two examples the method given by Mr. Percy, in his *Mechanical Engineering of Collieries*, has been adopted.

Question 71.—The feeders of water at a colliery are lifted at the rate of 3,000 tons in 12 hours by an engine going 12 strokes per minute, length of stroke 5 feet 6 inches. What diameter of pump will be required, and what is the feeder per minute in gallons?

To find the feeder per minute, which is dealing with the last part of the question first, 3,000 × 224 (the number of gallons in a ton) = 672,000 the number of gallons lifted in 12 hours, therefore $\frac{672{,}000}{12 \times 60} = 933{\cdot}\dot{3}$ gallons per minute, which is the feeder. Assuming the pumps to be single-acting and adopting Molesworth's rule as already given, $D = \sqrt{\frac{933{\cdot}\dot{3}}{{\cdot}034 \times 5{\cdot}5 \times 12}} = 20{\cdot}4$; or the same result may be arrived at as follows, $D = \sqrt{\frac{933{\cdot}\dot{3} \times 277{\cdot}274}{5{\cdot}5 \times 12 \times 12 \times {\cdot}7854}} = 20{\cdot}4$ as before (277·274 being the number of cubic inches in a gallon). This gives the net diameter of the pump plunger, and it is usual to increase the area of the plunger $\frac{1}{4}$th to allow for leakage, &c. Therefore $20{\cdot}4^2 \times {\cdot}7854 = 326{\cdot}74$ area of plunger and $326{\cdot}74 + \frac{326{\cdot}74}{4} = 408{\cdot}4$ which is the area the pumps should be, and $\sqrt{\frac{408{\cdot}4}{{\cdot}7854}} = 22{\cdot}804$ or say 23 inches, which is the diameter the plunger should be to pump 3,000 tons of water in 12 hours by an engine going 12 strokes per minute with a 66-inch stroke.

Question 72.—Describe the general arrangements, diameter of cylinder, length of stroke of pump, and pressure of steam by which 2,000 tons of water may be raised from a depth of 200 fathoms in 24 hours.

$\frac{2{,}000 \times 224}{24 \times 60} = 311$ gallons to be pumped per minute, but this would allow no margin for the engine to rest through accident, and as the depth is great, entailing many working parts, it would not be prudent to assume that the engine can go on continuously, and the colliery may have no storage arrangements. Assume that the engine works 16 hours out of the 24, then $\frac{2{,}000 \times 224}{16 \times 60} = 467$ gallons to be pumped per minute. It would be better to choose an engine placed on the surface to deal with this water, because it would be necessary to use large pumps of 18 or 19 inches diameter. If an engine were placed underground and one column of pumps used, the thickness of metal in the bottom part of the column would be excessive. For 19-inch pumps in a 200-fathom pit, it must be ${\cdot}000144 \times 200 \times 6 \times 19 = 3{\cdot}283$ inches, a thickness of metal for pumps never adopted and not practicable, and even for a 12-inch set it would require to be over 2 inches, a size not often exceeded for pumps.

The most approved engine to do the work would be Davey's Compound Differential Pumping Engine, but, assuming that a Cornish single-acting engine with beam overhead be adopted and that it works with a 10-foot stroke, it may be driven comfortably and steadily at 5 strokes a minute. To find the size of pumps then $\sqrt{\frac{467 \times 277{\cdot}274}{10 \times 12 \times 5 \times {\cdot}7854}} = 16{\cdot}57$ diameter, or allowing $\frac{1}{4}$th increase of area for leakage, it means 18·57, or say 19 inches as the diameter of the working barrel and plungers. In arranging the sets in the shaft divide them into 5 sets of 40 fathoms each; and, for reasons already given, the bottom one should be a lifting set, the others being all forcing sets.

To find the horse-power of the engine, assuming that there is an effective steam pressure on the piston of 30lbs. $\frac{467 \times 10 \times 200 \times 6}{33{,}000} = 169{\cdot}82$ or say 170 horse-power, but to which must be added 80 per cent., 20 per cent. to overcome friction and 60 per cent. more for contingencies $170 + \frac{170 \times 80}{100} = 306$ as the horse-power of the engine; and to find the diameter of the cylinder $\frac{306 \times 33{,}000}{10 \times 5 \times 30 \text{ (press. of steam)}}$ $= 6{,}732$ the area of the piston in inches and $\sqrt{\frac{6{,}732}{\cdot 7854}} = 92{\cdot}58$ or say 93 inches as the diameter of the piston, for the Cornish Beam Pumping-engine to do the required work.

> *Question* 73.—How many gallons are there in 1,200 tons of water? Describe the general arrangement and size of pumps for lifting that quantity daily from a pit 80 fathoms deep.

$1{,}200 \times 224 = 268{,}800$ gallons. If the pumping-engine were placed on the surface, the general arrangement would be to divide the 80 fathoms into two, and have a lifting set of 40 fathoms at the bottom, and a forcing set at the top, or a short lifting set of 10 or 15 fathoms might be fixed at the bottom, and the remaining height of 70 or 65 fathoms divided into two even divisions for two forcing sets above. The pumps would be single-acting, and to find their diameter, $\frac{268{,}800}{24 \times 60} = 186{\cdot}\dot{6}$ gallons per minute to be pumped. Assuming the engine to have a 10-foot stroke and to work 5 strokes a minute $\sqrt{\frac{186{\cdot}\dot{6}}{\cdot 034 \times 10 \times 5}}$ $= 10\frac{1}{2}$, to which add $\frac{1}{4}$th to the area, or say 12 inches as the diameter of pumps, if the engine worked continuously.

> *Question* 74.—Describe the class of engine best adapted for the above work, size of cylinder, length of stroke, and strokes per minute.

It would be better to use an underground pumping-engine, if there were no danger of its being drowned. The Differential Engine might be applied, or a Worthington steam pump, which would be double-acting as would the engine, and the engine should be condensing. Taking a piston speed of 250 feet per minute, and an effective steam-pressure of 30lbs. then $\frac{187}{250} = \cdot 75$ of a gallon for each foot the pump works. $\cdot 75 \times 277{\cdot}274 = 208$ cubic inches in each foot of the pump, $\frac{208}{12} = 17{\cdot}\dot{3}$ square inches area, say the pump-rod is 2 inches in diameter = 3 inches area, and add 3 to $17{\cdot}\dot{3} = 20{\cdot}\dot{3}$, and $\sqrt{\frac{20{\cdot}\dot{3}}{\cdot 7854}} = 5{\cdot}088$ inches diameter of pump, and to allow for leakage make them 6 inches. The pressure per square inch of water on an 80-fathom column will be $80 \times 6 \times \cdot 4333 = 208$, and $17{\cdot}\dot{3} \times 208 = 3{,}598$ lbs. total pressure on the pump $\frac{3{,}598}{30 \text{ (steam press.)}} = 120$ square inches, and adding $\frac{1}{2}$ for friction, $120 + 60 = 180$ square inches area, and $\sqrt{\frac{180}{\cdot 7854}} = 15\frac{1}{8}$ as the diameter of the cylinder in inches, the stroke might be made 3 feet, and it would then work $\frac{250}{3 \times 2} = 41\frac{2}{3}$ revolutions per minute.

> *Question* 75.—State the advantages in a deep shaft of having a series of lifts instead of one long lift to the surface.

It is not practicable to have unduly long lifts of pumps; they are limited by the strength of material required. If it was contemplated to put one long lift in a deep pit, on an estimate being made of the thickness needed for the pipes and the size of the spears it would be found impracticable to adopt that plan. Great wear would result to a bucket of one long lift, requiring constant changing and repairing; broken spears, especially near the bottom of the lift, would be a source of great annoyance and expense, and the arrangement would not give facility for counterbalancing the engine.

> *Question* 76.—A pumping-engine goes 14 strokes per minute for 13 hours a day, working two 18-inch sets with a 49-inch stroke. What is the feeder per minute?

$\frac{18^2 \times {\cdot}7854 \times 49}{277{\cdot}274} = 45$ gallons per stroke for each set, 90 gallons per stroke for the two, and 90 × 14 = 1,260 gallons as the feeder pumped per minute; but from the question it would appear that this quantity pumped per minute for 13 hours represents what is made in the pit during the 24 hours, and if so the feeder in the pit would be $\frac{1{,}260 \times 60 \times 13}{24 \times 60} = 683$ gallons per minute.

> *Question* 77.—Describe the pulsometer.

The following description of the pulsometer is taken partly from the *Colliery Guardian* of Sept. 29th, 1876, and partly supplied by the Pulsometer Engineering Co., Nine Elms Ironworks, London.

This pump may be said to raise water by the direct pressure of steam upon its surface, and then to turn this same steam to account in forming the vacuum necessary for effecting the suction for the next lift. This is accomplished without the intervention of any piston or plunger. The invention is, in fact, but a perfection of the principle of Thomas Savery in 1698, and by him carried out so clumsily that the idea seems to have been abandoned until now; and engineers have been content in the meantime to interpose between the power and its work a complicated set of working parts, adding greatly to the first cost of the pump, introducing a heavy item for repairs and maintenance, and diminishing the duty on account of friction.

The pulsometer as now constructed, consists of a hollow casting, called the body, composed of two pear-shaped water-chambers, with their necks joining above, between which is the air-vessel, while the discharge chamber is a kind of offshoot from the lower portion of the body. The steam admission-valve is bolted on at the top, at the junction of the ends of the water-chambers; and the suction pipe is similarly bolted on at the bottom. The construction will, however, be better understood on reference to the drawing, Fig. 365, which is a vertical section through the pulsometer. A A are the two water-chambers surmounted by a separate casting containing the steam ball-valve I, fitted so as to pass alternately between the seats formed in the junction, and connected at top with the steam pipe K. The water-chambers terminate at bottom in a suction-chamber, to which the suction pipe is bolted, while the suction valves E E, with their seats, are arranged at an angle of 45 degrees between them. Covers for closing the spaces are left in the casting for the insertion of the valves, and also for the removal of any obstructions. The air-vessel for rendering the flow constant is situate between the necks of the water chambers, and is in direct communication with the suction.

A discharge chamber common to the two water-chambers A A, and leading to the discharge pipe D, is provided, and this contains one or two valves, according to the purpose to be fulfilled by the pump, which are shown in dotted lines in the figure. Small air-cocks are screwed into the cylinders and air-chamber, for use as described hereafter.

The pump being filled with water, either by pouring water through the plug hole in the chamber or by drawing the charge, according to the printed directions, is ready for work. Steam being admitted by slightly opening the stop-valve, which allows it to enter the steam-pipe K, it passes down that side of the steam neck which is left open to it by the position of the steam ball, and presses upon the small surface of water in the chamber which is exposed to it, depressing it without any agitation, and consequently, with but very slight condensation, and driving it through the discharge opening and valve into the rising-main.

The moment that the level of the water is as low as the horizontal orifice which leads to the discharge, the steam blows through with a certain amount of violence, and being brought into intimate contact with the water in the pipes leading to the discharge chamber, an instantaneous condensation takes place, and a vacuum is in consequence so rapidly formed in the just emptied chamber, that the steam ball is pulled over into the seat opposite to that which it had occupied during the emptying of the chamber, closing its upper orifice and preventing the further admission of steam, thus allowing the vacuum to be completed; water rushes in immediately through the suction pipe, lifting the inlet valve E, and rapidly filling the chamber A again. Matters are now in exactly the same state in the second chamber as they were at the commencement in the first chamber, and the same results ensue. The change is so rapid that, even without an air vessel on the delivery, but little pause is visible in the flow of water, and the stream is, under favourable circumstances, very nearly continuous. The air-cocks are introduced to prevent the too rapid filling of the chambers on low lifts, and for other purposes, and a very little practice will enable any unskilled workman or boy so to set them by the small nut that the best results may be obtained. The action of the steam ball is certain, and no matter how long the pump may have been standing, it will start as soon as dry steam is admitted.

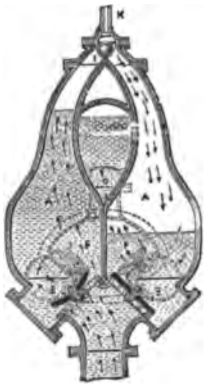

Fig. 365.—The Pulsometer.

The smallness of the quantity of steam used in raising the water and effecting the condensation, is proved by the very slight increase of temperature acquired by the water raised.

It will be noticed that great simplicity of form and fewness of working parts are attained in the pulsometer; there are, in fact, only 5 moving parts, the steam-admission valve, the two suction or foot valves, and the two delivery valves. These, again, are made as simple and efficient as possible. The steam-ball valve, when once made spherical, wears itself and its seats true, as it turns in its bed at every pulsation. The foot and delivery valves may be of the same character, though larger, with suitable guards, but for heavy work the flap-valve is found preferable. It consists of a cast-metal shoe, planed on its lower surface, and having a bored recess into which is fitted a seat made of hard wood boiled in oil, the end of the grain being the part exposed to wear. This surface is turned true, and the wood is fastened in the shoe by a bevelled brass ring secured with pins. The flap works in a bored bearing, and is kept in its place by a guard cast on the cover of the chamber, or by a small cover. Its play is regulated by a stop on the chamber cover, and noise is prevented by the wood lining of the horn of the valve. In some instances the flap valves are furnished with wooden faces. India-rubber is also used for some purposes, either in the form of a flat disc or the hollow valve designed by Perreaux.

In the majority of pumps, grid valves are employed, as shown in Fig. 365.

This is a type that has proved itself most convenient for general work, both on account of its original simplicity and durability and also of the facility it offers for repairs, it being but a matter of a few minutes for an unskilled labourer to take off the nut and guard, and to put on a fresh rubber. These grids are of brass as well as the spindle and nut, and are secured by bolts properly pitched round the circumference.

The ball valves, used instead of the clack or grid valves, are not advisable in pumps above the No. 3 or 4 size. They are made (1) all iron with iron seats; (2) gun-metal with phosphor bronze seats; (3) rubber balls with iron cores and gun-metal or iron seats.

As there are no cylinders, plungers, or complicated internal parts, but only the steam valve, and the suction and delivery valves of large area, this pump is very suitable for pumping water mixed with grit which would choke or wear out other pumps. Again, as it requires no fixing, but may be hung on a rope or chain and may be raised or lowered by a pair of blocks, it is well fitted for sinking pits. All that is required is to provide a boiler at a reasonable distance and lead the steam through a flexible hose or through a telescopic copper steam-pipe. The suction may be either of jointed cast-iron pipes terminating in a windbore, or consist of a flexible tube; the delivery pipe may also be flexible or have length after length added as the pulsometer goes down. The great feature is that when once the pump is started and the air valves adjusted, it may be left to itself, and will continue to work as long as it is kept supplied with steam.

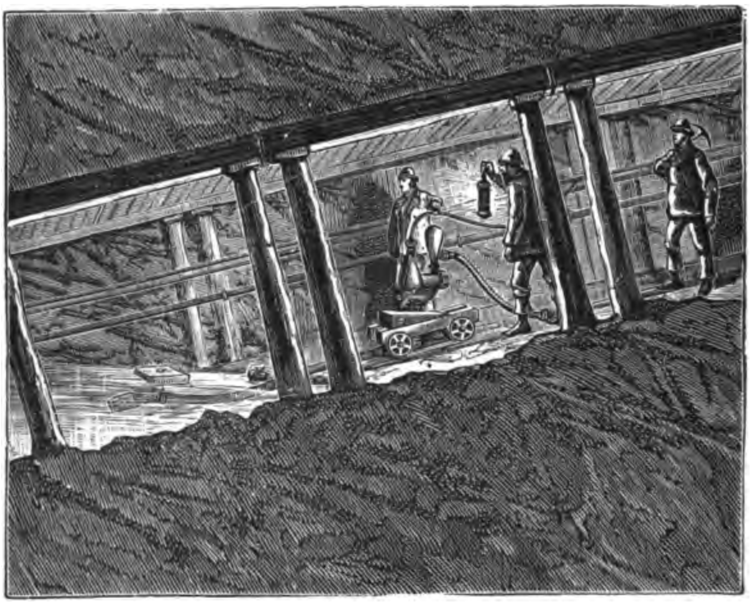

Fig. 366.—The Pulsometer Working in a Mine.

The pulsometer will pump water, and many other liquids and semi-liquids, to a total height of from 70 to 80 feet, or under special circumstances to much greater heights. The sizes (1) and (2) are best adapted to a suction not exceeding 6 to 10 feet; the sizes (3) and (4) to a suction not exceeding 8 to 12 feet; sizes 5 and upwards to a suction not exceeding 10 to 15 feet. Each size is adapted to a vertical discharge of from 60 to 70 feet. These figures are given as a guide, but may be modified by circumstances. The length of horizontal suction and discharge is not very material if sufficient size of pipe be employed to obviate the friction.

The steam pressure at the pump for lifts from 20 to 40 feet should not be less than from 20 to 30 lbs. per square inch, and for lifts from 40 to 80 feet not less than from 30 to 50 lbs.; for higher lifts greater steam pressure is necessary. For low lifts the pulsometer can be worked with exhaust steam.

The pulsometer may render most useful service in draining the underground workings of water of a very dirty or gritty nature, as shown in Fig. 366. For this purpose it is mounted on a frame running on wheels to suit the gauge of roadway. The steam-pipe is laid from the boilers to it, which may be at the surface. The discharge pipe is taken to the pit sump, or if that be at too great a vertical height, to an intermediate lodge-room or into the suction pipe of an upper pulsometer from which it is afterwards pumped on. The absence of exhaust steam, and the regularity of its working without constant attention, are strong recommendations to its use.

In the simple pulsometer it is impossible to obtain an expansive action of the steam, because the upper valve, whether a ball or other contrivance, must, from its construction, be allowing steam to pass into one or other chamber during the whole time the pump is at work. Where the work is constant, and there is a good margin between the pressure of steam and the pressure of water in the column, pulsometers (more particularly those above No. 4 size) may have the "Grel" automatic cut-off arrangement, which is patented by the Pulsometer Engineering Company. An economy of from 40 to 50 per cent. of the total steam consumption is said to be effected by this arrangement, which depends on the employment of a secondary cut-off valve, in addition to the distributing valve, so that a long interval may intervene between each pulsation, during which no steam can pass through the steam-pipe, although the work of pumping is going on continuously.

> *Question* 78.—Suppose a sinking pit to be 200 yards deep and a barrel in the form of a frustum of cone, whose inner dimensions are 4½ feet deep, 42 inches diameter at bottom, and 51 inches diameter at top, making its ascent and descent every 60 seconds, coming up full every time; what quantity of water in imperial gallons and in tons will be drawn in 24 hours? And, further, supposing another barrel, whose inner dimensions are 6·75 feet deep, 57 inches diameter at bottom and 45 inches diameter at top, making the same descents in the same time, the pit being the same depth as the one above described; what quantity of water would be drawn in one week from the pit in imperial gallons and tons?

Let D = diameter of water-barrel at top in feet.
„ d = „ „ „ bottom, in feet.
„ h = height „ „ in feet.
„ c = capacity „ „ in cubic feet.

$$\text{Then } c = \left\{ (D^2+d^2) + (D.d)\right\} h \times \cdot 2618 \quad (1)$$

if we denote the capacity of the barrel in imperial gallons by "g" then $g = 6{\cdot}24 \times c$. Hence is obtained

$$g = \left\{(D^2 + d^2) + (D.d)\right\} h \times 1{\cdot}632 \quad (2)$$

Again, to find the weight of water contained in the barrel, let "w" denote the weight in cwts.; then

$$w = \left\{(D^2 + d^2) + (D.d)\right\} h \times \cdot 14571 \quad (3)$$

Applying (2)

$$g = \left\{(4{\cdot}25^2 + 3{\cdot}5^2) + (4{\cdot}25 \times 3{\cdot}5)\right\} 4{\cdot}5 \times 1{\cdot}632 = 331{\cdot}85$$

As the capacity of the barrel is 331·85 gallons there would be wound in 24 hours, 331·85 × 24 × 60 = 477,864 gallons, or expressed in tons $\frac{477,864}{224} = 2,133\cdot32$ tons, which might have been found by formula (3). Applying (2) to the second example

$g = \{(3\cdot75^2 + 4\cdot75^2) + (3\cdot75 \times 4\cdot75)\}\ 6\cdot75 \times 1\cdot632 = 599\cdot68$ gallons as the capacity of the barrel: therefore, in a week of 6 days there would be hauled 599·68 × 24 × 60 × 6 = 5,181,235·2 gallons, or expressed in tons $\frac{5,181,235\cdot2}{224} =$ 23,130·5 tons, or the same result may be got by applying formula (3).

Question 79.—The water in a pit is hauled up the shaft 350 yards deep in two tanks each holding 350 gallons, one ascending the other descending; what horse-power is the engine which delivers a tank of water every three minutes at the surface, not including the time of stoppage?

The speed of the load is 350 feet per minute and its weight 350 × 10 = 3,500 lbs. $\therefore \frac{350 \times 3,500}{33,000} = 37\cdot12$ effective horse-power according to the work done, and assuming the engine yields 50 per cent. of effective work the horse-power of the engine would be as 50 : 100 : : 37·12 : 74·24 or 74·24 horse-power.

Question 80.—What are the special advantages and disadvantages of steam pumping-engines placed underground?

The principal advantages are,—a great saving of cost and maintenance, less room is taken up in the shaft, simplicity in construction, and an equal flow of water in the pipes. The disadvantages are, danger of flooding arising from a breakdown of the engine and access to it thus prevented, and it is not suitable for depths below 150 or 160 fathoms.

Question 81.—Upon what principle does the action of the lifting pump depend, and which is the proper position for the bucket to be placed in?

The action of the pump depends upon atmospheric pressure which is equal to the weight of a perpendicular column of water 34 feet high. On the bucket being drawn up in the working barrel its tendency is to form a vacuum beneath it. But the atmospheric pressure causes the water to fill such vacuum and follow the bucket for a distance not exceeding 34 feet. In practice the bucket must never be raised more than 32 feet above the surface of the water, the usual distance from the windbore to the bottom of the working barrel is from 9 to 12 feet, and lifting or forcing pumps are arranged so that the suction shall not exceed 20 feet in height. This allows a bucket having a 12-foot stroke (which is not often exceeded) to be as effective at the top of its stroke as at the bottom, when it is close to the suction pipes.

CHAPTER XI.

THE GASES OF THE COAL MINE: VENTILATION.

Compressibility and Elasticity—Specific Gravity—Diffusion—Chemical Names, Symbols, and Formulæ—General Character and Composition of Air: Oxygen, Nitrogen, Hydrogen, Methane, Carbon dioxide, Carbon monoxide, Hydrogen Sulphide—Explosions—Explosives—Natural Ventilation—The Furnace—The Waterfall—The Steam Jet—The Struvé Ventilator—Nixon's Ventilator—The Fabry Ventilator—The Lemielle Ventilator—Cook's Ventilator—Root's Ventilator—Guibal Fan—Waddle Fan—The Schiele Fan—The Capell Fan—The Medium Fan—Fan Arrangement for a Winding Shaft—Two Separate Engines to Drive Fan—Duplicate Fan and Engine—Ascensional Ventilation—Stoppings to Direct the Underground Air-currents—Advantages of Air-splitting — Regulators — Doors on Travelling Roads — Air-crossings — Brattice — Velocities of Air-currents in the Roads and Shafts—Relative Sizes of Downcast and Upcast Shafts—Anemometer and Measuring the Volumes of Air—Thermometer—Hygrometer—Barometer—Effect of Diminished Atmospheric Pressure in Seams yielding Fire-damp—Barometric Rules—Water-gauge—Motive Column and Rule to ascertain it—Horse-power of Ventilation—Useful Effect of Ventilating Fans—Theoretical Quantities of Air displaced by Fans—Rules for Total Volumes of Air required at Collieries.

GENERAL PROPERTIES OF GASES.

Compressibility and Elasticity.—Atmospheric air is the substance in which the general characters of gases were first studied.

It was found that air was a compressible and perfectly elastic substance. Its volume is reduced by pressure in proportion to the amount of the pressure which is exerted upon it, or in other words its volume varies inversely as the pressure. The original volume is regained as soon as the pressure is removed.

The volume of air which is under constant pressure also varies with the temperature, increasing at a fixed and uniform rate as the temperature rises, and diminishing at the same rate as the temperature falls.

Specific Gravity, or Density.—The discovery of other gases was followed by the study of their properties, which were found in the above respects to be similar to those of air. As soon as methods of weighing gases were discovered, however, it was found that they varied in their weight; or, to speak more correctly, they varied in their density or specific gravity.

Thus, if a vessel, which can be closed by a stop-cock, is weighed after being exhausted by an air-pump, and is then filled with air and once more weighed, the weight of air which it contains can be ascertained. In a similar way the weight of any other gas which fills the vessel is ascertained, and this is found to differ from the weight of the same volume of air. The weight of hydrogen gas which fills the vessel is about $\frac{1}{14}$th that of the air, whilst the weight of carbonic acid is about $1\frac{1}{2}$ times as great as that of the air. The number which expresses the relation of the weights of equal volumes of a gas and of air is called the specific gravity of the gas. It will be shown later on that there are many advantages in expressing this relation to hydrogen instead of to air.

Diffusibility.—The difference in the weights or specific gravities of gases is closely connected with another general property of gases, namely, the rate of their diffusion or spontaneous mixture.

All gases have the property of intermingling with one another, when they are in contact. From this fact it is inferred that a gas consists of a number of rapidly moving particles, and that the motion of the particles causes the gases to intermingle. Further, since a light gas intermingles or diffuses more rapidly than a heavy gas, it is inferred that the particles or molecules of the light gas move more rapidly than those of a heavy gas.

It is important to remember that a layer of light gas resting above a layer of a heavier gas does not remain permanently separate, as a layer of oil would do over a layer of water. Both gases intermingle, and after a sufficient time form a uniform mixture, from which the gases do not again separate. In fact, owing to this diffusive power of gases, a mixture of gases of widely different specific gravities shows no tendency to separate into layers: it tends rather to become more uniform in its condition of mixture.

Gases possess the tendency to mingle also with vapours; accordingly a gas which has been exposed to water, or to a moist surface, will always contain water vapour, which has evaporated from the liquid, and has mingled with the air by the process of diffusion. The maximum quantity of vapour which can mingle with a gas varies with the temperature, and increases as the temperature rises.

Solubility.—Gases vary in the extent to which they can dissolve in water. Each gas has its own degree of solubility, and for each gas the solubility increases as the temperature of the water is reduced. At a boiling heat, gases are expelled from their solution in water, but in the case of many gases, the water must be actually boiled for some time before the gas is completely expelled.

Chemical Nature.—The number of gases which are of importance to the miner is not large. The list includes some which consist of one constituent, or of one kind of matter only, such as oxygen, hydrogen, nitrogen; these are termed elementary gases or chemical elements. Other gases are composed of two or more constituents, or forms of elementary matter or elements: air, carbonic acid, and marsh gas or firedamp belong to this class; these are known as compound gases.

Chemical Notation.—It will be found of advantage to acquire some knowledge of the meaning of chemical names, and of chemical symbols and formulæ, from an elementary treatise on the science. An appreciation of these matters will be of considerable assistance in understanding the composition of substances, and the nature of important changes such as those occurring in the processes of combustion, of breathing, and of explosion.

A chemical name always declares as far as is possible the composition of a substance: thus carbon dioxide, which is the chemical name for chokedamp, declares that this gas is composed of carbon and oxygen, and further indicates the proportions in which these constituents are present in the gas.

A chemical formula declares precisely what is the composition of the substance, since the formula is made up of the chemical symbols, or initial letters, of the constituent elements. The formula CO_2 for carbon dioxide shows that the gas is composed of carbon (C) and of oxygen (O). For chemical purposes also a weight is denoted by each of these symbols. These combining or atomic weights of the elements can be found by reference to any chemical text-book. Thus C always denotes 12, and O always denotes 16. Hence the proportions of the constituents in carbon dioxide (CO_2) are 12 of C and 16×2 of O: or 44 by weight of this gas contains 12 of carbon and 32 of oxygen. A similar process of calculation enables the composition of any compound substance to be derived from its chemical formula.

In the case of gases a further useful fact can be deduced from the chemical formula. If the specific gravities or densities of gases are compared with hydrogen instead of with air as a unit, the specific gravity of the gas can be simply derived from its symbol or chemical formula.

The specific gravity of an elementary gas is identical with its atomic or combining weight. Thus the combining weight of oxygen is 16, and oxygen is 16 times as heavy as hydrogen: for nitrogen this number is 14. These numbers give the comparative weights of oxygen and nitrogen.

The specific gravity of a compound gas like carbon dioxide, referred to hydrogen as unit, is calculated by simply halving the total weight of the constituents of its formula. Thus the weight represented by CO_2 is 12 + 32 or 44; and this gas is therefore $\frac{44}{2}$ or 22 times as heavy as hydrogen.

Hence the relative weights of hydrogen, oxygen, nitrogen and carbon dioxide are 1 : 16 : 14 : 22. Since air has a density of 14·5 on this scale, it is only necessary to divide the above numbers by 14·5 in order to obtain the density of these gases in relation to air as unit.

It will thus be seen that the use of a correct chemical name furnishes a knowledge of the composition of substances, while chemical formulæ, supplemented by a reference to a table of atomic weights, enable the composition by weight of substances to be calculated; and in the case of a gas they enable the specific gravity to be calculated with the greatest ease.

The general character and composition of a few important gases is described below.

Atmospheric Air.—Density = 1, or referred to hydrogen = 14·43.

Air consists principally of the elementary gases nitrogen and oxygen, but it always contains in addition some carbonic acid gas. Water vapour is invariably present in air, but its proportion varies widely according to circumstances.

Leaving the proportion of moisture and carbonic acid out of consideration, the average composition of fresh air by volume is represented by the following percentage statement:—

Composition of dry, pure, fresh Air.

By measure.		*By weight.*	
Nitrogen	79·1	Nitrogen	76·23
Oxygen	20·9	Oxygen	23·77
	100·0		100·00

Taking a rough average for the proportion of moisture, and entering the usual and fairly constant proportion of carbonic acid in fresh air, the percentage composition will stand thus:—

Composition of fresh Air.

By measure.	
Nitrogen	77·95
Oxygen	20·61
Moisture	1·40
Carbonic Acid	0·04
	100·00

In this statement of composition all occasional constituents are omitted. The minor constituent, discovered recently and named Argon, is also not inserted, since its proportion certainly does not exceed 1 per cent. of the atmosphere; and its small proportion and its wholly negative and indifferent character cause it to be probably of little importance in connection with the changes hereafter referred to.

The uniformity in composition of fresh air in open places is strikingly constant. The only constituent which varies in quantity to any appreciable and important extent is water vapour. The oxygen, nitrogen, and carbonic acid are almost invariable in their proportions.

This uniformity of composition is maintained in spite of the changes which are constantly being caused in the air by burning and by breathing, and by the nutrition of plants through their green leaves. Any rapid change in the composition of air is prevented by the large mass or volume of the atmosphere which surrounds the earth to a depth variously estimated at from 45 to 100 miles. Since all parts of this vast atmosphere are constantly being intermingled with one another by the slow process of diffusion and by the more rapid action of winds, it is impossible that any local change of composition should rapidly affect the composition of the whole atmosphere. An alteration in the course of ages in the whole atmosphere is prevented by the fact that the changes produced by the processes of combustion and by breathing, which are constantly replacing oxygen by carbonic acid, are counteracted by the nutrition of plants, which by means of their leaves are constantly converting the carbonic acid of the air into oxygen.

The effect of breathing upon the composition of air is shown by the following statements of percentage composition by volume:—

Inspired Air.		Expired Air.	
Oxygen	20·81	Oxygen	16·03
Nitrogen	79·15	Nitrogen	79·58
Carbonic Acid	0·04	Carbonic Acid	4·38

Fresh air, of the composition stated above, is necessary for healthy respiration. Such air is only obtained in the open, or in closed spaces which are properly ventilated with fresh air. In all closed spaces the composition of air becomes changed by admixture with other gases, and by removal of oxygen; it is necessary therefore to constantly replace the impure air by fresh air: and it is necessary to ascertain that such replacement or ventilation is sufficient and ample. The gas which commonly renders air impure is carbonic acid, which is introduced by breathing or by burning; in proportion as this gas is introduced, oxygen is removed.

It is generally considered that if the air is sufficiently pure not to impair the combustion of a candle, the air is respirable. This standard errs on the side of safety, when carbonic acid or nitrogen only are the contaminating gases, or when reduction of oxygen only is to be taken into account. It must be borne in mind, however, that the candle test does not indicate the presence of directly poisonous gases, such as carbonic oxide.

Chemists have established the fact that the constituent elementary gases in air are simply mixed together, and are not united to form a compound gas. Hence the oxygen of the air is free to maintain combustion and respiration; when it is present in compound gases, oxygen does not possess these useful powers, or possesses them only to a very limited extent.

OXYGEN.—Chemical symbol O. Atomic weight = 16. Density = 16, or if referred to air = 1·1018.

This elementary gas is the important constituent of air which maintains all ordinary processes of combustion, and which maintains life when the air is breathed. Accordingly processes of combustion become more rapid and vigorous when oxygen is supplied more rapidly or in larger proportion; and respiration becomes insufficient to support life vigorously, or even to maintain it at all, when air is deprived of its oxygen until only 10 per cent. of that gas is present.

The process of preparing oxygen, and of making some striking experiments to demonstrate its properties, will be found in chemical text-books.

Oxygen is now prepared in large quantities from atmospheric air by the Brin process. It can be purchased compressed in strong steel cylinders; and it is therefore available at a moderate cost.

NITROGEN.—Chemical symbol, N. Atomic weight and density = 14. Specific gravity = 0·965 (if air = 1).

This elementary gas is present as the major constituent of air, which it nearly approaches in density. Nitrogen is unable to support combustion, and cannot maintain life when it is breathed. It is not, however, a poisonous gas, and if its proportion in air becomes unduly large, its presence is injurious mainly by excluding some of the oxygen.

Nitrogen is present as a constituent in many liquid and solid explosives, and the expansion which causes the explosion is in such cases due to the separation from the explosive of a considerable volume of nitrogen gas.

HYDROGEN.—Chemical symbol, H. Atomic weight and density = 1. Specific gravity = 0·069 (air = 1).

This elementary gas is the lightest gas known, hence if the densities of other gases are referred to that of hydrogen as unit, these densities are necessarily larger than 1.

Hydrogen occurs in nature only as a constituent of certain compound substances, of which water is the most common and abundant.

When this gas is kindled, it burns vigorously in the air with a very pale reddish-purple flame. This flame is intensely hot, and is very difficult to extinguish by being blown upon. The flame maintains itself in air containing large proportions of extinctive gases, owing partly to the small proportion of oxygen necessary for its combustion, and partly to the high temperature of the flame. The sole product of the combustion of hydrogen is steam.

The high temperature of the hydrogen flame burning in air is further augmented if the flame burns in air enriched in oxygen, and the temperature becomes extremely high when the flame burns in pure oxygen. The oxy-hydrogen blowpipe is an apparatus arranged to furnish this flame of hydrogen fed with oxygen; the flame of this blowpipe was originally used chiefly for heating a small cylinder of quicklime to produce the lime-light, but is now employed in many important metallurgical processes. Hydrogen explodes violently if it is mixed with half its measure of oxygen and the mixture is fired.

Hydrogen can now be cheaply bought compressed in strong steel cylinders, and is therefore available for useful applications, one of which will be described below.

METHANE, OR MARSH GAS.—Chemical symbol, CH_4. Density = 8. Specific gravity = 0·55 (air = 1).

This compound gas was formerly known as "light carburetted hydrogen," a name intended to denote its lightness and the fact that it is a compound of carbon and hydrogen. Its composition is now denoted by terming it a *hydrocarbon*, a name which is applied to all substances which contain hydrogen and carbon as their sole constituents. The name "marsh gas" was applied to the gas when it was obtained by stirring the mud of stagnant pools and marshes.

Methane is the inflammable constituent of *fire-damp*. Some samples of fire-damp consist almost entirely of this gas, whilst others contain it in mixture with carbonic acid and with nitrogen.

The name "gas" is frequently applied to methane in the coal-mine: it is sometimes wrongly termed hydrogen by the miner, the name being probably a curtailment of the old chemical name for this gas.

Methane is readily kindled; it burns with a somewhat luminous flame in the air, yielding carbonic acid and steam as the products of its combustion. When the gas is kindled in the air of the mine the flame burns and scorches seriously. If the supply of air to the flame is restricted highly poisonous carbonic oxide is formed.

Fire-damp escapes from the coal of many seams, in the substance of which it has probably been pent up since the formation of the coal. It naturally issues in largest quantity from the "working face" of the coal, and its escape diminishes in quantity as the coal is exposed to the air.

The lightness and buoyancy of fire-damp cause it to rise rapidly to the roof in the workings of the mine, and it will take the highest place possible. If it is kindled during its escape, or immediately after it has escaped, it will burn with a flare. If, however, it has had time to *diffuse* and become mingled with the air, its behaviour in contact with a flame will vary with the proportion of air with which it has mingled.

Mixture with small proportions of air only tends to hasten the burning of the gas. When the mixture is diluted until only 16 per cent. of methane is present, it begins to be explosive. But as the proportion of methane in the air approaches 10 per cent. the mixture begins to burn with greater rapidity, and when fired in large quantity becomes explosive. When the proportion reaches 10 per cent. the explosion is the most violent possible. If the air increases gradually above this proportion, the mixture becomes less and less violently explosive, and at last ceases to kindle at all in contact with flame.

When the proportion of methane in the air falls below 6 per cent. the gas cannot be fired, but it still shows its presence by burning as a pale halo or "cap" above a flame. The paleness of this cap is highly detrimental to its visibility over a bright flame, but it is distinctly seen over a bright flame of sufficiently small dimensions: the cap is visible over a reduced safety-lamp flame until the proportion of methane in the air falls to 3 per cent. or under the most favourable conditions to 2 per cent. Smaller proportions still give visible caps over the pale flames of hydrogen or alcohol. The hydrogen flame gives the most distinct and visible caps, owing to the paleness of its flame and its high temperature. A small hydrogen flame, such as can be easily burnt in an ordinary safety-lamp, shows a distinct cap, when only 0·1 per cent. of methane is present in the air.

The most satisfactory means of detecting and measuring small proportions of methane in the air is by the observation and measurement of this "cap" over a suitable flame (see Chapter XV.).

The system of ventilation in a coal-mine is designed to dilute the fire-damp to such an extent as to make it uninflammable and harmless when breathed.

It should be understood that this gas is not poisonous when breathed, but, that by its presence in the air, it may reduce the proportion of oxygen below the amount necessary for maintaining life.

The temperature at which mixtures of marsh gas and air ignite is between 650° and 700° Centigrade.

Some seams of coal give off much more fire-damp than others, and many give off none at all, consequently, at some collieries it is never found, at others only in small quantities, whilst in many such large quantities are yielded as to cause considerable anxiety to all engaged in the mine.

The origin of this gas in coal is the change which has produced coal from vegetable matter. It is probable that during the process of decomposition, new strata accumulated to such an extent as to cause considerable pressure. Where these newly-deposited strata were of a porous nature, the gases given off during decomposition would escape through the strata to the surface, but where the cover over the buried vegetable matter was impermeable, the gases were retained at increased pressure as decomposition proceeded and the coal-seam fully matured. Most probably the gas is contained in coal-seams as water is in a porous rock, and remains there in a more or less compressed condition so long as the enclosing walls are gas-tight and able to withstand the pressures from within. Probably the coal is still in a state of chemical change or decomposition, which may be proceeding very slowly, and thus gases are continually evolved and accumulated within the pores of some coals. If the roof over the coal be of an impermeable rock, and the thill be composed of porous sandstone, the latter will be found to contain gas, or if the cover immediately over the coal be porous, it becomes a gaseous zone. If the shafts are shallow, and there are no impermeable rocks between the surface and the coal-seam to retain the gas, it will be drained off to the surface. A seam of coal may in some parts of its course have but little cover over it and be quite free of gas at those points, whilst at others, owing to thicker stratification above, or the setting in of an impermeable rock which retains the gas, it may contain large volumes under great pressure. Again, two coal-seams may be separated by impermeable strata, but on working the upper seam the intervening strata, robbed of so much support, become unable to resist the pressure of gas in the lower seam, and, consequently, the floor of the seam being worked is raised and fractured. The fissures formed, enable large quantities of gas to pass from the seam below into the workings. The amount of gas contained in any seam does not depend on its depth from the surface, although it seems to be present more abundantly in mines of moderate depth. Seams of lignite coal are usually free from it, and it is only in true coal-seams that large quantities are found.

The gas, for the most part, disengages itself from the pores of freshly-wrought coal, or from the joints of recently-exposed coal surfaces; in most seams the issue occurs in the first few hours, and ceases after a few days. The largest volume is usually found at the working faces, where it issues in innumerable jets, as the workmen constantly extract coal and so lay bare fresh coal surfaces. In damp or wet seams, the discharge of gas from the coal is accompanied by a hissing sound, and in these the discharge, instead of ceasing in a few days, continues for months or even years.

In mines where the largest volume of fire-damp is not found at the working faces, but at a point behind them, it will generally be found that the gas has drained to the point of discovery, through the strata, from a lower seam of coal in course of working, whose line of advancing face lies outside that in which the gas is, or it may come from a lower unworked seam, or from an exceptional outburst.

Exceptional outbursts are either *blowers* or *outbursts*. Blowers are continuous discharges of fire-damp for long periods of time, without apparent decrease, through well-defined cracks or fissures in the roof or the floor of roadways. Sometimes the discharge of gas is accompanied by large quantities of water. When the gas can be detected at the point of issue by a safety-lamp it is sufficient to constitute a blower. The fracture through which the gas is discharged—whether a simple fissure or a fault—pre-existed, and on the driving reaching it the gas is tapped. The open fissures in the coal seam communicate with some reservoir filled with large quantities of gas in a high state of tension. Where the fractures do not allow sufficient gas to escape, or where the pressure of the gas is so great as to cause it to find vent before the fissures are reached by the drivings, or where reservoirs of gas exist without fractures, falls of the roof or upheavals of the floor occur, followed by the release of enormous quantities of gas into the roadways. As showing the power of these blowers, it may be stated that at Outwood, in West Lancashire, at a depth of 328 yards, a blower of great magnitude was encountered in a stone drift, causing operations to be suspended. The gas was dammed off, and taken to the surface by means of pipes 2 inches in diameter, where it burned with a flame 3 feet 6 inches high for a year.

At Merthyr Vale, in South Wales, so large a quantity of gas was met with on crossing a fault into a new area of coal, about a mile from the shaft, that it was deemed prudent to convey the gas separately to the surface, where it was utilised for several months in raising steam by being burned at the boilers. The pipes, which were 6 inches in diameter, were laid from the inside of a strong dam near the fault, through the return airway to the upcast, and thence to the surface.

Outbursts of inflammable gas are distinguished from blowers by the suddenness of their appearance, and their comparatively quick exhaustion. An outburst may occur at the working face, or from the roof or the floor. Those from the floor present the most marked features, and it is in the Barnsley and Silkstone seams of Yorkshire that they occur with the greatest intensity and frequency.

These outbursts have been commented upon in the chapter devoted to "Methods of Working," where the working of the Barnsley seam is described. As the workings approach the reservoirs of gas under the seam the floor is upheaved, and fractured in a line approximately parallel to the advancing face. This is the result of the pressure from the imprisoned gas, which finds its way first into the working faces through the fracture, and thence into other parts of the mine. In 1876 an outburst occurred at the New Oaks pit in the rise long-wall workings of the Barnsley seam, ventilated by a current of 10,000 cubic feet per minute. The returns became so loaded, that Mueseler lamps at the bottom of the upcast were extinguished in a current of 140,000 cubic feet per minute.

Outbursts from the roof result from large quantities of gas following the falling of the roof, or from the escape of gas from fissures in the roof without a fall.

At an outburst at Pelton, in the county of Durham, 47,000 cubic feet of gas issued from the coal at a pressure of 912 pounds to the square inch.

A very remarkable outburst of firedamp occurred at L'Agrappe, near Frameries, in the district of Mons, in Belgium, causing the death of 121 persons in April, 1879. The current of firedamp penetrated to the downcast shaft, up which it went and was ignited in the engine-house. A column of flame, the full size of the pit, at once shot upwards over it for a height of over 120 feet. This continued burning for three hours, and it was estimated, from the velocity of its upward movement, that this colossal outburst must have given off about 80,000 cubic feet of gas per minute.

Instances are recorded where the pressure of the gas has been sufficient to force out the coal in large masses of many tons' weight at the working faces, after which it escaped suddenly through the opening thus formed in such volumes as to constitute an outburst. The enormous volume given off has been powerful enough to foul a considerable area of workings and so prevent access to them for many days, notwithstanding the fact that a large current of air has been sent continuously into the district. In an occurrence of this sort, after a time the pressure of the gas decreases and the fresh air-current sweeps the roadways to a point nearer and nearer the face, and as the pressure subsides, to the face itself.

Occasionally rows of props which were originally fixed securely for the protection of a roadway, have been observed to sink at the foot until they become loose and eventually fall out. This is accounted for by the tapping of gas into the working seam from a stratum below, which previously served to support the floor in which the props were set. The gas pressure being removed, the coal and underlying strata have sunk, and the props collapsed.

The shots fired in sinking pits have been often known to liberate fire-damp from fiery seams on the sinking approaching the coal. In cases where the gas has caught fire, the flame has reached up the shaft to a considerable height, requiring much trouble to extinguish it.

In 1879 experiments were made by Mr. Lindsay Wood, for the Royal Commission on Accidents in Mines, in the various seams at Elemore, Hetton, Eppleton, Boldon, and Harton Collieries, with a view to ascertain the pressure of fire-damp in the solid coal, and also the quantity yielded in a given time from a certain face area where the coal was newly wrought.

Many experiments were made by boring holes of different lengths, either horizontally or slanting slightly upwards, but always at freshly-worked faces, the holes being rapidly bored, a pipe smaller than the hole drilled being subsequently inserted, of sufficient length to reach nearly to the end of the bore-hole. The longest bore-hole was 47 feet, and the shortest 3½ feet. The space between the pipe and the bore-hole was equally stemmed from a point near the inside end of the pipe, which there carried a protecting flange the full size of the bore-hole back to the working face. The flange prevented the stemming from being pushed too far, and ensured its equal distribution round the pipe. At its outer end the pipe was provided with a tap and pressure-gauge, whilst its inner end was open to the gas space reaching from its extremity to the end of the bore-hole. The highest recorded pressure was obtained at the Boldon Colliery, where about the third day after attachment the pressure-gauge indicated 461 lbs. per square inch. The greatest observed amount of gas given off from the holes was 5·927 cubic feet per hour per square foot of surface, and this occurred in an experiment at the Eppleton Colliery. Pressures varying from 200 lbs. to 461 lbs. per square inch were repeatedly observed; the depths of the places also varying from 750 feet to 1,268 feet below the surface.

Besides the exceptional outbursts just described, discharges of fire-damp from old into existing workings may occur. Old workings or goaves frequently receive discharges of gas, and become, as it were, magazines in which the dangerous element is stored. The gas in the old workings may come from the normal discharge of the seam being worked, or it may come from adjacent seams.

A large number of shots fired in the coal simultaneously at the working face may result in liberating an undesirable quantity, and therefore only a small number are fired at once in fiery mines, and that only by competent persons.

Under the ordinary conditions of a well-regulated mine, no inconvenience, perhaps, arises from the presence of this gas; but it becomes an acknowledged source of danger when, as frequently happens, it finds its way, owing to the sudden decrease of atmospheric pressure, or falls of the roof on a large scale, into the working roadways closely adjoining the edges of the goaf or old workings.

Attempts have been made to capture fire-damp at the moment of its disengagement from the freshly-wrought coal. The method consists in laying a series of pipes into the face of all the working places, connected with a main pipe leading to a powerful exhausting machine upon the surface. Similar attempts have been made to withdraw the fire-damp from goaves, but the method is not capable of very practical application.

Earthquakes, accompanied by earth motions, sometimes produce abundant escapes of gas at the surface, and it is possible that the position of reservoirs of gas may be also affected by fissures accompanying such motions. An earthquake may change the position of a gas reservoir in the earth's crust, or form others in fissures left by it which did not previously exist. Investigation is now being made as to whether abnormal issues of fire-damp are caused by seismic movements.

In approaching faults in fiery seams and after crossing them, the gas is frequently discharged freely. It sometimes rushes from the neighbourhood of faults, and rises through a column of water many yards high.

Mixtures of fire-damp and air may become altogether inexplosive if sufficient carbonic acid gas, free nitrogen, or other gas is mingled with them. Even small quantities of these gases, when present in the mixture, lessen the explosive force on ignition. *After-damp*, resulting from an explosion of fire-damp and air, varies in composition. It usually consists of 71·2 per cent. of free nitrogen, 9·6 per cent. of carbonic acid gas, and 19·2 per cent. of steam, or in round numbers out of

10 parts, after-damp contains 7 of nitrogen, 1 of carbonic acid gas, and 2 of steam. Directly after the explosion the steam condenses, and there is then left out of 8½ parts, about 7½ of nitrogen and 1 of carbonic acid gas. Breathing after-damp soon causes death, and many who escape the force of a fire-damp explosion in mines fall victims to the deadly after-damp.

If a mixture is exploded which contains more than 1 of fire-damp in 8 or 9 of air, its force is less than that of the most destructive proportions of fire-damp and air, as a certain amount of carbonic oxide is formed, which, if it could have burnt, would have increased the temperature and force of the explosion. Its presence after the explosion renders the after-damp more deadly. Again, if an explosion occurs in which the fire-damp forms less than 1 in 8 or 9 of air, its force must be less than that of the most destructive proportions of fire-damp and air, but in this case the after-damp will be rather less deadly than that resulting from the most violently-explosive mixture, as a part of the oxygen in the air remains unchanged. Whatever the proportion of fire-damp to air, the after-damp left from its explosion is unfit to breathe.

When ignited, the flame temperature of fire-damp and air is extremely high, and where there is a large volume of the explosive mixture present, the temperature of the portion being consumed increases the volume of the rest. From the centre or seat of explosion a great pressure is caused by the flames and heated gases, which proceed in every direction, driving the air away with great force; this pressure is exhausted at a distance near to or far from the seat of the explosion, according to its violence. As it progresses the condensation of steam in the after-damp and the cooling of the gases reduces its volume and pressure until it descends sufficiently to cause a backward movement, and a partial retreat of the ignited mass towards the seat of the explosion frequently ensues.

The ravages committed by these destructive blasts are familiar enough to explorers after explosions. As the conflagrations rush along the roadways, they for the most part take a course opposite to that of the intake air, with occasional "kick-backs," or slight splits at the junctions of roads, and as the flames are fed by the fresh air, the blast, if supplied by sufficient fire-damp, or fire-damp and coal-dust, it may be reaches the bottom of the downcast shaft, and exhausts itself in the shaft, disarranging or perhaps blowing out nearly all the shaft fittings with so much violence and noise as to cause the utmost terror to those employed on the pit top. In its destructive progress everything presenting an impediment, unless strong enough to resist the blast, is hurled to one side or overthrown; doors, air-crossings, trams, horses, men, the timbers for securing the roadways, &c., usually offer no obstacle to the fury of the explosion. The road timbers being knocked down the roof falls in, and the ventilation is arrested. In other parts of the roadways the timbers are considerably charred and deflected from an upright position, their altered state and appearance pointing almost as certainly as a finger-post in the direction of the blast. Too often the evidence of such mute objects is all that is to be obtained, for those who escape the violence of the explosion are poisoned by the after-damp, and not one is spared to throw light on the calamity.

Wet roadways naturally prevent accumulations of dust and the progress of an explosion, for it is in dry and dusty mines that the worst occur. The violence of the blast in these is arrested through any portions of the roads which are wet, however the dampness is caused.

CARBONIC ACID, OR CARBON DIOXIDE.—Chemical symbol, CO_2. Density = 22. Specific gravity = 1·5 (air = 1).

This compound gas is the product of the combustion of the elementary substance carbon, and it is formed when any substance containing carbon is burnt with a sufficient supply of air. The gas is also formed during respiration. It escapes in some quantity from certain kinds of coal, and is also present in the soil.

The gas cannot be inhaled if it is unmixed with air, or if it only contains a small amount of air; hence it is not inappropriately termed "choke damp," since the attempt to inhale produces a sensation of choking. Carbonic acid is an important product of an explosion of fire-damp, and is present invariably in large proportion in the "after-damp" produced when "gas" is fired or exploded in the coal-mine.

Carbonic acid gas issuing from the coal or soil tends by its weight to fall in the air and to collect on the "floor" of the workings, or in the "sump" or well, but the process of diffusion causes it to slowly mingle with the air. When carbonic acid issues from the lungs, or is produced by a flame, however, it is rendered light by its high temperature, and tends to rise in the air.

Atmospheric air, as has been already stated, contains usually about 0·04 per cent. of carbon dioxide. This proportion has no effect upon the respirability of the air; but if the proportion of the gas increases it will at last reach an amount which renders the air unfit to support life. The extreme limit has been stated to be about 15 per cent., but even less proportions than this in the air cause drowsiness when the air is breathed, and should not be inhaled for any length of time.

Pure carbon dioxide gas at once extinguishes flame, and the gas retains this extinctive power even when it is mixed with a considerable proportion of air. Air containing 15 per cent. of carbonic acid extinguishes at once the flame of candle, oil, and alcohol; coal-gas flames require about 33 per cent. of carbonic acid for their extinction, and a hydrogen flame is not extinguished until the proportion reaches 58 per cent. Hence the hydrogen flame is able to maintain itself in air containing four times the quantity of carbonic acid which proves extinctive to a candle, oil, or spirit flame, and nearly twice the amount which extinguishes a coal-gas flame.

CARBONIC OXIDE, OR CARBON MONOXIDE.—Chemical symbol, CO. Density, 14. Specific gravity = ·965 (air = 1).

This gas is composed of the same constituents as carbon dioxide, but these constituents are present in different proportions. It does not occur in coal, and is only produced by artificial processes. It is formed when any substance containing carbon is incompletely burnt; also when carbon dioxide gas is acted upon by red-hot carbon.

It is therefore generally considered that one of the conditions essential to the formation of this gas in the coal-mine, is the burning or explosion of fire-damp with a quantity of oxygen insufficient to burn it completely into carbon dioxide. Another cause of the formation of the gas is the presence of coal-dust, which becomes heated by the fire-damp flame and changes carbon dioxide into carbonic oxide. The smothered combustion of coal in the mine or of a gob-fire also furnishes the gas.

This gas is also produced in the coal mine by the firing of blasting-powder, and of nitro-cotton. It is an important constituent of "water-gas."

Carbonic oxide is combustible in air; it burns with a blue flame, and becomes explosive when it is mixed with air and fired. But the chief danger arises from its powerful, poisonous action when it is inhaled, even in small proportions, probably less than 1 per cent.; it is rapidly fatal when breathed. Those who have been poisoned by the gas retain the appearance of life and health, and the colour of the skin is not changed.

The gas has a faint sickly smell, and when inhaled produces a tight feeling in the brain and giddiness. It shows a distinct "cap" over the hydrogen flame when less than 0·25 per cent. is present in the air, and may therefore be detected in proportions which are not at once fatal. The "cap" test, however, can only be applied to the detection of the gas when other inflammable gases are known to be absent, since the appearance of the "cap" is not easily distinguishable from that given by fire-damp. The fatal proportion, when it is present in air, is not sufficient to affect the burning of a flame; and no warning is afforded of its presence therefore by the extinction of the lamp flame.

Sulphuretted Hydrogen, or Hydrogen Sulphide.—Chemical symbol, H_2S. Density = 17. Specific gravity = 1·172 (air = 1).

This is also an extremely poisonous gas. It is seldom met with in any quantity in the coal-mine, but occasionally escapes from water, and is also produced in small quantity in the firing of blasting-powder and gunpowder. Its strong and highly offensive smell renders its presence easily detectable. It is combustible, and burns with a blue flame.

EXPLOSIONS IN THE COAL-MINE.

It seems to be now definitely established that there are at least two distinct sources of danger from explosion in the coal-mine. The one is due to fire-damp and the other to fine coal-dust suspended in the air. But there is little doubt that to these must be added the carbonic oxide gas which will be introduced into the air by a blown-out shot when blasting-powder is used; this gas, though only sufficient in quantity in itself to produce a small local explosion, is competent to originate a more extensive explosion from coal-dust.

In order that an explosion of any magnitude from the above causes shall be produced, it is necessary that the combustible substance shall be intimately mingled in the necessary proportion with air. The mixture must further be present in some considerable quantity, and it requires to be heated in a suitable manner in order to be kindled.

If fire-damp is the combustible gas which produces the explosive mixture, it must be present in the air in proportions between 5 and 13 per cent., and such a mixture is easily kindled by a flame or other white-hot surface, provided the gas is in contact with the source of heat for a few seconds. Any such mixture becomes more readily explosive if its density is increased by pressure, or if its temperature is raised. It is also more readily fired from below than from above or laterally.

Some flames from blasting explosives, though possessing far more than the necessary temperature, cannot inflame the explosive fire-damp mixture, owing to the extremely brief period of existence of the flame.

If coal-dust is the combustible substance in the air, the dust must be extremely fine and dry, and must be well mixed with the air. The air containing the dust must then be exposed to a flame of large dimensions in order to be fired. Thus, while a candle-flame is unable to fire the mixture, the large flame from a blown-out shot readily does so.

The kindling of the fine coal-dust in air is very much facilitated by the presence of even a minute proportion of inflammable gas. As little as 1 per cent. of fire-damp makes air charged with fine coal-dust dangerous. It is extremely probable that the carbonic oxide gas from blasting-powder may also suffice to add to the danger arising from inflammable gas when fine coal-dust is present.

The methods of preventing danger of explosion from fire-damp by means of efficient ventilation are described later on in this chapter; the methods of testing for the gas in the air in order to ascertain that the ventilation is adequate, are described in Chapter XV. of this work. Of the various methods of gas-testing, that by means of flame-cap is undoubtedly the most delicate and trustworthy, and the most distinct and delicate results are yielded by the hydrogen-flame.

Danger from coal-dust is removed by watering or removing the dust; and the risk from the employment of blasting-powder may be reduced by employing other explosives, which give no carbonic oxide or inflammable gas and which do not produce so lengthy and durable a flame.

EXPLOSIVES.

The explosive substances used for blasting and bringing down the coal are solid or semi-solid substances, which owe their power to being able to generate instantaneously a considerable volume of gas, and to produce at the same time a high temperature. The production of the gas alone would lead to great increase in volume of the explosive, or to production of great pressure if the explosion occurred in a closed space; but the high temperature imparted to this gas at the moment of its production further adds very considerably to its volume and to the pressure which it exerts if confined. The gases usually evolved by explosives are carbon dioxide, nitrogen, and steam.

Explosives are caused to explode by two distinct methods:

(1) *By firing with heat*, so as to cause the properly prepared mixture to catch fire and burn rapidly; of such explosives, gunpowder and blasting-powder are examples. These are usually fired by a combustible fuse.

(2) *By shock or detonation*, the explosive being caused to undergo its explosive change by being subjected to the shock of a suitable substance being exploded or detonated in contact with it: dynamite and blasting-gelatine are instances of this class. They are exploded by so-called detonators, which are themselves caused to explode by electricity.

The applicability of an explosive for use in the coal-mine must be judged from several standards, of which the following are the most important:

1. Its safety from being exploded whilst undergoing transport and handling.
2. Its incompetence to fire "gas" or coal-dust in the air of the coal-mine.
3. Its non-production, when exploded, of any amount of gas which is poisonous or inflammable, such as carbonic oxide.
4. The non-emission of burning particles which could fire gas or coal-dust.

Some of the chief explosives which are in use or have been proposed for use in the coal mine are described in the following pages. They are classified according to their method of being exploded. Certain modern explosives are omitted or shortly treated of on account of their unfitness for mining purposes, because they do not comply with all the above conditions. It may be at once stated that the only explosives which satisfy the third condition are nitro-glycerine, which is used in the form of dynamite, and the so-called Sprengel explosives.

I. *First Class of Explosives.*

Mining powder, Blasting-powder and Bobbin powder.—These explosives have until recently been most largely used in the coal-mine. They are fired by a fuse which in itself is a source of danger. These explosives are practically gunpowder, modified in composition so as to lessen the cost and to increase the volume of gas produced when it is fired, while the temperature produced by the firing is lowered. The average percentage composition of these powders in England is as follows:

Potassium nitrate	65
Sulphur	20
Charcoal	15

Although these powders are adapted to the purpose of the coal-mine by the useful effect which they can produce, they are to be condemned on account of the large proportion of inflammable gas which they produce when exploded, which includes a large volume of poisonous carbon monoxide: they further discharge hot burning particles which can kindle inflammable gas.

The following statement of the composition of the gas produced by these explosive powders will prove the above assertion:—

COMPOSITION OF GAS GIVEN BY MINING POWDER WHEN FIRED.

Carbon monoxide	33·75	48·82
Hydrogen sulphide	7·10	
Hydrogen	5·24	
Methane	2·73	
*Carbon dioxide	32·15	51·18
*Nitrogen	19·03	

All these gases, with the exception of those marked with an asterisk, are inflammable gases and can burn with slow flame in the case of a blown-out shot. The projection of red-hot solid particles from such a shot is also common, and these can kindle either fire-damp in the air or the inflammable gas produced by the powder: this flame in its turn easily fires coal-dust. No less than one-third of the gas which is given off consists of highly poisonous carbon monoxide, whilst 7 per cent. is still more poisonous hydrogen sulphide. Almost one-half of the whole gas produced is inflammable, or can become explosive with air.

If an ordinary charge of about 1¼ lb. of powder is used in firing a shot, and the firing results in a blown-out shot, about 10 cubic feet of explosive atmosphere will result, and this, if fired by the flame or by the red-hot particles from the powder, will be amply sufficient to kindle air laden with fine coal-dust.

A sample of air collected immediately after the firing of such a shot was found to contain 8 per cent. of carbon monoxide, an amount which would certainly be sufficient to render air mixed with fine coal-dust easily explosive when kindled. This points to a serious source of danger arising from the use of mining-powder.

These slow-burning powders suit soft coal, but for stone and hard coals, the rapid explosives which follow are much more suitable.

II. *Second Class of Explosives.*

These explosives can be fired by a flame or fuse, but they require to be fired by a detonator in order to develop a rapid and powerful explosion.

These explosives fall into two classes:

1. Those prepared with nitro-glycerine, $C_3H_5(NO_3)_3$. These include:—

Dynamite	Nitro-glycerine	75
	Kieselguhr (Silica)	25
Blasting gelatine	Nitro-glycerine	92
	Nitro-cotton	8
Carbonite	Nitro-glycerine	25
	Potassium nitrate	30
	Barium nitrate	4
	Wood meal	40
	Sodium carbonate	1

Gelignite	Nitro-glycerine	56·5
	Nitro-cotton.	3·5
	Wood meal.	8·0
	Sodium nitrate	32·0

Since nitro-glycerine is a liquid substance, none of the above mixtures are hard solids. They present special danger owing to liquid nitro-glycerine being apt to exude from them. They are further sensitive to shock, more especially when they are frozen. In the frozen condition they therefore present danger to which the ordinary explosive is not subject. The necessary thawing of these substances is also attended with serious danger.

Nitro-glycerine is a powerful explosive, and presents the advantage over many other explosives, that it contains more than enough oxygen to completely oxidise its carbon, hence no carbon monoxide is produced when it is exploded. It is converted for use into a semi-solid form by mixing it with solid powders. The powder used in making dynamite does not add to the explosive force, but in each of the other mixtures the powder which is added is itself explosive.

2. Another kind of explosive falling under this second class is that composed of nitro-cotton powders, such as Tonite, which is a mixture of nitro-cotton and barium nitrate.

Nitro-cotton itself, when detonated, furnishes carbon monoxide, owing to the oxygen which it contains being too small in amount to convert its carbon into carbon dioxide.

III. *Third Class of Explosives.*

This class of explosives can only be exploded by a special detonator composed of fulminate of mercury or of nitro-glycerine. These explosives accordingly obviate risks from explosion by heat or by spark, or by any ordinary shock. They include the following:—

Roburite	Ammonium nitrate	86
	Chlorinated dinitrobenzol	14
Bellite	Ammonium nitrate	80
	Dinitrobenzol	20
Ammonite	Ammonium nitrate	88
	Nitro-naphthalene	12
Securite	Ammonium nitrate	80
	Dinitrobenzol	17
	Ammonium oxalate	3

These explosives produce a lower temperature when they are detonated than other explosives do; and, although the temperature is high enough to kindle fire-damp, the kindling does not occur on account of the short duration of the flame. It is found that fire-damp must be exposed to flame for several seconds in order that it may be kindled with certainty. This is not the case with carbon monoxide. These explosives produce no carbon monoxide or other inflammable or poisonous gas, and no smoke when they are detonated; and since they contain no substance which remains solid after detonation, there is no risk of red-hot solid particles being projected into fire-damp or fine coal-dust and causing fire or explosion. Further, since they are exploded by a detonator, there is no danger of a burning fuse being projected and causing damage for a similar reason.

The ammonium nitrate (NH_4NO_3), employed in the preparation of these explosives, presents several advantages over the potassium nitrate (KNO_3) and the potassium chlorate ($KClO_3$) used in the preparation of certain other mining powders. It is completely converted into nitrogen gas and steam during detonation, and at the same time furnishes some oxygen for the combustion of the other constituents of the explosive. Further, it absorbs a considerable amount of heat during its detonation and thus tends to lower the temperature when the explosive is in use. This absorption of heat makes it impossible to kindle these explosives and cause them to burn in a large mass, and renders necessary a very powerful detonation to explode them.

That these explosives require a high temperature for their kindling seems to be established by the fact that when a mixture of gunpowder and roburite is ignited the pieces of roburite are scattered without change.

A similar set of safe high explosives are in use in France in which ammonium nitrate is mixed with dynamite, with blasting gelatine, with nitro-cotton, and with trinitronaphthalene.

In connection with these comparative statements concerning explosives, the following tabulated information will be of interest.

PERCENTAGE OF INFLAMMABLE GAS PRODUCED BY VARIOUS EXPLOSIVES.

Explosive.		Carbon Monoxide.	Hydrogen and Methane.	Total percentage of Combustible Gas.
Powder	Gunpowder	10·5	3·1	13·6
	Blasting-powder	33·7	7·9	41·6
Sprengel explosives	Roburite	Nil.	Nil.	Nil.
	Ammonite	Nil.	Nil.	Nil.
Nitro-glycerine explosives	Nitro-glycerine	Nil.	Nil.	Nil.
	Gelignite	7	Nil.	7
	Carbonite	15	26	41
	Blasting gelatine	32·5	8·6	41·1
Nitro-cotton explosives	Tonite	8	Nil.	8

TEMPERATURE PRODUCED BY FIRING VARIOUS EXPLOSIVES.

	Degrees Centigrade.
Blasting-gelatine	3,090
Nitro-glycerine	3,200
Dynamite (20 p.c. silica)	2,940
Gun-cotton (11-nitro)	2,650
Tonite	2,648
Picric acid	2,560
Roburite	2,100
*Ammonium nitrate	1,130

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* Ammonium nitrate cannot be completely detonated by itself, but it is completely detonated in the explosive mixtures which contain it.

THE VENTILATION OF MINES.

Air is an *elastic* body. If it or any gas (as they have the same physical properties) be confined in a vessel, it exerts a pressure against the sides altogether apart from its weight, and the pressure is exerted on the upper part of the vessel as well as the lower. If the volume occupied by it be in any way diminished, that is, if the same quantity is made to occupy a smaller space, the pressure will be increased, or if the space occupied remain the same, and the temperature be raised, the pressure will also be increased.

Air is *compressible*. Boyle and Mariotte's law of compression is:—The temperature remaining the same, the volume of a given quantity of gas varies inversely as the pressure which it bears. The law showing the relation between the temperature and the volume of any gas was discovered by Charles, and may be stated as follows:—If any gas be allowed to expand freely under a constant pressure, its increase of volume when raised from 32° F. to 212° F. will be equal to 0·366 of its original volume, and this law of increase holds true in the same proportion for intermediate temperatures. From this it follows that as the difference between 212 and 32 is 180, then $\frac{1}{180}$th of ·366, or about $\frac{1}{492}$, (strictly $\frac{1}{491 \cdot 8}$ or $\cdot 0020\dot{3}$) is the expansion for each degree, and this fraction is taken as the co-efficient of expansion. In other words, a gas expands $\frac{1}{492}$ of its volume at 32°, or $\frac{1}{460}$ of its volume at 0° for each degree that it is raised above that point. Supposing such a question as the following be given:—A quantity of gas is measured at a temperature of 70° and is found to occupy 400 cubic inches, what is its volume at 60°? To find the proportion between the space a gas occupies at 60° and 70° of Fahrenheit's thermometer, 492 cubic inches at 32° occupy 520 at 60° and 530 at 70°. The volumes therefore for any other quantity must be in this proportion, and therefore 400 cubic inches at 70° will occupy a volume at 60° in proportion as 530 is to 520. As 530 : 520 : : 400 : 392·4 cubic inches; or say that $1 + \frac{70 - 60}{460 + 60}$ of the volume at 60° = 400 at 70°, and therefore $400 \div 1\frac{10}{520} = 392 \cdot 4$. But this is assuming the pressure to be the same in each case; if not, a correction must be made for it. For instance, if in the above example the barometer stood at 30 inches when the 400 cubic inches of gas were at a temperature of 70° but at 29 inches when at 60°, the correction for the difference in pressure would be made thus:—As 29 : 30 : : 392·4 : 406; or it may be expressed thus, $400 \times \frac{30 \times (60 + 460)}{29 \times (70 + 460)} = 406$. Or, expressing these rules as formulæ, if v be the volume of any given weight of elastic fluid under any pressure and at 32° F., the volume v_1 which it will occupy under the same pressure and at any other temperature t of F. will be $v_1 = v + v \times \cdot 0020\dot{3}\,(t - 32)$.

This will be true if the ratio of the relative volumes be put u and u_1 instead of the ratio of the absolute volumes v and v_1, thus: $\frac{u}{u_1} = \frac{1 + \cdot 0020\dot{3}\,(t - 32)}{1 + \cdot 0020\dot{3}\,(t_1 - 32)}$.

Applying this to the question given,

$$\frac{u}{400} = \frac{1 + \cdot 0020\dot{3}\,(60 - 32)}{1 + \cdot 0020\dot{3}\,(70 - 32)} \therefore \frac{u}{400} = \cdot 9814,$$

and $u = \cdot 9814 \times 400 = 392 \cdot 5$.

The formula to find the relative volumes when both temperature and pressure change at the same time is,

$u = 1 \times \frac{p_1}{p} \times \frac{1 + \cdot 0020\dot{3}\,(t - 32)}{1 + \cdot 0020\dot{3}\,(t_1 - 32)}$, and applying this to the question in its second form,

$$u = 400 \times \frac{30}{29} \times \frac{1 + \cdot 0020\dot{3}\,(60 - 32)}{1 + \cdot 0020\dot{3}\,(70 - 32)},$$

$$u = 413 \cdot 8 \times \cdot 9814 = 406.$$

When the atmosphere is charged with moisture the mercury falls, because the pressure on the exposed surface is reduced. At first, to the student who remembers that water is about 815 times heavier than air, it will appear strange that air saturated with moisture should weigh less than air in a dry state.

Dry air is heavier than air impregnated with vapours, because of the extreme tenuity or thinness of watery vapour, the density of which is less than that of atmospheric air, whereas the density of water is much more. The average amount of water held in suspension in 1,000 cubic feet of air is reckoned at about two-fifths of a pint. When there is as much watery vapour present as the air can contain at the existing temperature, the air is said to be *saturated*. A thousand cubic feet of air are capable of holding half-a-pint of water, and this may be regarded as the point of saturation, which, however, necessarily varies with the temperature of the air. If the temperature of air in this condition be raised a few degrees, the air is not then saturated, because the quantity of moisture necessary to produce a state of saturation is greater as the temperature of the air is increased; but if, on the other hand, the temperature slightly falls, clouds begin to appear, and a still further decrease of temperature condenses this watery vapour to water, and it falls as rain. The density of water is a very different thing from the density of vapour of water; the former, as stated before, is 815 times the weight of air, but watery vapour is only about a half the weight of dry air. Strictly, the proportion is as ·6235 : 1 for watery vapour and air at the same temperature and under the same pressure.

The temperature of the atmosphere is greatest at the surface of the earth, and decreases 1° Fahr. for every 340 feet in height. The temperature of the air on the surface of the earth also varies with the height above sea-level, and with local circumstances.

In Atkinson's *Practical Treatise on the Gases met with in Mines* will be found the following rule to ascertain the weight of a cubic foot of air at any temperature and under any pressure :— $\frac{1·3253 \times I}{459 + t}$ where I = the height in inches indicated by the barometer, and t = the temperature by Fahrenheit's thermometer. The reason of the figures 1·3253 being used is because 459 cubic feet, when the barometer reads 30 inches, weigh 39·76 lbs., but at 1 inch it is only $\frac{1}{30}$th of this, or $\frac{39·76}{30} = 1·3253$.

Wind is air in motion, and varies in direction and velocity. It has the same pressure as the still air, but may exert considerably more force in one direction, as a result of its velocity.

From what has been said of the gases in mines it will be readily seen how necessary it is to provide a current of fresh air to take the place of that which has been rendered impure. A supply must be produced and distributed through the workings. Without provision being made, a certain amount of natural ventilation takes place in mines.

The reason of this is, that the same law which causes a rise or fall of the atmospheric currents over the face of the earth operates also below its surface. The temperature of the rocks for the first 50 feet or thereabouts is influenced to a slight extent by the changes of temperature on the surface, but below that depth it is the same all the year round. Beginning with 50° F., at a depth from the surface of 50 feet, this temperature increases 1° F. for about every additional 60 feet of depth. The temperature of the rocks in the workings of a shaft 700 feet deep would be then about 61° F., a higher temperature than that of the air on the surface in winter, but a lower temperature than that of the air on the surface in summer, so that the air in workings of moderate depth will be warmer in winter and cooler in summer than the air at the surface. The colder column of air will press upon and displace the warmer. Suppose (Figs. 367 and 368) two shafts at some distance from each other, having a different surface level and connected at the bottoms of the shafts by a gallery.

In the winter, when the air on the surface is colder than that of the gallery, the column of air in D E is cooler and heavier than the column of equal height A B; consequently the cold air displaces the warm, and so produces a current in the direction shown by the arrows in Fig. 367. This result, as regards direction of current, would not be affected by the size of the shafts or of the gallery connecting them. In the summer, when the air on the surface is warmer than that of the gallery, the column A B is cooler and heavier than the column of equal height, D E, and the direction of the air current would be reversed as seen in Fig. 368. But at certain seasons between these two extremes the two columns would balance each other and the current of air cease to circulate. The more nearly the two shafts approached the same surface level the more feeble would be the current in summer and winter and the more liability there would be for it to cease at other times. In by far the largest number of mines natural ventilation is totally inadequate, and even under the most favourable circumstances it cannot be relied upon. Therefore artificial means are necessary to promote ventilation in a certain definite direction, so as to ensure sufficient currents of air circulating from the downcast through the workings to the upcast shaft.

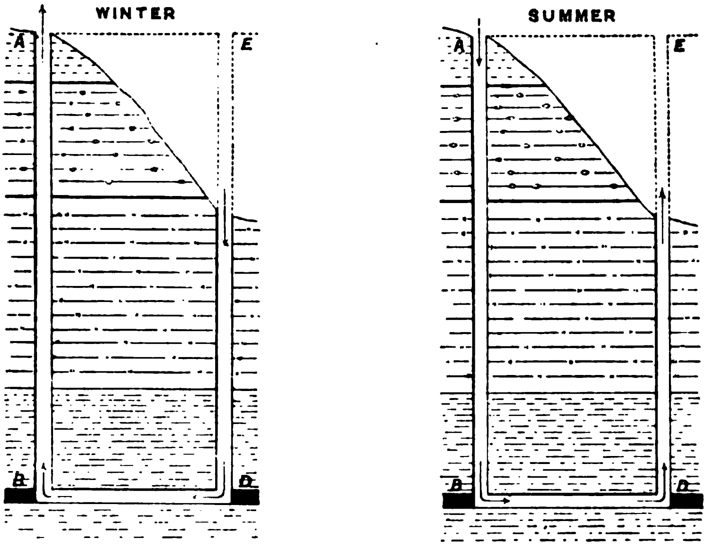


Fig. 367. Fig. 368.

NATURAL VENTILATION.

In the early history of mining the *furnace* was resorted to for ventilation, and is still used in many mines, the furnace being placed at the bottom of the upcast shaft. The fire heats the air and by expansion renders the column of air lighter, and in consequence of this the colder and heavier air passes down the downcast to displace it. The furnace may be placed in the return airway if the shaft is only used for ventilation and no explosive gas is given off in the mine. If the position chosen for it be in rock, a single archway may be built over the furnace with fire-brick, but if it has to be placed where coal or shale forms the sides or roof, it will be necessary to remove some of it on each side, so that side arches may be built to protect the sides and roof from fire, and provide a passage for the circulation of air.

If it is necessary to use the return airway for purposes other than ventilation, some other place must be selected for the furnace and a connection made with the return airway. The furnace is usually from 5 to 10 feet wide and the fire-bars 6 feet long, the space above being from 3 to 5 feet to the arch and below about 4 feet. A good plan is to let the breadth equal the diameter of the upcast shaft and to let the sectional area of the furnace drift be not less than the sectional area of the upcast shaft. The length of a furnace should not exceed 8 or 9 feet, and its breadth should not exceed 10 or 12 feet, because of the difficulty to feed and attend to the fire, and it is much better to have two furnaces than one large one. Figs. 369, 370, and 371 show in plan and section a double furnace with side arches.

All the work in contact with the flames of the furnace must be of fire-brick. The arches over the furnaces may be built with a thickness of fire-brick next the furnaces and have a blank space of a few inches interposed between the fire-brick and the ordinary brickwork beyond. The fire-bricks should be cemented together by ground fire-clay so as to effectually stand the heat from the furnace.

A well-constructed furnace will yield about 6,000 cubic feet per minute for each foot in breadth of fire-bars.

With a double furnace each grate of which is 8 feet wide, there would be therefore about 16 × 6,000 = 96,000 cubic feet per minute passing over it. But the work yielded by a furnace is affected considerably by the depth from the surface at which it is placed. In all cases the quantity of air will be as the square root of the difference between the temperatures of the downcast and upcast shafts and also as the square root of the depth from the surface. All other things being equal, the same furnace which is placed at a depth of 200 fathoms will produce double the quantity of air that it would yield at 50 fathoms. The horse-power of a furnace is calculated from the ascertained weight of the air in the upcast and downcast shafts, which will be more particularly referred to later on. Sometimes the fires of underground steam-boilers act as furnaces assisted by the heat of the exhaust steam from the engines. Where the return air is liable to be charged with explosive gas, it is often made to enter the upcast shaft by a dumb drift, the point at which such drift enters the shaft being not less than 8 fathoms above the end of the furnace drift, so as to ensure that the return air, if inflammable, shall not be ignited by the furnace. By the Mines Act, 1887, the return air, unless it be so diluted as not to be inflammable, must be carried off clear of the fire by means of a dumb drift or airway. The dumb drift should have an inclination of not less than 1 in 6, or the smoke is liable to flow back from the running of the cages or the opening of doors. If all the return airways to the shaft are charged with inflammable gas, none of them will be available to supply the furnace with air, and in that case it must be fed with fresh air from the downcast. The fire should be kept thin and coal thrown on frequently, so that the air may pass freely through the burning fuel.

The *waterfall* is an expedient for producing ventilation. It may be caused by allowing the pump cisterns to run over, or pipes may be laid for the purpose, the water being scattered and not falling in one stream. It is not a very efficient means of ventilation, nor a very economical one, as the water has to be pumped again, unless under the exceptional circumstances of the mine having an adit by which the water would run level free to the surface. However, it is a very ready way to obtain air under exceptional circumstances, such as after an explosion.

The *steam jet* is another of the artificial methods of ventilation. It consists of steam, which may be brought down the shaft, being allowed to issue in small jets

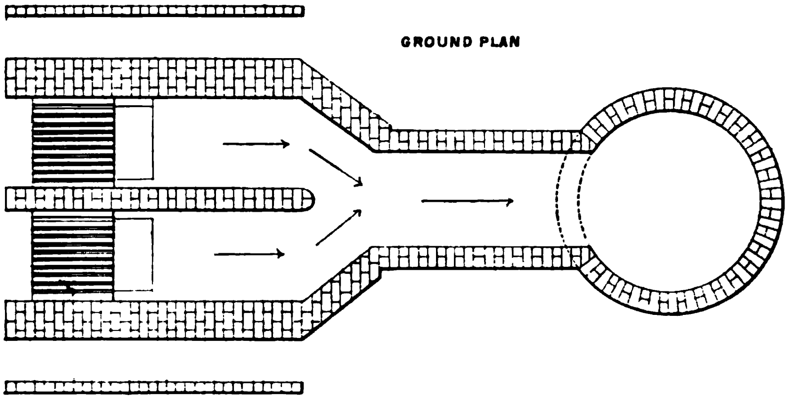


Fig. 369.

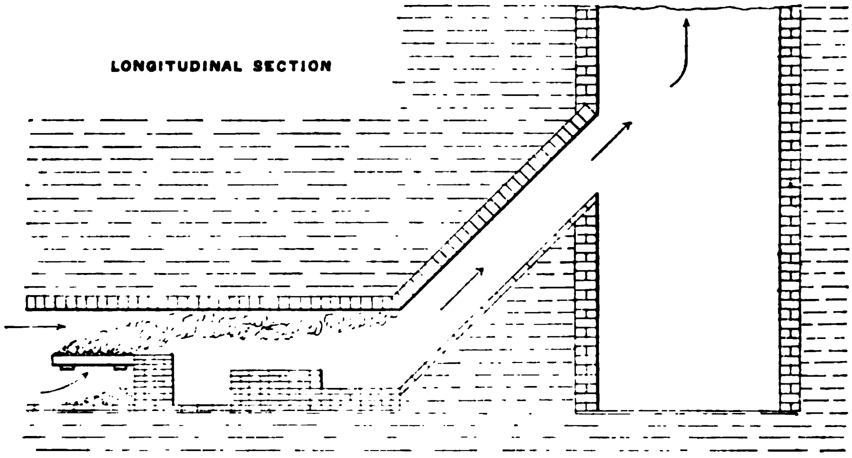


Fig. 370.

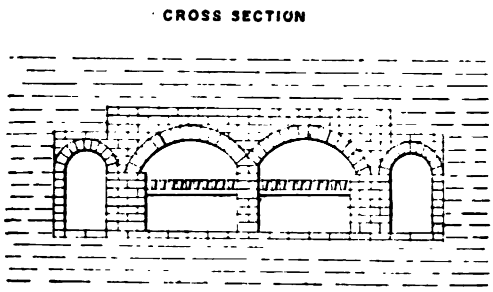


Fig. 371.

Double Ventilating Furnace.

from $\frac{1}{16}$th to $\frac{1}{8}$th inch in diameter directed upwards and placed in concentric circles round the bottom of the upcast shaft. It is not nearly so efficient as the furnace, and except in cases of emergency is not much resorted to.

MECHANICAL VENTILATION, OR THAT CAUSED BY THE USE OF MACHINERY.—This machinery may be divided into two classes, viz., the varying capacity or displacement machines, and the centrifugal.

Dealing first with the *Displacement* machines, the oldest form of mechanical ventilator is the air-pump, worked by a steam-engine.

The *Struvé* is one of this kind, shown in Fig. 372. It consists of two close-topped airometers or pistons made of sheet-iron in the form of a gasometer, worked alternately, by means of a beam, up and down in a ring of water formed between the brickwork. Only one of the airometers is shown in Fig. 372, the other, similar to it, being placed at the other end of the beam. At the top and bottom of the walls of the chambers (of which there are two to each piston) are placed flap-valves, hung upon vertical gratings in the wood framing, and these valves are so arranged that in making the up-stroke as in making the down-stroke

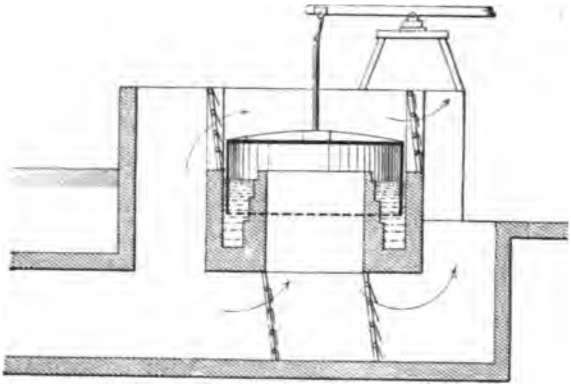


Fig. 372.—THE STRUVÉ VENTILATOR.

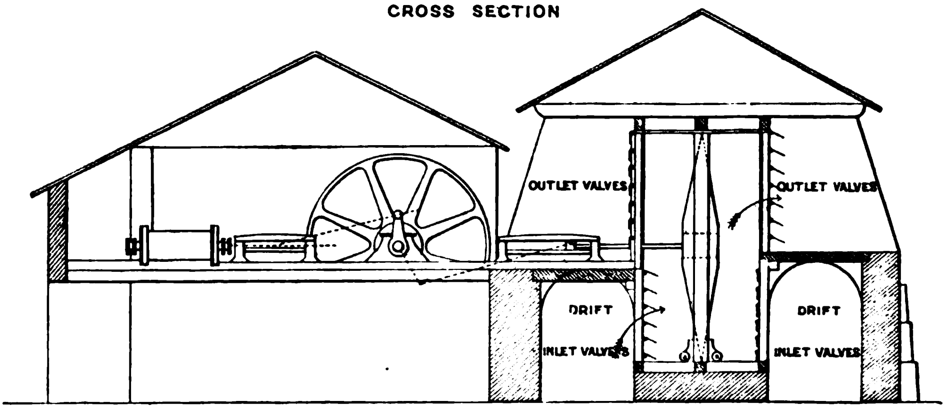


Fig. 373.—NIXON'S VENTILATOR.

the air is both being drawn in and forced out. They are of course connected with a passage leading from the top of the upcast shaft, and the steam-engine placed on the surface usually works the two pistons by means of a beam, so that there is an even flow of air up the shaft. The useful effect varies according to whether the flap-valves are new and in good order, or in an ordinary working condition. In the Report of the North of England Mining Institute Committee on Mechanical Ventilators, issued in 1880, they give the useful effect of the Struvé working at Cwm Avon Colliery, South Wales, at 57·8, and it stands highest on their list. It is not at present in use.

Nixon's ventilator, Fig. 373, is another of the air-pump kind. It is a horizontal double-acting air-pump, and the two rectangular pistons, which are of large size, are supported on wheels which run on rails. The lower half of each chamber is in connection with the pit when the inlet-valves are open; on the upper half of the chamber the outlet-valves are hung, and these communicate with the atmosphere. The valves are much the same in construction as those of the Struvé. With this kind of ventilator, if it were possible to close the downcast shaft and absolutely exclude the air from entering there, the ventilator, if perfect, would create a vacuum in the mine. The useful effect of the Nixon's ventilator, like that of the Struvé, must to a great extent depend upon the condition of its valves. In the Report already referred to, the useful effect of the Nixon ventilator working at the Navigation Colliery, South Wales, is given at 45·91.

The *Fabry* ventilator, Fig. 374, consists of two wheels of equal diameter on different shafts. Each axis is fitted with three broad blades from 6 to 10 feet wide, and each blade is formed with a cross arm, and, as the axes revolve in opposite directions, two of these are always in contact during revolution in the style of toothed wheels. The machine may be used for exhaustion or compression according to the way the wheels revolve. The bottom half of them fits into a brick casing, and the air from the shaft, if being exhausted, is taken up between the blades and afterwards discharged above the casing. The shafts which bear the two wheels are connected together by two ordinary wheels having the same circumference.

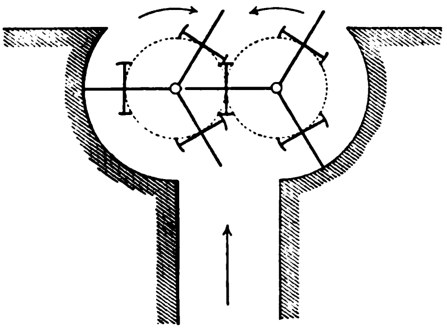


Fig. 374.—THE FABRY VENTILATOR.

In the *Lemielle* ventilator, Figs. 375 and 376, a six-sided drum is placed eccentrically in the cylindrical brickwork, and revolves upon a vertical axis. Three vanes are attached by means of hinges placed at alternate corners of the six sides of the drum. The other ends of these vanes are joined to two connected rods which turn round a fixed shaft in the centre of the cylindrical brickwork, and this causes the vanes to set themselves to the constant change of position of the drum. By the revolution of the drum,

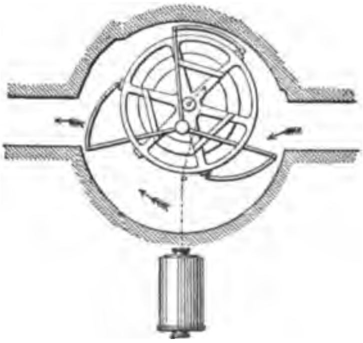


Fig. 375.—THE LEMIELLE VENTILATOR.

spaces varying in capacity are formed, the largest occurring at the inlet from the mine and the smallest when passing the exhaust drift. The air is expelled by the vanes (which act in turns as pistons) as they approach the point of outlet. According to the North of England Mining Institute Committee's Report, the useful effect of the Lemielle ventilator at Page Bank Colliery, Durham, is given as 23·4, which is very low as compared with some of the other ventilators.

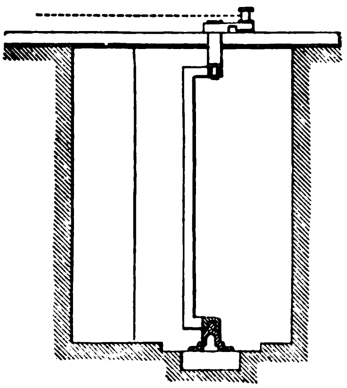


Fig. 376.—The Lemielle Ventilator.

Cooke's ventilator, Figs. 377 and 378, consists of two cylindrical drums, which revolve eccentrically in two cylindrical casings placed side by side. Each drum has a swinging shutter, and this shutter receives its motion from the crank and is so arranged as to be always close to the drum ; it thus forms a partition between the outlet and inlet air while the air is being drawn in from the drift leading from the shaft. The drums are placed opposite each other on the driving-shaft, so that the flow of air is uniform, and that the drums in revolving

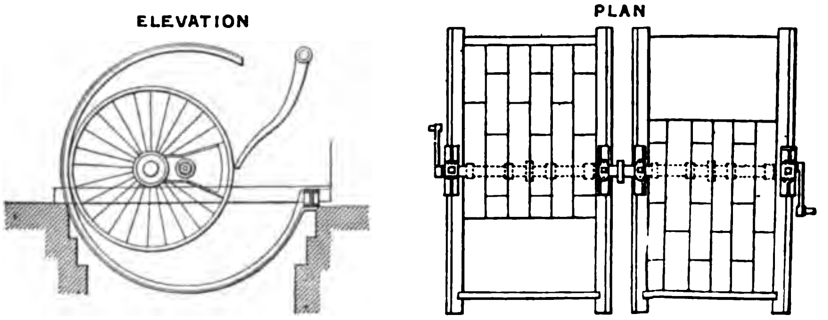


Fig. 377. Fig. 378.

Cooke's Ventilator.

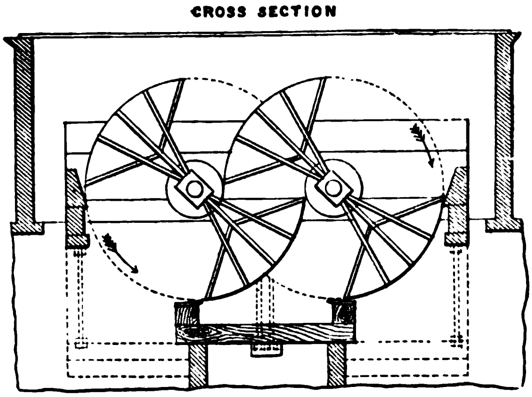


Fig. 379.—Root's Ventilator.

balance each other. In the Report of the North of England Mining Institute Committee the useful effect of this ventilator working at Hutton Henry Colliery, Durham, is given at 37·33.

Root's ventilator (Figs. 379 and 380) is a rotary displacement machine, which may be used to exhaust the air out of a mine, or as an air-compressing machine. It has two rotary pistons of the same size placed on separate shafts, and revolving in a casing having inlet and outlet openings. These may be at the top and bottom or at the sides, but the outlet opening communicates with the atmosphere. The pistons are worked by gearing on the shafts, and they revolve without being in actual contact with each other or the casing, but the clearance between them is only ⅛th of an inch. They discharge the air in four distinct volumes during each revolution. The engines are placed at right angles to the ventilator, as seen in the drawing. The percentage of useful effect given in the Report previously referred to, for the Root ventilator working at Chilton Colliery, Durham, is 47·84.

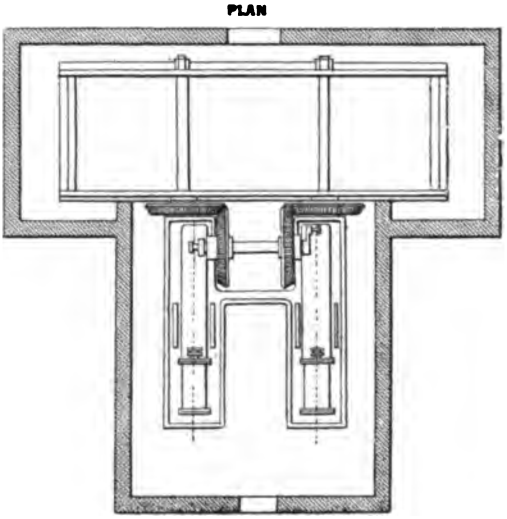


Fig. 380.—Root's Ventilator.

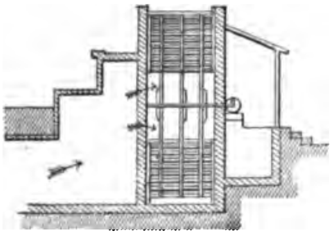


Fig. 381.

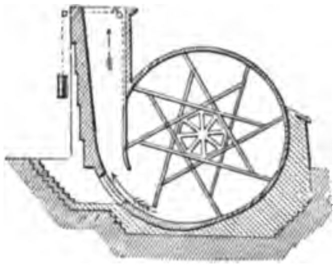


Fig. 382.

The Guibal Ventilating Fan.

The objection to all these varying capacity ventilators is that they are not simple in construction, are liable to be deranged, and (excepting the Struvé) give a low useful effect.

The *centrifugal* are the most popular of all the ventilators. Early forms of these were Nasmyth's, Brunton's, and Biram's fans, but those now most in use are the Guibal, Waddle, and Schiele, though several others are before the public, one of the most recent being the Capell.

Of these, perhaps the *Guibal* (Figs. 381 and 382) has met with most favour, but all three fans have merits of their own. The Guibal varies from 20 to 50 feet in diameter, it has 8 or 10 blades inclined backwards and curved at the tips. The air can enter at one side or both. The casing is not attached to the fan, but the latter is placed in it. The air is discharged at one particular place through a sliding or adjustable shutter, so that the exit opening may be regulated into an expanding chimney, larger at the top than at the bottom, the effect of which is to gradually reduce the velocity before reaching the outside, where the current is out of the influence of the fan. The sliding shutter allows of the

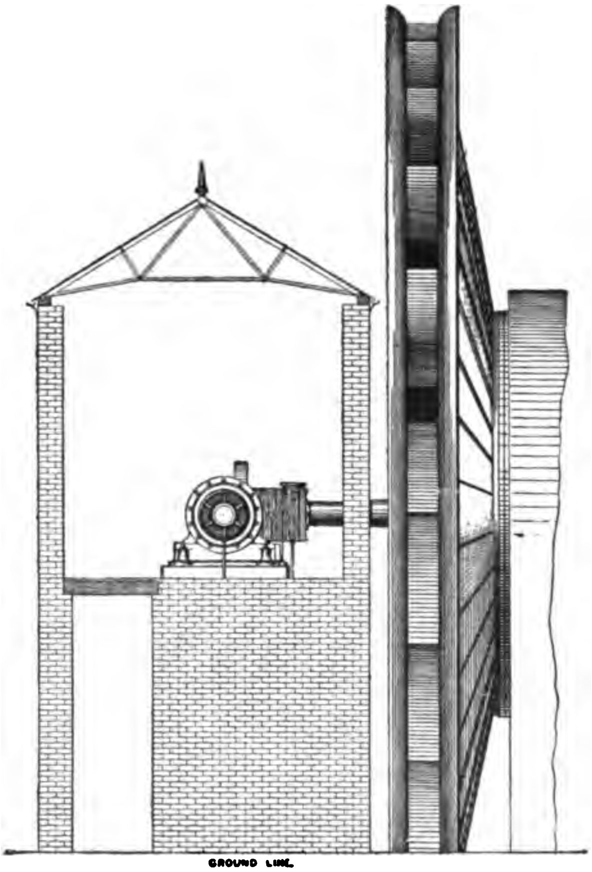


Fig. 383.—The Waddle Ventilating Fan.

outlet being regulated to suit the volume of air under various circumstances. The most effective opening of the adjustable shutter is ascertained and fixed at any particular colliery experimentally. The percentage of useful effect given in the same Report as quoted before, is 40 for the Guibal ventilator at Hilda Colliery, South Shields, but as 52·95 for that at Pemberton Colliery, Wigan.

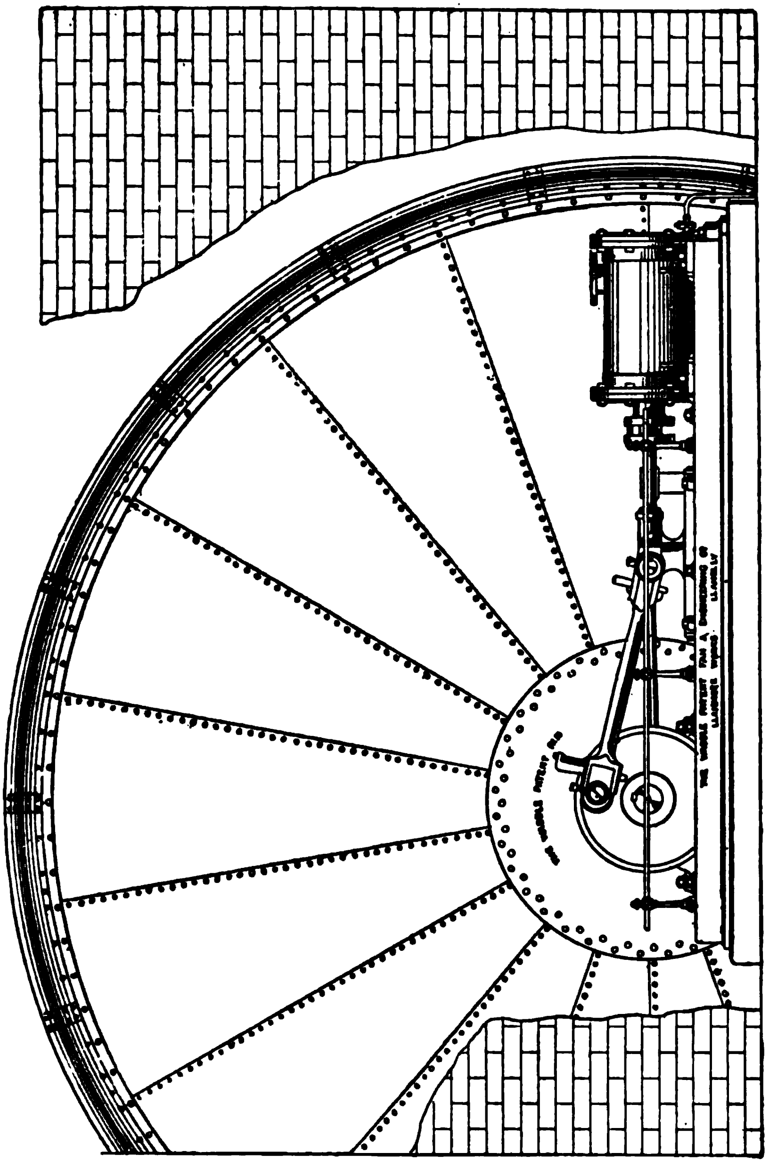


Fig. 384.—The Waddle Ventilating Fan.

The *Waddle* (Figs. 383 and 384) is an open-running fan, because it delivers the air all round the circumference into the atmosphere. For this reason its width is reduced at the periphery, and it is therefore very narrow in proportion to its diameter. The air is received on one side only. The blades are inclined backwards, these and the casing are all in one revolving piece which works completely free of vibration; when well built it forms an extremely compact, rigid ventilator. A high velocity of periphery is obtained by a moderate number of revolutions. The Report of the North of England Mining Institute Committee gives a useful effect of 52·79 for this fan at Celynen Colliery, South Wales.

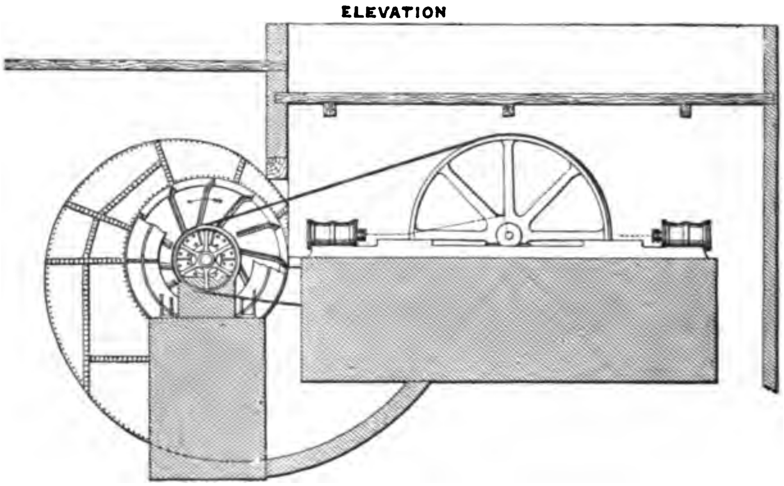


Fig. 385.

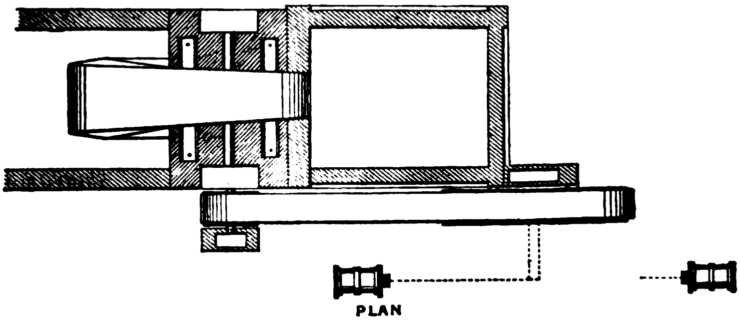


Fig. 386.

The Schiele Fan.

A 30-foot one at Cwmaman Colliery—when tested in October, 1888, by the combined committees of the North of England, the Chesterfield, and the South Wales Institutes of Mining Engineers—gave a useful effect of 55·8 per cent. and circulated 134,394 cubic feet of air per minute with a water-gauge of 5·53 inches. Mr. Hugh Waddle, the inventor and patentee of the improved fan (Figs. 383 and 384) made by the Waddle Patent Fan and Engineering Co., Llanmore Works, Llanelly, asserts that it utilises from 15 to 20 per cent. more power than the old form of fan made by the predecessor of the Waddle Patent Fan and Engineering Co.—viz., the late Mr. J. R. Waddle—and erected at Celynen, Cwmaman, and over 200 other collieries.

The *Schiele* (Figs. 385 and 386) runs fast, the moving part of the fan being small, and constructed wholly in wrought iron, the heaviest portions being disposed round the centre. The disc or blades of the fan taper from the tip, widening towards the centre. This disc revolves between two cast-iron side walls of a section following the taper of the blades. The air enters at each side of the fan in equal quantities. The casing of the fan is in wrought iron, and takes the form of a gradually-increasing volute air-chamber, surrounding the periphery of the blades, and culminating in the exit which forms the widest point of the air-chamber. The object of this is to give a uniformly-increasing space beyond the tips of the fan blades, from nothing up to the full area of the discharging aperture. The Schiele, like the Guibal, has an expanding chimney, so that the air is not passed directly into the atmosphere, but, owing to the increased area at the outlet end of the chimney, the air has a gradually-reduced velocity, moving up through the chimney more and more slowly until it meets the external atmosphere. Some of the Schiele fans are driven direct from the crank shaft of the engine, but usually the power is transmitted by belting, the engine having a broad fly-wheel for the belt, and another, of small diameter, fixed on the fan shaft. This arrangement is necessary to obtain a proper velocity of fan with an economical speed of the engine, and, moreover, prevents the shocks of the fan being communicated to the engine, thus resulting in much less wear and tear in the engine. In the report referred to in connection with the other ventilators, the Schiele is said to give 49·27 per cent. of useful effect at Car House Colliery, Rotherham.

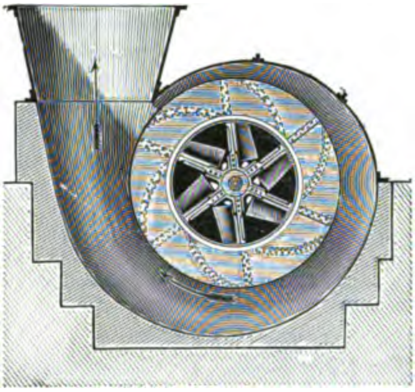


Fig. 387.—The Capell Fan.

The new *Capell* fan is shown in Fig. 387, and is thus described by the inventor, the Rev. G. M. Capell, of Passenham, near Stony Stratford.

This fan is the result of numerous experiments, and in practice is said to have proved itself 30 per cent. more powerful than the original Capell fan, while the cost is reduced. The alterations consist principally in placing the inner wings at an angle, and terminating them in the fan inlet by special shaped *scoops*, which present more surface near the centre of the fan, than the point where they join the inner wings. They act as collectors, and their effect is to increase gauge and volume. The inner wings, being inclined, lead the air across the fan, to be discharged by the outer wings, which are parallel to the axle.

These fans are made with single or double inlets.

Only a little masonry is required in their fixing, especially for single inlet fans. They are made in diameters to 25 feet for direct driving, with a special internal regulator, as open running fans. This new construction results in a stronger and lighter fan, whilst also adding to its stability when in work. The new fan is now in use at the following places:—Garswood Colliery, Wigan (2); Allerton Main, near Leeds; Hutton Henry, near Wingate; Sanquhar, N.B.; Shipley Collieries, Derby (2); Backworth Colliery, Newcastle-on-Tyne; Moon Run Colliery, Pennsylvania; Spring Valley Colliery, Illinois, United States; besides several in Germany and Austria.

The Moon Run fan has beaten all records in the United States. It is 8 feet in diameter with two inlets. At 215 revolutions its volume was 140,000 feet, W. G. 1·5 inch. At its guarantee speed of 350 revolutions its volume is 210,000 cubic feet and W. G. 3·5 inches. It is too soon to tabulate the work of the new fan, but the comparative results of the old and new in work are given here, four fans of equal sizes, under old and new types, being placed in contrast:—

Capell Fan (Old Patent) at Stanton Iron Works Colliery, near Mansfield.

Diameter	12ft. 6in.	Water Gauge . . .	4·1in.
Width	11ft.	Cubic Feet per minute measured in returns . .	199,500
Two Inlets . . .	7ft. 6in.		
Revolutions per minute . .	215		

Capell Fan (New Patent), at Allerton Main Colliery.

Diameter	12ft. 6in.	Revolutions per minute . .	215
Width	10ft.	Water Gauge . . .	4·5in.
Two Inlets . . .	7ft. 6in.	Cubic Feet per minute . .	240,000

The New Fan has less width, and gives more air and a higher gauge.

Capell Fan (Old Patent), Single Inlet, at Digby Colliery, Nottingham.

Diameter	15ft.	Revolutions per minute . .	220
Width	5ft. 6in.	Water Gauge . . .	6in.
Inlet	9ft.	Cubic Feet per minute . .	149,200

Capel Fan (New Patent), Single Inlet, at Hutton Henry Colliery, Wingate, Co. Durham.

Diameter	15ft.	Revolutions per minute . .	220
Width	5ft. 6in.	Water Gauge . . .	6·14in.
Inlet	9ft.	Cubic Feet per minute .	216,700

This difference of volume at same gauge is very remarkable, being no less than 67,500 cubic feet per minute, at a rather higher gauge, both fans being of equal cubic contents.

The useful effect of a Capell fan varies with the water-gauge. Thus the 12 ft. 6 in. by 3 ft. single fan at Saare et Longchamps mine, Belgium, passes

113,200 cubic feet per minute at 8 inches water-gauge and gives a useful effect of 61 per cent., while the electric driven 8 ft. by 3 ft. 6in. single fan at Youghiogheny River Colliery, Pennsylvania, when running at 220 revolutions, passes 85,600 cubic feet per minute at 1·05 inch water-gauge, and gives a useful effect of 88·4 per cent.

The high results of electric driven Capell fans have been repeatedly shown in large installations on the Continent.

There are upwards of 220 collieries ventilated by Capell fans, and in Germany gauges up to 14 inches have been obtained.

The material used in the construction of these fans is of the highest quality of Siemens mild steel. The shafts are of either forged or compressed steel, the suspensory arms and minor forgings being of finest quality iron, and Siemens steel castings are used for attaching the fan centre to the shaft. Special care is taken to provide long bearings for the shaft, in order to ensure cool running.

SINGLE-INLET ENCASED FANS.

Diameter.	Width.	Revolutions.	W. G.	Volume in cubic feet per minute.
FT. IN.	FT. IN.			
8 0	4 0	300	2·5	75,000
10 0	4 6	240	3·0	95,000
12 0	5 4	210	3·5	115,000
12 6	5 8	210	3·75	130,000
15 0	6 6	180	4·5	170,000

DOUBLE-INLET ENCASED FANS.

Diameter.	Width.	Revolutions.	W. G.	Volume in cubic feet per minute.
FT. IN.	FT. IN.			
8 0	7 0	300	3·0	150,000
10 0	8 6	240	4·0	170,000
12 0	10 0	210	3·0	210,000
12 6	11 6	210	3·5	250,000
15 0	11 6	180	4·5	350,000

The above fans will, in all cases, give a greater volume and W. G. at a higher speed; or a greater volume at a lower W. G. at the same speed; or a higher W. G. at a reduced volume at the same speed.

These tables are useful as indicating the capabilities of the fans, but the actual figures in the last two columns for any colliery can only be ascertained experimentally. A fan working at a certain number of revolutions and yielding a certain result at a colliery, will probably yield a different one when working at the same speed at another. Again, the conditions under which it works at the same colliery vary from year to year as the workings are extended, and the air-courses lengthened or altered.

Mr. William Hopton, of St. Helens, Lancashire, has invented a fan, in the shape of a drum, having eight blades or scoops. Colliery managers and others connected with collieries will be much interested in trials and figures connected with experiments of this new fan.

Mr. Arnold Lupton has patented a fan in which he has endeavoured to combine the best qualities of other fans. It is called the *Medium* fan, being made of a size between the Waddle and Guibal on the one hand, and the Schiele and Capell on the other. It is a compound of the Waddle and the Guibal. The results of experiments showing the capabilities of this fan are awaited with much interest.

The principle on which all centrifugal fans act is this:—When the fan revolves the centrifugal force drives the air from the centre, which tends to create a vacuum, and air rushing in to take its place a current is produced. In general, ventilators are used to exhaust the air, and are, therefore, placed at the top of the upcast shaft, but they are capable of being used as forcing or compressing machines, in which case, of course, they would be placed at the top of the downcast. The object to be attained, whether by furnace or a machine, is to make the two columns in the upcast and downcast of different densities, the difference in the weight of the two columns of air of the same height forming the ventilating pressure. Where a shaft, with a fan working on it, is required for winding as well, it is necessary to have some covering over the top of the shaft, to prevent the entry or exit of air. For this purpose, covers are placed over the shaft, one of which is lifted when the cage ascends, the cage itself then filling up the opening. When the distance between the fan drift and the pit top admits of it, another plan is to place two sets of doors in this space, and arrange these so that the cage on leaving the surface opens the lower doors, at the same time closing the upper, and, on coming to the surface, it opens the upper doors and closes the lower.

It is desirable in most cases, and indispensable in very fiery collieries, for a ventilating machine to have two separate engines, either of which would be capable of working it, and these engines should be placed on opposite sides of the fan. The object of this is to provide a second means to work the ventilator in case of an accident to the first. A better plan is to have a duplicate fan and engine connected with the fan drift, so that if an accident occur, either to the fan or to the engine, the ventilation of the colliery would be ensured. In such a case each engine and fan could be worked alternately, for a week or fortnight at a time, so as to maintain them in good working order.

Fans of the Guibal and Waddle class, with high periphery speed resulting from a few revolutions per minute, have engines coupled direct, but small fans, like the Schiele or Capell, must make more revolutions per minute than the others to get a high circumferential velocity, and these are usually driven by broad straps, and the engines may be geared to suit the speed required.

These small fans make from 100 to 300 revolutions per minute, and a fan-engine should not have a greater piston speed than 250 feet per minute. The most effective engines work expansively, and are condensing, and compound engines are the most economical in working.

As regards the economy or efficiency of furnaces, much depends on the depth of the upcast shaft, but with fans the depth is not an element requiring consideration. Besides being in every case a much safer and steadier means of producing ventilation, fans undoubtedly give better results in shallow mines. No doubt a furnace can be advantageously applied in a deep, dry shaft, the bottom of which is considerably below the bottom of the downcast. If, however, the mine be fiery, as most deep mines are, a fan should be used to produce the ventilation, and in arranging the upcast and downcast pits the air should be made to descend that sunk to the lowest level, and by what is called "ascentional ventilation" pass into each district. This is the most natural, and also the safest, means of ventilating, for the currents, on becoming warmer by their passage through the mine, are more easily carried up-hill than down, and, moreover, the gas is much more readily carried away with ascentional ventilation.

As the air, if allowed to follow its own course, would go by the easiest and shortest route, from the downcast to the upcast, the different currents must be guided by stoppings and doors into the various divisions of the mine. With a downcast 15 feet in diameter, giving an area of 176·715, there might be five separate currents branching off, each 35 feet in sectional area, without any increase in velocity. The benefits of thus splitting the air may be briefly stated. The same ventilative power will produce a larger volume of air by splitting than by carrying one body or current of air from the downcast round the workings to the upcast. Each district is supplied with fresh air, which comes from the downcast and returns to the upcast quite independently of the other currents, so that each is thus rendered purer and pleasanter for the workmen.

But as it almost invariably happens that the lengths of these air-currents are unequal, the air if allowed to take its own course would not flow in equal quantities into each division, or in such other proportion as from the circumstances of each district may be desirable. If left to itself the shortest air-course would probably get the largest share of the air; but there are other circumstances, such as sectional area and rise or dip workings, which would affect the quantities flowing into each district. To balance these splits, if they are very unequal, regulators are fixed in such a way that we can enlarge or diminish the aperture through which the air passes at pleasure. Usually this is done by means of a sliding door which moves horizontally in a groove in the wooden framing. Sometimes the door slides vertically, but whichever way it moves it should be kept locked in its proper position, by a properly-appointed person. It does not matter whether these regulators are placed in the intake or return airway, so far as regards restricting the passage for the air, but as they would form obstructions in the intakes which are used generally for haulage, they are placed in the return air-ways. The amount of opening for the passage of air is determined by the circumstances of the mine, and the relative quantities desired for each district.

Care must be taken both in the construction and erection of a regulator, a skilful workman being employed to fix it. The frame may be square or rectangular, and must be well morticed together. The kind and size of timber used depend upon the size of opening and its position, but must always be of sufficient strength. The grooves, whether formed at the sides or at the top and bottom, must fit the sliding shutter accurately, and yet allow of its being moved smoothly and easily. The boards used for the door must be a proper thickness and tongued and grooved in order to ensure tightness at the joints. The wood used in the construction must be well seasoned to prevent subsequent warping.

Fig. 388 shows a regulator formed in an ordinary underground door which is hinged at the side and provided with a latch. It is hung to ensure its self-closing, and, if intended only for occasional use, may, at other times, be kept locked. E is the door-frame having a little brickwork built over and at the sides to ensure a close fit and prevent the passage of air. A is the sliding shutter moving horizontally in the slots B. C is the door opening on the hinges D. The shutter is kept securely in its proper position to form the desired opening by means of the chain and padlock shown, or by some other equally satisfactory fastening. There should be in use only one padlock key kept by the man having charge of the ventilating arrangements.

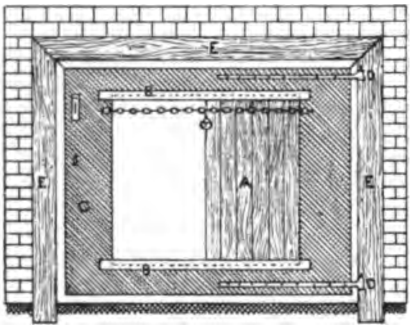


Fig. 388.—Regulator or Scale Door.

Where the door C is not required in the return airway a smaller door-frame may be used. This is fixed in surrounding brickwork built to reduce the size of the air-way, which, however, must not be too small for the formation of the largest air passage likely to be wanted. The amount of opening then left by the locked sliding shutter must be sufficient for a man to pass through to allow of all parts of the return airway being available as a travelling road.

However carefully regulators and doors are erected, they require constant supervision and re-adjustment, for the weight of the superincumbent strata squeezes and displaces them. If neglected, breaches in the masonry and timber, sooner or later, appear, and then the air finds its way through crevices and large openings behind the framing and through the open spaces in the broken or badly-fitting door. Where the inclination of the strata is great, there is constantly a weight on the high side of a level course road, tending to push a door or timbers placed in it towards the low side. This is more noticeable in deep collieries.

Where it is necessary to have regulators placed in airways traversed by horses, it is usual to hang sheets of brattice cloth from the roof across the road. This is a very convenient arrangement, as the air passes underneath the sheet which forms no impediment to the horses, but yields to their pressure on either side and falls into place by its own weight when the horses and trams have passed.

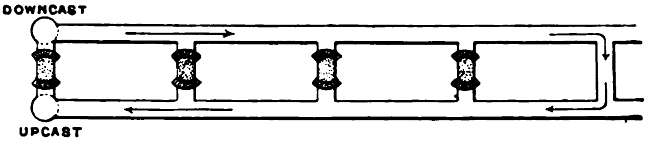


Fig. 389.—Permanent Stoppings in Fiery Mines.

The effect of placing regulators in the airways is fully dealt with later.

The stoppings used to prevent the air from being diverted from the desired course are variously constructed. For a non-fiery mine it may be found sufficient to build a brick or stone wall across the mouth of the road leading out of that intended for the main air-course and to back it up with rubbish to prevent leakage. In the case of fiery mines, however, these stoppings cannot be too strongly built, as they may possibly have to resist the effect of an explosion, and it must be remembered that a blown-out stopping allows the intake air to return through it to the upcast shaft. The best stoppings have two walls 15 or 20 feet apart, each with a curved outline, the convex sides of which are presented outwards and the space between the walls filled closely with stone or rubbish (see Fig. 389).

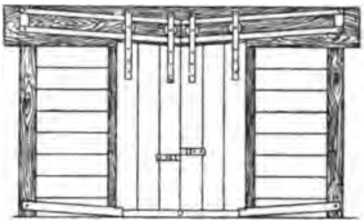


Fig. 390.

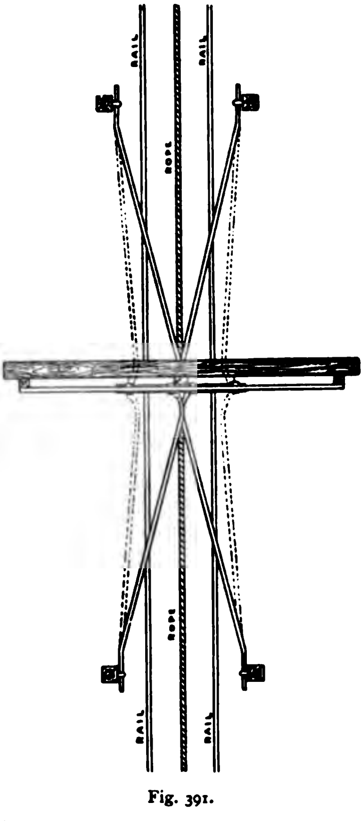


Fig. 391.

Figs. 390 and 391.—Elevation and Plan showing a Self-acting Ventilating Door used on a Main and Tail Rope Engine Plane at Hetton Colliery.

If only stoppings were built strong enough to resist the force of an explosion of fire-damp, the air-currents, unless stopped by falls, would still flow into the workings after the explosion, and many lives might thus be saved which would otherwise be lost through the after-damp. Occasionally it is necessary to have a travelling road through some of these bye-roads, and it is then impossible to have the way bricked up altogether. Doors of course must be fixed in such cases, but as it is impossible for these to resist the effects of an explosion, they should be avoided if possible. Where they must be put, they should have considerable attention paid to their design. If only required for travelling through, the frame should be set in masonry and the doors hinged from the top and open towards the intake side. This will ensure their falling to, despite of any want of thought on the part of the person passing through, and if instead of being placed vertically they form an angle of 70° or 80° with the floor, they will remain closed, the force of the intake air helping to keep them firmly in that position. Where the road has to be used both for travelling and the passing of tubs, a different kind of door is required so as in opening to allow the horse and tubs to pass. The framing of such doors should not be set quite upright, but sufficiently inclined for the door to fall and close by its own weight. A boy is stationed near to open it as required. Two similarly-constructed doors or more should be used between all main intake and return airways, and wherever the escape of air during the passing of the tubs is important, they should be placed at such intervals apart as to allow of the train or set of tubs and the horse to be between them, to prevent the two doors from being open at the same time. In fixing these doors care should be taken to make them airtight at the different joints and spaces behind the framing. Safety-doors are sometimes fixed in the roof, so that in the event of an explosion blowing the ordinary doors away, these safety-doors may fall into use or be dropped as soon as possible, and so take the place of the others.

Figs. 390 and 391 show the elevation and plan of a self-acting ventilating door used on a Main and Tail rope engine plane at Hetton Colliery, in the county of Durham, a description of which appears in *The Iron and Coal Trades Review* of July 31, 1891. The door is made in two divisions, which do not swing on hinges in the usual way. Each division is hung by means of pulleys which run on rails above the door. The rails are placed one above the other, and are secured to a stout beam crossing the road. Such an inclination is given to these rails as to cause the two sections of the door to run together by their own weight, and close tightly. The pulleys are secured by iron straps to the door, as shown in the drawings, and can be moved backwards and forwards so as to give a side motion to the halves, causing them to separate or come together without much friction.

About two feet from the bottom of the door, and at the junction of the two divisions, are placed hinges, each of which receives one end of a stout piece of angle steel, about 8 feet long. The other end of each of these bars rests in an eye-bolt secured to a road-post. Each bar is free to move backwards and forwards through the eye-bolt, which is placed as near the rails as possible without actually touching the tubs when passing. A similar pair of these bars is placed on the in-bye side of the door to the pair on the out-bye side. One rope works under the door, the other at the side.

When the first tub of a set going in or out comes in contact with one of the pair of bars, the pressure brought to bear on the doors causes them to open quietly and gradually, without jerk or shock, and when fully open the bars take up the position shown in dotted lines in Fig. 391. When the last tub of a set has passed through the door, the pressure being removed from the bars, the door, if in working order, closes tightly of its own accord.

Two of these doors are placed at such a distance apart as to ensure that the longest set of tubs run on that particular plane shall be clear of one door before opening the other.

It is often necessary to take one current of air over or under another. Usually the "returns" are taken over the "intake," and this is done by means of an air-crossing or air-bridge. In fiery mines these should be made so strong as to resist the effects of an explosion, and the better plan is to pass the return below the intake. In all explosions there is more force exerted in the intakes than returns, and it often happens that an arch of a crossing is lifted up and blown away. Wherever practicable the intake and return should be separated by some feet of solid rock. Fig. 392 shows an air-crossing suitable for a fiery mine. A great deal of care must also be taken in fiery mines to keep the brattice well up to the face of the workings, and if the brattice does not equally divide the road the smaller side should be the intake. The brattice usually consists of canvas nailed to the posts and divides the road into two equal divisions vertically, providing one of such divisions is large enough for the passage of the tram or tub, or if not, one division is made wider than the other. Brattice may also be made by nailing boards to the posts, and for a more permanent brattice, in the case of a stone drift to be driven a considerable distance, a division is formed by building a brick brattice.

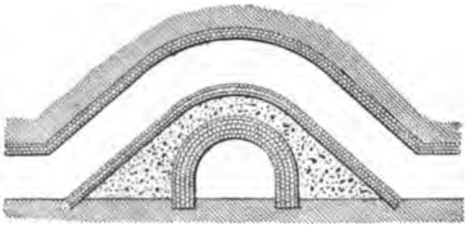


Fig. 392.—Air-Crossing.

As to the velocity at which the air is taken along the roads much will depend upon the circumstances, but it is always desirable to have large air-courses, and as each split of air must have from 5,000 to 6,000 cubic feet per minute, and perhaps more, there would be in a road 6 feet by 5 feet, or 30 feet area, a velocity of

$\frac{6000}{30}$ = 200 feet per minute for the latter quantity. There is no inconvenience in such a velocity, but if the same quantity of air were taken along an extremely small road the velocity would be unpleasant to those traversing it. A velocity of from 100 feet to 300 feet per minute is safe, convenient, and economical. At some collieries the main airways are swept by current velocities of 1,200 feet per minute. If an ordinary Davy lamp be used at the colliery 300 feet per minute is the extreme velocity the air should travel at in the return air-courses, where the air is mixed with fire-damp to a dangerous degree, because the flame of an ordinary Davy lamp will pass the gauze in a current of 6 feet per second, and it would not be safe to use such a lamp in currents approaching the explosive point. In shafts, in the main intake and return airways, the velocity cannot in practice be kept so low as in the further parts, because the former are necessarily limited in number. From 900 to 1,000 feet per minute is a fair velocity for the air to travel at in the shafts, and generally if it is required to pass from 150,000 to 200,000 cubic feet per minute, the diameter of the shaft should be not less than 14 feet.

There is a difference of opinion as to whether the upcast or downcast should be the larger. Owing to the expansion of the air, the volume ascending the upcast would require more area than the downcast, if the same velocity were maintained in the two shafts. But there is no absolute necessity for this, and the velocity of the air may be much higher in the upcast shaft. Irrespective of areas, it is more important that the upcast shaft be on the rise side of the downcast, but in the case of two shafts of equal depth whose surface levels are the same, a slight advantage will be derived from making the shaft whose area is least the downcast, so that the velocity of the air may be the same in each shaft. Other considerations may, however, suggest a different arrangement; for instance if a furnace were used, the larger shaft being wet, and the smaller one dry, the circumstances point to the advisability of making the larger shaft the downcast. Again, where a furnace is used, there may be wire ropes or tubbing that the smoke of a furnace would injure in the one shaft and not in the other, so that in making a decision as to which shaft shall be the upcast, the particular circumstances of the case must be well considered.

Experiments have been made with fans worked by compressed air placed underground at long distances from the bottom of the shaft. The object of these fans is to assist the main or general ventilating power. Where there are far-off splits otherwise difficult to ventilate, these auxiliary fans give good results and increase the total quantity of air circulating.

The usual method of ascertaining the quantities of air in mines is by the use of the *Anemometer*—an instrument for measuring the velocity of the air. The wind gives the vanes of the anemometer a speed proportional to that of the current, and the number of revolutions is registered upon the face of a dial fixed on the central part of the instrument. Biram's anemometer is the one in general use, and in it each revolution of the first pointer corresponds to one hundred feet in the linear motion of the air. To get the number of cubic feet passing per minute, multiply the velocity per minute, or in other words the recorded revolutions per minute by the sectional area of the airway. A slight correction should, however, be made owing to the friction of the anemometer. This correction is supplied by the maker along with the instrument.

Another method of air measuring formerly practised in mines worked with naked lights, but now seldom used, consists in exploding a small quantity of gunpowder and noting the time the smoke takes to traverse a certain distance, from which the velocity of the air is ascertained. The method of holding the anemometer is worthy of remark, as different results may be obtained in the same airway owing to the difference in the velocity of the current at different parts of the airway. It should be held at arm's length in front of the body, the vanes should be kept square with the current of air, and the anemometer should be slowly moved uniformly over the whole area of the airway from a point near the floor to a point near the roof. Several trials should be made in the same place and the average result taken. In large airways and fan drifts, where the greatest accuracy may be required in testing the efficiency of fans, the most correct method is to fix fine wires or strings across the road from side to side and from floor to roof, at regular distances apart, the one set of strings being placed at right angles to the other, so as to divide the airway into a number of even divisions, and then to note the revolutions of the anemometer during 1, 2, or 3 minutes at each division. The mean of these are taken. But the operation should be repeated in cases where great accuracy is desired, and if the mean result differs materially from the first mean result, it will be necessary to go through the operation again, and perhaps again.

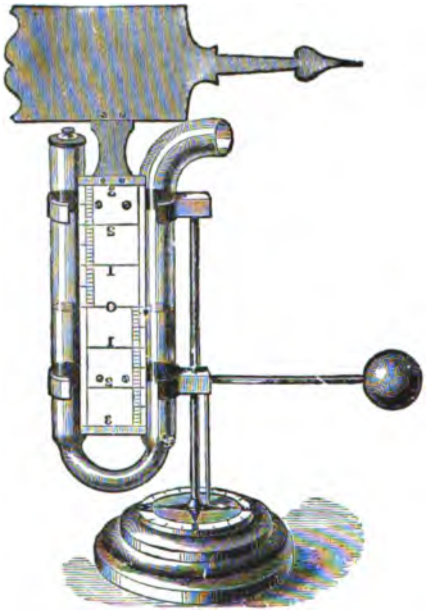


Fig. 393.—Lind's Anemometer.

Wolf's Anemometer.—Wolf is said to have invented the first anemometer. It has four sails, like those of a wind-mill, which turn on a horizontal axis; this axis is connected by wheel-work with another in which is inserted one end of a bar carrying a weight. The wind pressing upon the sails causes this bar to turn in a vertical plane; and, when it is in such a position that the weight on it counterbalances the pressure of the wind, the angle which it makes with a vertical line passing through the axis affords a measure of the wind's force.

Wolf's anemometer is no longer in use, as it has been superseded by later forms.

Lind's Anemometer.—The anemometer or wind-gauge invented by Dr. Lind is shown in Fig. 393. It consists of two glass tubes about 8 or 9 inches long and about $\frac{1}{4}$th of an inch in diameter. As now made they are placed vertically, and connected at the base by a small bent glass tube only $\frac{1}{10}$th of an inch in diameter to check oscillations of the water caused by sudden gusts of wind. The upper end of one tube is bent to receive the wind blowing horizontally into it, while the upper end of the other is covered by a brass cap with a hole in it. The hole allows water in the tube to be under the influence of atmospheric pressure while the cap shields the surface from the action of gusts of wind. The instrument is upheld by two brass clips, one near the top and the other near the bottom; the clips terminate on one side in square ends. The square end of the lower clip is pierced right through by a small circular opening, and the square end of the upper one is partly bored out by a similar sized opening. By this means the instrument can at any time be placed on the steel spindle, which is screwed vertically into the stand, or taken off for part-filling. A scale of inches and tenths is fixed between the two legs of the glass tube, the zero point of which is about half way up the tubes. The scale is graduated above and below its zero line, the upper divisional lines being cut on the opposite side to that of the lower markings so as to suit the water displacement in the two limbs. Before using the instrument water is poured in until the tubes are half full and the scale is then adjusted with its zero line level with the water in the tubes. The stand must be placed perfectly level to ensure the same height of water in the two tubes before it is disturbed by the wind. The whole instrument being free to turn easily about the upright spindle, the mouth of the bent tube is made to face the wind by the application of a weather-cock. The water is then forced down the one tube and correspondingly up the other. The difference between the heights of the surfaces of the water in the gauge will be the height of a column of water, whose pressure is equal to the force or momentum of the wind, blowing against an equal base.

The stand has the cardinal points marked and as the direction of the wind changes, the instrument responds, carrying with it the index-pointer, which moves round the stand dial-plate and thus shows the direction of the wind. The projecting handle terminating in the knob on the right hand side of the illustration is for lifting the instrument on and off the stand, and for balancing it on the spindle.

Robinson's Anemometer.—This has been much used at meteorological stations. It was invented by Dr. Robinson, of Armagh, and has been modified and improved by Mr. Casella. It consists of four equi-distant hemispherical cups upon which a passing current of air acts, see Fig. 394. The cups are fixed to four folding horizontal arms which at their other extremity are attached to a vertical shaft or axis. The wind presses forcibly on the concave surface of a cup when its diametrical plane faces it, and feebly on the convex surface of the cup on the opposite arm which is at the same time presented to it. The difference in the pressure of the wind on the two cups causes the arms and the vertical shaft to revolve. Each cup in turn receives the full force of the wind, and the speed attained by the cups amounts to one-third of the wind's velocity. A thread is cut at the lower end of the vertical shaft, into which a toothed wheel fits. The revolving shaft causes this wheel to move on its axis, and, by means of gearing, this in turn gives motion to the other wheels, which carry suitable indices showing the space traversed by the wind for any timed period the graduations are capable of registering.

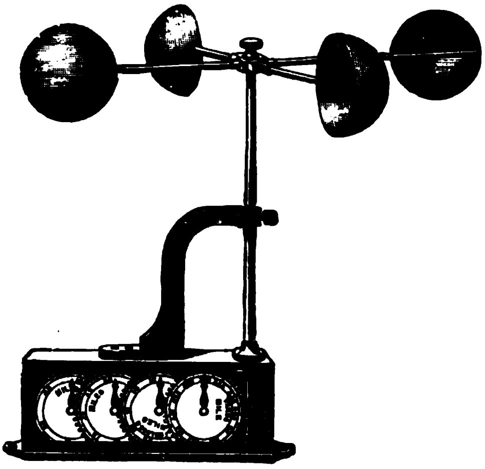


Fig. 394.—Robinson's Anemometer.

Robinson's anemometer folds up and is carried in a small case. It is very useful for registering the velocity of the wind in the open, but is not so well adapted for use in mines.

The *Biram* anemometer has been manufactured by Messrs. John Davis & Son, of Derby, since the year 1847, and is adapted to indicate the velocity of air currents in mines, sewers, or any confined passage within certain limits. It is made in 6-inch, 4-inch, and 3-inch sizes, as shown in Fig. 395, and in a 12-inch size with a different arrangement of the vanes. In the illustration the ten vanes are set at an angle of 45° to the axis, like the sails of a wind-mill. There is a central dial-plate, the circle on which is graduated up to 100, and round which the first pointer moves. The axis of the vanes carries a pinion which gears into a toothed wheel, with which this first pointer is connected. There is a second pointer moving round a small circle in the lower portion of the dial-plate like the second-hand of a watch, and this is marked from 0 to 10. The mechanism actuating the pointers is carried in a central cylindrical box. The arrangement of wheels and pinions within the central case is such that when the revolutions of the vanes correspond to 100 feet in the lineal motion of the air the first pointer moves completely round its circle while the second pointer moves only the $\frac{1}{10}$th part of its circle. A small lever or disconnector is placed conveniently near the handle by which the instrument is held, so that the registering apparatus can be thrown into or out of gear at will. Legs to support the instrument and a handle are attached directly to the cylindrical frame, which also carries the central box by suitable bracing. The fan wheel is free to revolve within the frame through the bracing pieces. As thus arranged the anemometer may be read up to 1,000 feet, but other forms of self-registering apparatus are applied to the 12-inch and sometimes to the smaller anemometers, allowing of a reading up to 10,000,000 feet. The apparatus consists of six small circles marked respectively X, C, M, XM, CM, and M, the divisions on which denote units of the denominations for the respective circles; in other words, the X index in one revolution passes over its ten divisions and registers 10 by 10 or 100 feet; the C index in the same way 1,000 feet, and so on. Thus, the observer has only to record the position of the several indices at the time of his first observation, by writing, in their proper order, the lower of the two figures on their respective circles between which the index points, and to deduct the amount from their position at the second observation, in order to ascertain the velocity of the air which has passed during the interval.

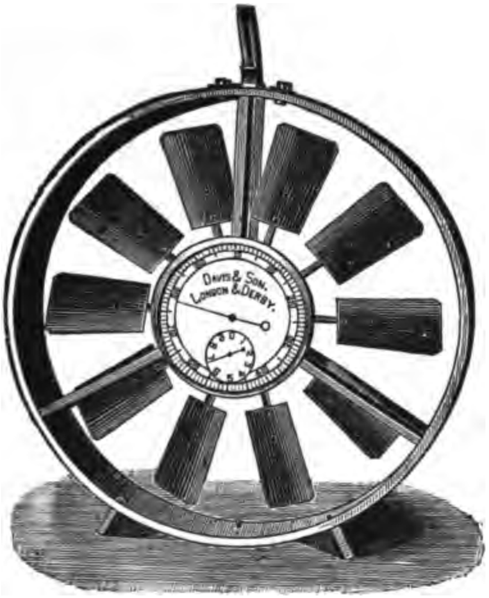

DAVIS & SON.
LONDON & DERBY.

Fig. 395.—Biram Anemometer.

The velocity of the wind in feet per minute divided by 88 will give the velocity in miles per hour, because $\frac{1,760 \times 3}{60} = 88$. To find the force of wind, multiply the square of the velocity of the wind in feet per second by ·0023.

If the velocity of the current in an airway 6 feet by 6 feet be 300 feet a minute, the quantity of air passing would be $6 \times 6 \times 300 = 10,800$ cubic feet per minute.

To ascertain the state of the ventilation in the different parts of the mine, the anemometer is put into a case and carried from place to place.

Fig. 396 shows Casartelli's improved Biram's anemometer, with aluminium fan, the 4-inch size of which reads up to 1,000 feet. The instrument is made by Mr. Joseph Casartelli, of Manchester. The fan is pivoted at both ends of the axis; one pivot being long, enters the case containing the recording wheel work.

On this pivot is fixed a toothed wheel which gears into suitable intermediate wheels for reducing the revolutions of the fan into feet, and communicating with the pointer on the dial. If there are two or more dials other corresponding wheels are supplied, gearing into each other. In Fig. 396 there are two dials of equal size placed near the centre of the fan. The disconnector acts by throwing the fan spindle in and out of gear, and is conveniently placed near the handle of the instrument.

The watch form of Biram's anemometer, as made by Mr. Casartelli in hunter case for carrying in the pocket, is shown in Fig. 397. It is 2 inches in diameter, and reads up to 1,000 feet.

In order to reduce the friction as far as possible anemometers are very delicately constructed and are quite unsuited for rough or careless usage. Even when in perfect condition, however, they will not measure a very light current of air because the force is not sufficient to cause the vanes to revolve. The best anemometers require a current with a velocity of 30 feet per minute to cause motion of the fan wheel, and many require a much higher velocity. The fewer the number of parts in the mechanism the less will be the friction. In perfecting the instrument for higher readings the friction is slightly increased.

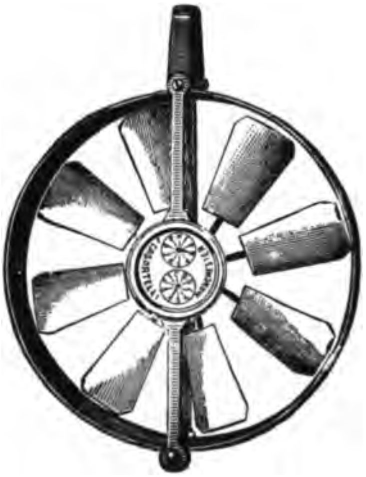


Fig. 396.—Casartelli's Improved Biram's Anemometer.

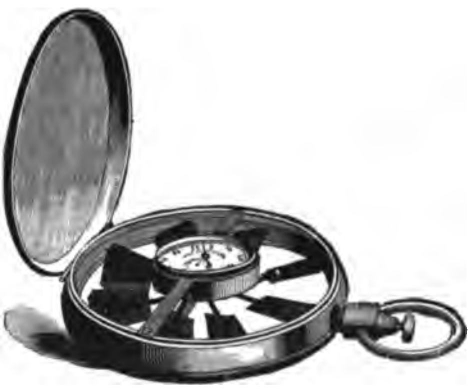


Fig. 397.—Casartelli's Watch Form of Biram's Anemometer.

To ascertain the velocity of the current with the Biram anemometer just described it is necessary to hold it out in the airway at right angles to the course of the flowing air and at arm's length, so as to be free from all obstruction. The back of the instrument must be held towards the current to be measured, and kept there long enough to note the number of revolutions made. Before holding it out in the current the readings of the index circles may be entered in a book and the disconnecting lever thrown out of gear. Two persons are generally required in taking the observations, one to hold a lamp and a watch with a second hand, the other to hold the instrument. After being held in the current until the vanes attain their full velocity, a signal is given by the timekeeper to the observer, who then moves the disconnector into gear; at the expiration of a stated time another signal is given, the disconnector is again thrown out of gear, and the reading taken. Some operators prefer, before making an observation, to let the air give motion to the vanes with the disconnector thrown into gear until the first pointer on the dial reaches its zero, after which a push of the lever with the forefinger throws it out of gear. By this action an even number of hundreds is entered in the book and it becomes easier to make the requisite deduction after the completion of the observation.

Sometimes a small tripod stand is used to which the anemometer is screwed, the tripod being planted in the middle of the roadway. This is a relief to the observer in case of prolonged investigations, but it is unnecessary, and objectionable owing to the increased weight, for ordinary observations which occupy about a minute each. Besides, the air-flow being more rapid in the centre of the airway than at the sides, ordinary observations should be made by moving the instrument quietly backwards and forwards across the airway at different levels, beginning at the top and gradually descending during an observation, or else a number of observations should be taken with the instrument in different positions and the total readings afterwards averaged.

By the 49th section of the Coal Mines Regulation Act, 1887, the quantities of air in the respective splits must be measured at least once a month, and the result entered in a book kept for the purpose at the mine.

The Special Rules often state that the air currents must be measured weekly, and it is the custom at some collieries to measure them twice a week, the results being entered in a tabulated book, with columns for the area of airway, the position of the observation, the duration of the experiment, the velocity of the current, the volume passing in the intakes, the volume passing in the returns, the height of the barometer, the temperature, the water-gauge measurement—the last three being for readings at the shaft at the time of observation—and a column for general remarks.

It will be generally found on completing the whole of the air measurements, that the sum total of the return quantities is considerably in excess of the sum total of the intake splits. This is due partly to the expansion of the air and partly to the emission of gas at the working faces, which, if the coal be very gaseous, considerably augments the volume of air and gas in the returns. A further increase arises from gases given off from the chemical actions which are always taking place more or less in every mine.

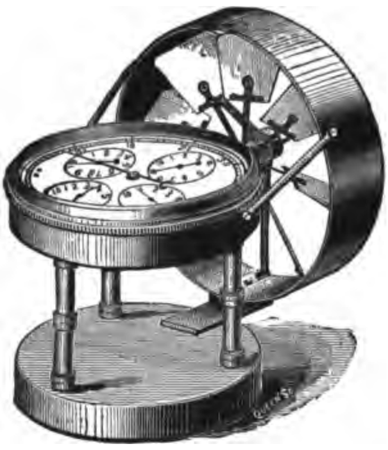


Fig. 398.—Casartelli's Air Meter.

Casartelli's Air Meter.—This is another form of anemometer, especially constructed for indicating the velocity of the current of air in mines, sewers, &c. Fig. 398 is an illustration of the instrument from which it will be seen that there are five dials reading up to 10,000,000 feet. These are arranged horizontally on a stand which admits of the fan being placed vertically on one side with its axis extending through the circumference of metal which encloses the wheel-work and so operating the registering apparatus within. Different sizes of this air meter are made, having a smaller or greater number of indices, reading in one instrument up to 1,000 feet, and in another up to 10,000 feet. To measure the velocity of the air it may be placed with the stand on an even part of the floor with the fan facing the current, or be screwed to a rod carried by a short tripod and provided at the upper end with a shifting head by means of which it will slide up and down.

The Davis Patent Self-timing anemometer shown in Fig. 399 dispenses with the use of a watch or timer, and one observer is sufficient to measure the air. By holding this instrument in the current of air to be measured for a few minutes, it correctly indicates the velocity in feet per second and not the number of revolutions of the vanes. To make an observation with it the instrument is held out at arm's length, and on the vanes acquiring the full speed due to the air-current, the spring button at A is depressed and the large hand then indicates feet per second. The pointer does not at once return to zero on releasing the button, but remains at its registering point on the circle by a locking arrangement. After the reading has been noted, the milled head at B is screwed down until the plunger A is released, the milled head is unscrewed as far as it will go, and the index-hand returns to zero. The instrument is then ready to take another observation. If the velocity is such as to cause the large pointer to travel more than one revolution, the inner circle of figures is read. The small hand shows whether the outer or inner circle should be read. Each instrument is graduated by actual experiment, so that no allowance for friction has to be made in this instrument.

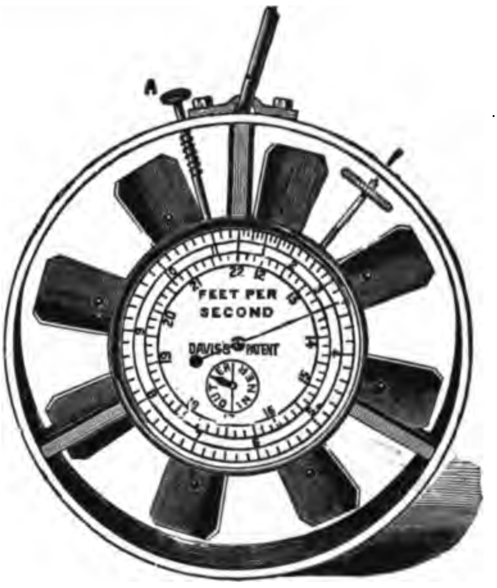


Fig. 399.—The Davis Self-timing Anemometer.

The *Dickinson* anemometer is another which requires no timing. Its simplicity in construction and the readiness with which it may be read render it suitable for the use of subordinate officials, but the oscillations of the flap render accurate readings impossible. It is made by Mr. Casartelli to indicate a velocity of from 250 to 800 feet per minute, as shown in Fig. 400. It consists of a flap made of mica or tala hinged at the top and exactly balanced by the lever and weight attached near the hinge and extending above the frame; when the instrument is

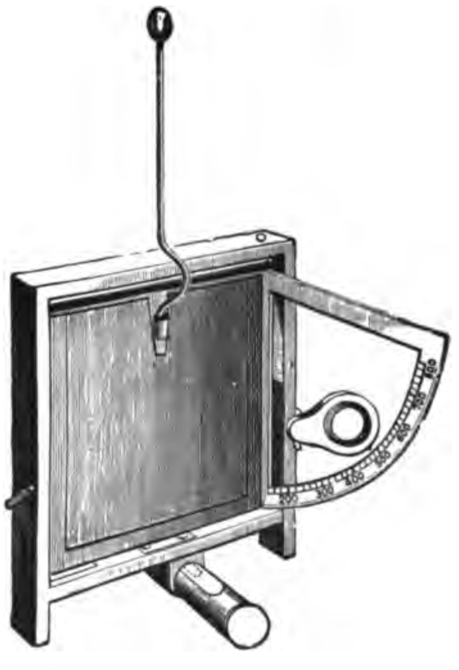


Fig. 400.—The Dickinson Anemometer.

not in use this flap closes in its frame. The spirit-level underneath is fixed at right angles to the frame in order that when an observation is to be made the instrument may be placed with the flap in an upright position until the force of the air current deflects it. The height to which the flap is lifted varies with the velocity of the air. The quadrant at the side is graduated so as to show the velocity of the air corresponding to the height of the flap during an observation, the scale engraved on it having been laid down by previous actual experiment on a special whirling machine. A handle is fixed to the side of the frame by which the anemometer may be conveniently held.

The anemometers so far described are incapable of indicating excessively high velocities of air without endangering their mechanism. The *Capell-Davis* anemometer shown in Figs. 401 and 402 is a more recent invention and is designed expressly to stand the highest known air velocity and the effects of sudden shocks. Fig. 401 is a front view showing the six dials, disconnector, and stick-holder, while Fig. 402 is a back view showing the vanes. It differs from anemometers on the Biram principle in the arrangement of the vanes. These are fitted rigidly to a solid disc, and are somewhat similar in construction to the blades on the Capell ventilating fan. The instrument is held with its back or fan side facing the current of air to be measured and so receives the wind pressure equally over its whole surface. As at first constructed, when exposed in extremely high velocities of air, the fan of the instrument "ran away," thereby preventing the proper action of the registering apparatus, in consequence of which a check has since been attached to the angle of the plates causing greater resistance as the velocity increases. The connection between the fan disc and the indices is similar to that of the Biram anemometer, and consists of a shaft which runs through the centre and connects through a train of wheels by pinions. It is intended to be used in velocities of 30 feet or more per second.

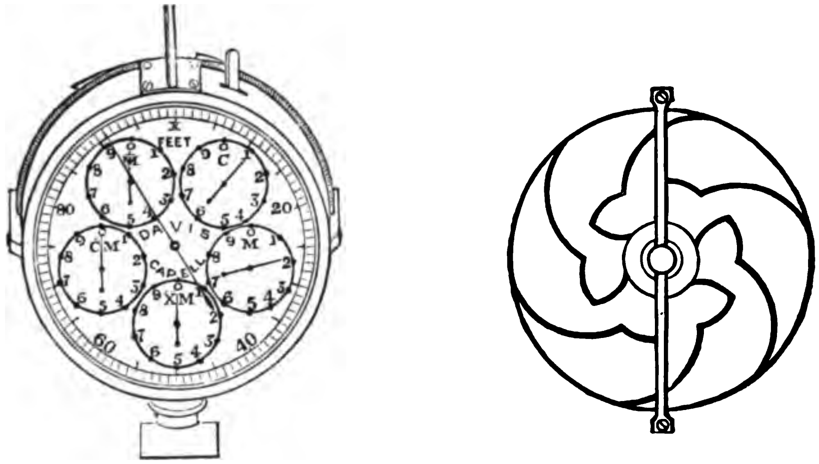


Fig. 401.—Front View, Showing Dials, Disconnector, and Stick-holder.

Fig. 402.—Back View, Showing the Vanes on the Capell Fan Principle.

Capell-Davis Anemometer.

Anemometers on the Biram principle require correction to bring the velocity recorded by their registering apparatus to the true velocity at which the air moves.

According to Messrs. Atkinson and Daglish's experiments the true velocity may be determined by the formula,

$$v = a\,\mathrm{R} \times b$$

where a is a constant proportional to the number of lineal feet travelled by the air per revolution, R is the number of revolutions registered by the anemometer, and b the losses of velocity due to friction of machine, the loss again being determined experimentally by a whirling machine. Maker's corrections, however, are not based on this formula, but are made by adding or deducting numbers which vary for the different instruments and for different velocities in the same instrument. The inaccuracy of each anemometer at different velocities is carefully ascertained by Atkinson and Daglish's whirling machine shown in Fig. 403, and a certificate is issued with each instrument showing the amount of air in feet to be added or deducted, being the result of experiment with each anemometer separately. The whirling machine is placed where there is no current of air. In it the frame shown in the side and end views carries a vertical axis attached to a horizontally revolving bar at its upper extremity and geared into toothed wheels near its lower extremity. A cord is secured to the barrel or drum, and then wound a few times round it, and afterwards carried over a suitable arrangement of pulleys to a weight which, in descending, pulls at the barrel, causing it to revolve. The shaft of the barrel is connected by bevelled toothwheels to the vertical axis, which turns with the barrel and causes the horizontal bar to revolve. The anemometer is placed on this bar, and being moved in a circle, say of 30 feet circumference at any speed by a regulated weight, the distance through which it is moved is recorded on the dial of the machine. The bar moving against motionless air causes the vanes to revolve and the distance recorded on the anemometer should agree with that of the whirling machine; if not, the difference is noted on a card, to which reference should be made whenever the anemometer is used in order to correct its registered readings.

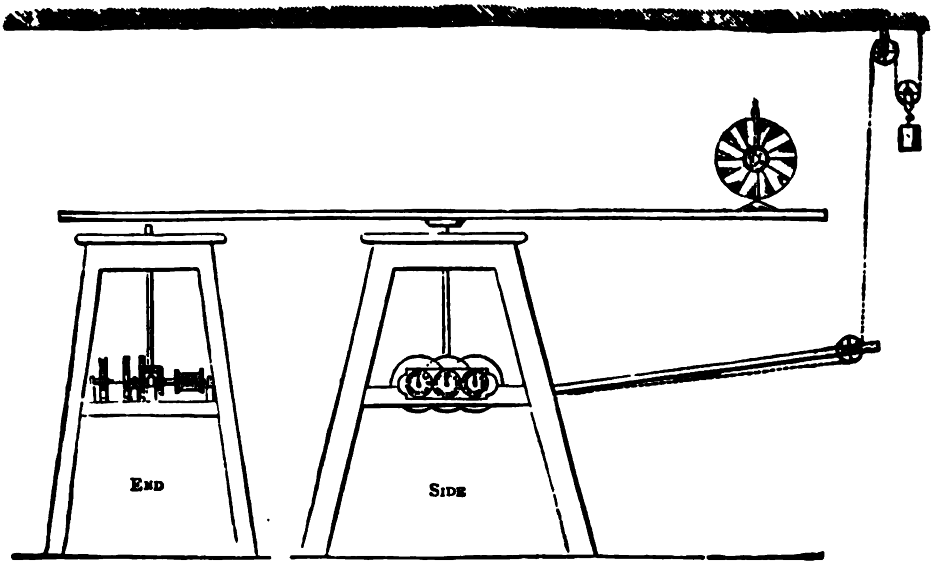


Fig. 403.—Anemometer Testing Machine.

To be sensitive, anemometers are delicately and lightly made, and even with careful usage are easily deranged. In order to be sure of accurate readings they should be tested from time to time. For that purpose they may, on payment of a fee, be verified at Kew Observatory.

The *Thermometer* is a measurer of temperature, mercury being used for ordinary temperatures. It depends for its action on the fact that all bodies with the rise and fall of their temperatures expand and contract. It consists of a glass tube closed at the top, with a bulb at its bottom end, and having mercury placed in it. A scale of degrees is fixed to the tube. As used in mines the thermometer registers the temperature of the air, and we are able to measure the difference of temperature between the air in the downcast and upcast shafts or at any desired point in the workings.

Thermometers are graduated according to three scales, viz., Fahrenheit's, which is that commonly used in England; the Centigrade scale, which is that generally used in the scientific world; and Reaumur's scale, which is that taking its name from a French philosopher, who constructed his thermometer with alcohol of such a strength that 1,000 parts at the freezing point of water became 1,080 parts at its boiling point.

On Fahrenheit's thermometer 32° indicates the freezing point and 212° the boiling point of water, and the space between these two fixed points is divided into 180 even divisions; these even divisions are produced above and below 32° and 212°.

In the Centigrade thermometer 0° indicates the freezing point and 100° the boiling point of water, the space between these two points being divided into 100 even divisions. In Reaumur's 0° indicates the freezing and 80° the boiling point of water, the space between these two points being divided into 80 even divisions. It is plain therefore that—

$$180^\circ \text{ Fah.} = 100 \text{ Cent.} = 80^\circ \text{ Reaum.}$$

and therefore $$1^\circ \text{ Fah.} = \tfrac{5}{9} \text{ Cent.} = \tfrac{4}{9} \text{ Reaum.}$$

To transfer Fahrenheit degrees to the other scales we must first subtract 32°, in order that the number of degrees from the freezing point may be ascertained. These multiplied by $\frac{5}{9}$ths will give the equivalent number of Centigrade, and by $\frac{4}{9}$ths the equivalent number of Reaumur degrees.

To reduce Centigrade and Reaumur degrees to the Fahrenheit scale, multiply by $\frac{9}{5}$ and $\frac{9}{4}$ respectively and add 32°.

If the temperature be below the zero in any of the scales, a minus sign (—) is placed before the number thus: —5° Fah. means 37° below freezing point.

The following examples may be tested:—

Fah.		Cent.		Reaum.
190°	=	$87{\cdot}\dot{7}^\circ$	=	$70{\cdot}\dot{2}^\circ$
155°	=	$68{\cdot}\dot{3}^\circ$	=	$54{\cdot}\dot{6}^\circ$
128°	=	$53{\cdot}\dot{3}^\circ$	=	$42{\cdot}\dot{6}^\circ$
3°	=	$-16{\cdot}\dot{1}^\circ$	=	$-12{\cdot}\dot{8}^\circ$
-15°	=	$-26{\cdot}\dot{1}^\circ$	=	$-20{\cdot}\dot{8}^\circ$
-40°	=	$-40{\cdot}^\circ$	=	$-32{\cdot}^\circ$

It must be borne in mind that a thermometer does not give the absolute expansion of the mercury, but the difference between the expansion of the mercury and that of the glass. Mercury expands about 7 times more than glass.

Mercury, or quicksilver, is a metal having the remarkable property of being fluid at ordinary temperatures. Mercury boils at 660° Fahr., giving off vapour of specific gravity 6·976, and it may therefore be distilled. It always forms one of the metals of an amalgam, and is much used in the process of amalgamation and decomposition in order to separate gold and silver from their ores. Mercury is well adapted for use in the construction of thermometers for a certain range of temperature, as it expands at a uniform rate, when heated, from considerably below 0° to above 300° Fahr. In colour, it is silvery white, with a striking metallic lustre. It congeals at $-40°$ Fahr., and in its solid state is malleable. Liquid mercury has the specific gravity of 13·59 at 60° Fahr. It contracts greatly when it solidifies, so that the density of frozen mercury is 14. Under ordinary circumstances mercury does not alter in the air, nor, if pure, will it tarnish. It gives off vapour at all temperatures. When pure it runs in small spherical drops over smooth surfaces; when impure the drops become elongated and leave a grey stain on porcelain or glass. By distillation it is easily rendered pure. Other recommendations for the use of mercury in the construction of thermometers are its low capacity for heat, in consequence of which its temperature soon changes, and the fact that it does not adhere to the glass.

The glass tube for a thermometer has a capillary or hair-like bore, and its uniformity is tested by running about half an inch of mercury down the tube and measuring it at different points in its descent. A uniform bore will show equal measurements of the descending mercury throughout its course along the tube. The fine glass tube is blown into a bulb at one end, and great care is necessary in the blowing. The capacity of the bulb is much greater than that of the tube, so that the expansion and contraction of the mercury in the former are rendered more perceptible in the latter. To fill the bulb, the tube is held in a slightly inclined position, a funnel of paper is fastened to the top of the tube into which is placed purified mercury. By means of a lamp the bulb is heated and the contained air expands and is driven upwards until it bubbles up through the mercury in the funnel. The lamp is then removed, the air in the tube cools and contracts, forcing some of the mercury down into the bulb. The inrushing mercury replaces the air which was driven out by the heat. The lamp is again applied to the bulb, and the process repeated a few times until the mercury reaches a convenient height in the tube when at the common temperature. The quantity of liquid introduced is generally little more than sufficient to fill the bulb at the lowest temperature which the thermometer is intended to indicate. The lamp-flame is again placed under the bulb; the heat causes the mercury column to rise to the top of the tube and ooze out; whilst so doing the tube is hermetically sealed by bringing a blowpipe flame to play upon the opening in the

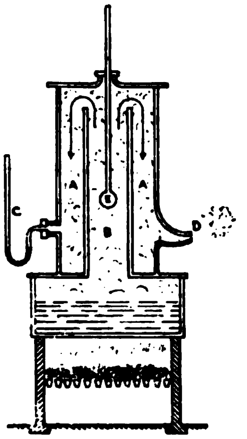


Fig. 404.—Method of Fixing the Boiling Point on a Thermometer.

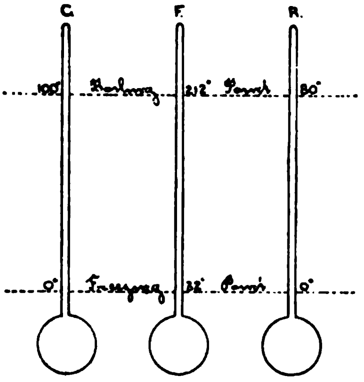


Fig. 405.—Graduating Thermometer Scales.

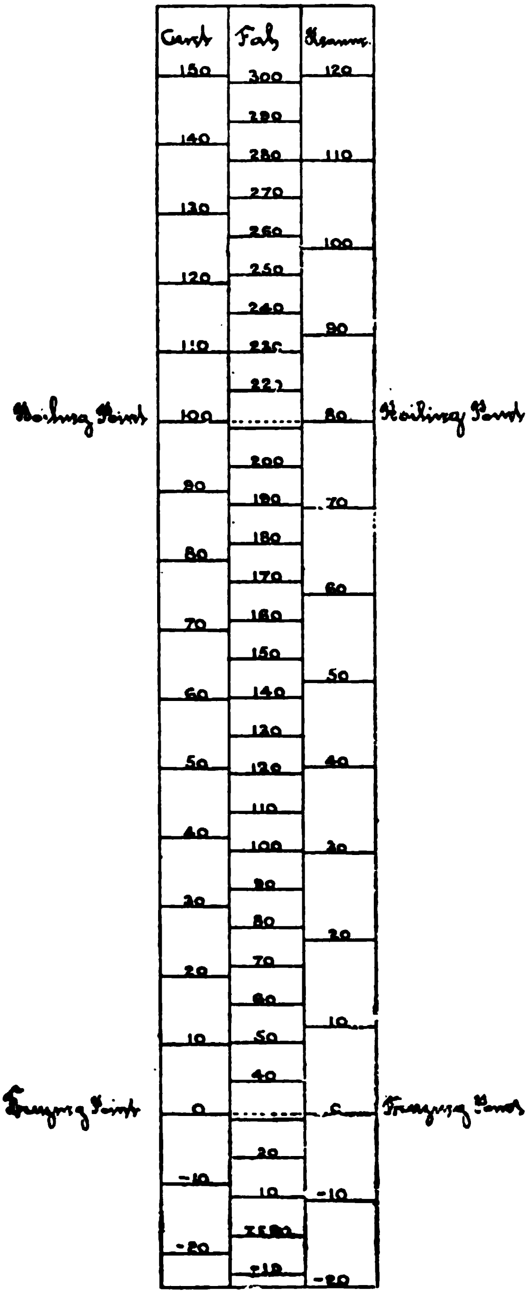


Fig. 406.—Comparative Scale of English and French Thermometers.

tube. The mercury on cooling leaves a vacuum, which is essential to the perfection of the instrument.

To graduate a thermometer it is immersed in melting ice and the point to which the mercury falls is marked off with a file. Ice invariably melts at a fixed temperature, although water does not always freeze at the same temperature. If water be kept perfectly quiet, and its temperature be gradually reduced, a few degrees below 32° Fahr. may be reached before crystallization begins. The file mark on the tube indicates the freezing point on the thermometer. The boiling point is always constant if the barometric pressure is also constant, and is fixed by placing the thermometer in a vessel in which distilled water is made to boil, see Fig. 404. The generated steam passes up the partition B, thus surrounding the thermometer placed at the top, after which it takes the direction of the arrows to an outer division in the same vessel: it then descends to the orifice D, where it escapes. The steam as it surrounds the thermometer is in this way unaffected by the temperature of the air outside the generator. C is a glass tube bent like the letter U, and having a little mercury placed in the bend. It guards against a break in the uniformity of the steam temperature. If the steam is throttled, at D, the pressure will increase and give evidence of it by driving the mercury up the tube C. The point which the mercury in the thermometer reaches when it has ceased to ascend without afterwards showing signs of fluctuation is marked off with a file as the boiling point. The thermometer is then attached to a board upon which the scale is marked into arbitrary divisions. If Centigrade, see Fig. 405, the space between the two fixed points is divided into 100 equal parts, which are produced above and below as far as required. For Fahrenheit the same space is divided into 180°, the freezing point being 32°. For Reaumur the division is into 80°, the freezing point being 0° as in the Centigrade.

The experiments are made when the barometer is at 29·95 inches. A mercury thermometer cannot be graduated lower than —40° on Fahrenheit's scale, because the mercury then congeals. It may be graduated upwards to 600° Fahr., but not higher, to be trustworthy, for mercury boils at 660° Fahr.

If the vacuum in the tube of a thermometer be good, on the instrument being inverted, the mercury strikes against the top of the tube with a clear ringing sound. If, after being in use some time, it is tested by placing it in melting ice, and the mercury then does not stand at the freezing point mark, the error is called "the displacement of zero." It arises from the curious fact that sometimes bulbs do not perfectly contract for two or three years after blowing. In the very best instruments the bulbs are kept empty for that length of time before being filled. A thick glass bulb is less likely to change than a thin one.

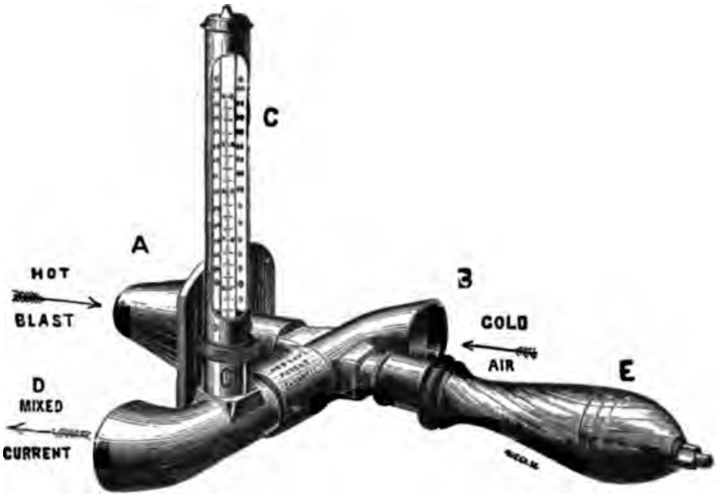


Fig. 407.—Hobson's Patent Hot Blast Pyrometer.

The three thermometer scales referred to are shown in diagram form in Fig. 406. It is drawn to scale, so that a rough comparison of the three sets of figures corresponding to a height of the mercury may be seen at a glance within certain limits.

As it is almost impossible to freeze the liquid, alcohol is used in the manufacture of thermometers to register very low temperatures. There is no limit to the downward graduation of such thermometers, but their upward range is much more limited than that of mercurial thermometers.

High temperatures are measured by pyrometers. These are instruments for measuring all gradations of temperature above those which can be directly indicated by the mercurial thermometer. Many pyrometers depend for their action upon the uniformity of the expansion and contraction of metal.

In Hobson's patent hot blast pyrometer, made by Mr. J. Casartelli of Manchester, mercury is used, and although this is incapable of directly indicating temperatures above 600° Fahr., it is made to do so by the temperature of the hot blast being first reduced to a known degree. Fig. 407 shows the instrument. By means of the handle E it is readily attached at the cone A to the sight-hole of the tuyere. The hot blast passes through the tube, and issues out of it at right angles facing the thermometer, and induces a current of cold air to enter at the opening B and mix with it: the size of the jet tube being much smaller than the bore of the body of the instrument. This diluted current, in passing to the outlet D, acts on the bulb of the thermometer, raising its temperature in proportion to the heat of the blast. The temperature of the mixed current is indicated on the glass stem, and the side scales are graduated to mark the actual heat of the hot blast, as ascertained by Siemens' copper-ball pyrometer. The outer scales are so arranged that the one marked in Fahr. degrees is placed with the 400 opposite the 152° Fahr. on the thermometer tube. The instrument should be examined occasionally to see that the tube has not slipped. The thermometer is shown in a vertical position in the illustration, but may be turned down at right angles in a line with the handle when it is not in use.

If the column of mercury in an ordinary thermometer becomes divided in carriage, the thermometer must be inverted, and the head tapped gently on a table until the mercury flows out of the bulb to the top of the tube: it must then be shaken from side to side until the mercury re-unites and returns to the bulb in one volume.

Solids expand too little, gases too much, to be practically serviceable in the construction of thermometers for ordinary temperatures, hence the use of liquid thermometers.

The precise form which these take depends entirely upon the purpose for which they are required. If they are to be kept steadily in one position, without agitation, to register the temperature of the air in the open, in a room, or at a colliery above or below ground near the downcast shaft, they may be arranged to accompany a barometer as shown in Figs. 415, 416 and 417, or alone in common boxwood form with a scale graduated from 0° to 120° or 130° Fahr. on one side of the tube, and a Centigrade scale to correspond on the other; such scales meet the requirements of range in this country.

For use in an upcast shaft a self-registering thermometer is made enclosed in a stout cylindrical copper case. It is usually graduated from 40° or 50° to 600° Fahr., the reading of a mercury thermometer being quite reliable to this point.

The case is 14½ inches long by 2 inches in diameter, and is open at both ends. A hinged door is provided to close over an opening cut throughout the length of the cylinder, and this door is kept securely fastened, except when access to the interior is desired. A metal scale 14 inches long is fixed internally in the centre of the cylinder. The scale is graduated sometimes from 50° to 550° Fahr., and sometimes from 40° to 600° Fahr., the divisions being cut every 2° Fahr. The glass tube containing the mercury is 12¾ inches in length by ¼ of an inch in external diameter, and is inserted in the centre of the scale by means of small brackets within the case. Immediately above the bulb the bore of the tube is reduced in size, and then gradually increases till it attains its full diameter, which it afterwards maintains throughout the tube. The diameter of the bore nowhere exceeds that of an ordinary sized sewing needle. At the point where the full sized bore begins the mercury is broken into two detached portions of column by the intervention of an air speck which is about ⅛th of an inch in height when the space separating the mercury is at its minimum. When the instrument is to be used, a shake is given it by tapping the bottom of the copper cylinder against the palm of the hand, or otherwise so as not to cause injury, and yet ensure the falling of the top portion of mercury into its normal position close to the lower column. The registered temperature recorded by it, which may be that of the surface or of some point below ground, is then read and the result entered in a book, or if below 40° Fahr., another thermometer is used to ascertain this. The upcast thermometer is then lowered down the shaft by means of a thin wire capable of standing such heat as it will be subjected to. If a furnace shaft, the temperature at the bottom is very great. Usually the thermometer is lowered to a point half way down the shaft, that being considered to give the average temperature of the upcast shaft.

After reaching the point at which the observation is to be made the thermometer is allowed to remain quietly there for about 15 minutes to ensure the column of mercury rising to the point due to the temperature of that position, after which it is gently withdrawn to the surface. Here, if necessary, it is allowed to cool before being examined and read. If proper care is taken to avoid jerks and rough usage the upper detached portion of mercurial column remains in the position it reached when at its highest temperature, even after cooling down. It is then read, the reading being entered in the book by the side of the surface or the previously taken temperature, so that a comparison can at once be made.

Special thermometers are designed for deep-sea experiment and for many other purposes.

In 1782 Mr. J. Sixe, of Canterbury, invented his self-registering thermometer which is still known by his name. The object of a self-registering thermometer is to show the highest or lowest temperature, as the case may be, which has occurred during the absence of the observer. Sixe's registering thermometer for maximum and minimum temperatures is shown in Fig. 408. It consists of two vertical exterior tubes connected at the bottom, and one of them being bent over and brought down to form a bulb between the outer tubes. A rise of temperature causes the spirit placed in the middle tube or bulb to expand and drive downwards the mercury in the left-hand tube and upwards in the right-hand tube.

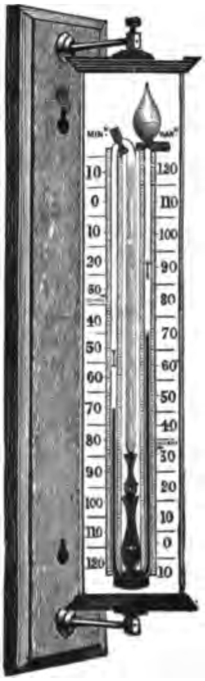


Fig. 408.—Sixe's Self-Registering Thermometer for Maximum and Minimum Temperatures.

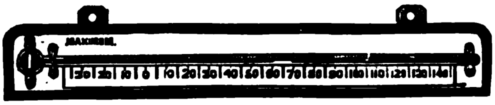


Fig. 409.—Maximum Thermometer.

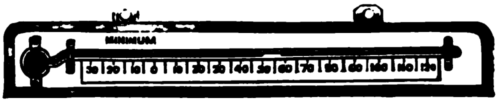


Fig. 410.—Minimum Thermometer.

A suitable scale is graduated downwards from the top of the left-hand tube and upwards from the bottom of the right-hand tube, so that the indicated readings are the same in the two tubes. An index, movable by the mercury in an upward direction, is placed in each of the exterior tubes. The mobile spirit presses that in the left-hand tube without disturbing it, but the dense mercury rising in the right-hand tube carries the index on its surface to the highest level it reaches, and as it remains there when the mercury descends it marks the maximum temperature. Upon cooling, the spirit contained in the bulb contracts and acts like a spring to force the mercury in the other direction, pushing up the index in the left hand tube, which remains at the highest level it is carried to, and so marks the lowest or minimum temperature. Sixe's thermometer is open to several disadvantages, particularly to liability of the mercury to oxidize, notwithstanding improvements which have been made in it since its invention.

Figs. 409 and 410 show respectively standard maximum and minimum thermometers which are verified at Kew, and which have certificates supplied with them by Messrs. John Davis & Son, Derby. The maximum thermometer usually employed is a mercurial one on the improved plan, invented by Professor Phillips. In it a portion of the mercurial column is separated by a small air speck which separates a portion of the column and acts as an index. In the minimum thermometer, the tube is usually filled with spirit, and has a steel index floating in it. As the spirit contracts the index is carried with it, but when the temperature rises again, the spirit passes it, by which means it is left to mark the lowest temperature reached. The thermometers with steel indices are set by inclining the instrument so that the indices may go to the top of the column, or in case of those on Sixe's principle, by means of a small magnet.

Maximum and minimum thermometers are read off and set at regular frequent intervals throughout the day, the readings, usually hourly, being notified in a suitably arranged book. A mean daily temperature from the readings may then be obtained at the close of the day.

Messrs. Negretti and Zambra make an inverted maximum thermometer for the determination of underground temperatures. The thermometers are inserted in boreholes, and register the temperature of any stratum without being affected by the after-cooling consequent on withdrawal from the boring.

The *Hygrometer* is an instrument used for the purpose of ascertaining the humidity of air or of any gas. Much ingenuity is shown in its various forms.

Fig. 411 represents Daniell's instrument. In it a glass tube is bent in two places at right angles, and terminates with a bulb blown at each end. The tube is about seven inches long, and is bent into three arms of unequal length. In the bulb A is a delicate thermometer. This bulb A is then about two-thirds filled with ether, a spirit-lamp is placed under it and kept burning until the vapour of the ether has expelled the air from within through a capillary tube, which is left open for the purpose, and afterwards hermetically sealed. A muslin covering is then placed over B, after which the instrument is placed on a stand, which carries a small thermometer, by means of which the temperature of the air is ascertained. When it is required to make an observation a small portion of ether is dropped on the muslin. This ether rapidly evaporates, and lowers the temperature of B, thereby causing a rapid condensation of the ether vapour contained in the tube. As the condensation takes place in B, evaporation from the surface of the ether in A continues, until the temperature of the ether becomes so reduced, as to cause a deposit of moisture from the air on the polished surface of the glass. At the instant the exterior of the glass is dimmed an observation is taken of the inner thermometer, which must always indicate the temperature of the ether, because its bulb dips into it, and thus the dew-point is ascertained, *i.e.*, the temperature at which the atmosphere begins to precipitate moisture. The thermometer in the stand is read at the same time, and this gives the temperature of the air.

The quantity of ether consumed, its costliness and difficulties of transit, and application in making observations, especially in hot climates, are objections to the use of this instrument.

A more convenient form of hygrometer is now mostly used, consisting of two thermometers. These are placed side by side, as shown in Fig. 412. The bulb of one is covered with muslin, which is kept wet by capillary attraction, a small vessel of rain-water being placed near, into which the muslin, or an absorbent wick connected with it, dips. As evaporation proceeds from the muslin, the bulb is cooled. The evaporation will increase in proportion to the dryness of the air, and the reading indicated on the scale of the wet-bulb thermometer will then be considerably lower than that on the dry. The difference between these readings will be less and less the more humidity there is in the air, and will become zero when the air is completely saturated. Hence the difference of the readings on the scales of the two thermometers is a measure of the dryness of the air. From published tables, it may be ascertained from the two thermometer indications, the relative humidity, 100° being saturation.

Watery vapour is never altogether absent from the air we breathe, and on the other hand, the air is very rarely saturated. The amount of watery vapour which can be contained in a given space of air, depends chiefly on the temperature. Directly the temperature of saturated air is raised, it becomes capable of holding more watery vapour, and is therefore no longer at the point of saturation. If on the other hand its temperature be lowered, it can no longer retain all its vapour in solution, and a portion passes from the gaseous to the liquid form, and will be deposited as dew or rain. After the sun has set, many bodies which were warmed by its rays, radiate their heat until they become cooler than the air around them. A clear and calm night, free from wind, is favourable to rapid radiation; moisture condenses on the ground, the condensation being most abundant on those bodies which radiated their heat most. Dew is thus formed by the cooling of the air in contact with the ground : rain is produced by the cooling of a mass of air below its dew-point.

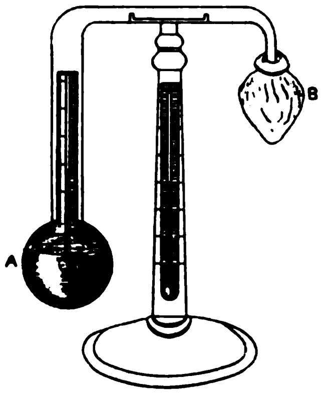


Fig. 411.—Daniell's Hygrometer.

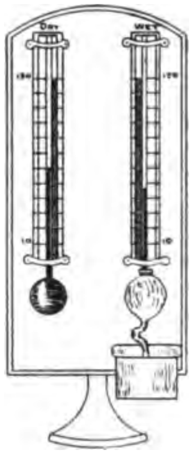


Fig. 412.—Hygrometer in Ordinary Use.

The temperature of air as it descends a mine may be raised or lowered, according to the season of the year, and the depth of the shaft. Workings of moderate depth will be warmer in winter and cooler in summer than the air at the surface, the temperature of the flowing air being closely assimilated to that of the galleries soon after leaving the shaft. As the air travels onward, its temperature is raised by workmen, horses, and lights in the mine. The exact state of the air is obtained at different points of the mine by means of the hygrometer. The Royal Commissioners on Coal in the United Kingdom, give several observations of this kind in their Report of 1871.

The following, taken from their Summary for the County of Durham, may be of interest as showing variations in the hygrometrical condition of the air at different collieries, having shafts of varying depth.

SUMMARY OF HYGROMETRIC OBSERVATIONS IN COAL MINES IN THE COUNTY OF DURHAM, ALL OBSERVATIONS HAVING BEEN MADE IN A WORKING FACE.

Name of Mine.	Depth in Feet.	Distance from Down-cast Shaft in Yards.	Dry Bulb.	Wet Bulb.	Relative Humidity, 100° being Satura-tion.	Remarks.
			°	°		
Jane Pit, Eppleton Colliery .	1,395	4,332	73·5	73·5	100·	
	1,395	4,440	74·	74·	100·	
	1,395	4,560	74·5	74·5	100·	
Caroline Pit, Eppleton Colliery	838	2,560	64·	64·	100·	
	1,040	3,364	65·5	65·	97·2	
	1,012	3,328	65·5	65·	97·2	
	1,030	3,365	65·	65·	100·	
LadyPit,Elemore Colliery. .	970	1,866	67·5	67·5	100·	
	930	2,660	66·	66·	100·	
Wharton Colliery	88	3,454	65·	64·5	97·2	
	900	4,246	64·5	64·	97·2	
	924	3,696	68·	67·	94·7	
	1,254	1,826	71·75	70·	91·	
Monkwearmouth Colliery . .	1,646	3,256	81·25	79·5	92·	
	1,640	3,216	82·25	81·25	95·6	
Ryhope Colliery	1,560	2,762	73·	71·	90·2	Under the sea.
Murton Colliery	1,374	4,532	70·	69·5	97·4	
Monkwearmouth, 2nd observations . .	1,646	3,256	81·	78·	86·8	These observations were taken by Mr. L. Wood and Mr. Dickinson conjointly.
	1,640	3,216	81·	78·	86·8	
Monkwearmouth Colliery, 3rd observations .	1,646	3,256	81·	74·	70·4	It is the practice in this colliery to water the roads to keep down the dust, but this practice had been suspended for 11 days immediately preceding this 3rd set of observations.
	1,640	3,216	81·5	77·	80·6	
Seaham Colliery .	1,995	2,200	78·	67·	60·3	

Watering the roads affects the air as seen by comparing the different observations at the Monkwearmouth Colliery with each other.

The *Barometer* is an instrument used for measuring the pressure of the air. If a glass tube a yard long and closed at one end be filled with mercury and inverted with the finger placed over the open end until that end be placed in a vessel containing mercury and then removed, a part of the mercury will run out, but the tube remains filled to a height of about 30 inches above the surface of the mercury in the vessel. That is, the ordinary pressure of the atmosphere is sufficient to balance a column of mercury 30 inches high. But the pressure of the atmosphere varies in this country between 28 and 31 inches of mercury. The ordinary barometer has a scale and a sliding vernier fixed to it, by means of which it may be read to the $\frac{1}{100}$th part of an inch. In a former part it was stated that the average atmospheric pressure was sufficient to balance a column of water nearly 34 feet high, and as mercury is 13·59 times heavier than water, it is consistent that it should balance a column of mercury 30 inches high, for $\frac{34 \times 12}{13 \cdot 59} = 30$ inches. The barometer is of service in showing atmospheric changes. The issue of fire-damp from the goaves and working faces of the mines is checked when the barometer is high, and liberated rather more freely when the barometer is low. A consideration of the difference of the quantity of gas yielded under the two extremes of the barometer is not so important, however, as the result following from a falling barometer, because the escape of fire-damp is facilitated when, after the barometer has stood steadily at a high or moderately high reading for some days, it is succeeded by a rapid and sudden fall. The gas under those circumstances issues more freely from the coal, and also owing to the reduced pressure it finds its way out of the goaves into the roads; it is important therefore to exercise increased vigilance in fiery mines during a falling barometer, whether it stood high or not before it began to fall. Numerous cases are recorded where the airways of mines were found to contain large quantities of fire-damp with a steady barometer of long continuance, though the airways were a short time previously clear of the gas at the same indication of the barometer. This has been followed by a falling barometer, proving that the fire-damp of our mines is more sensitive to atmospheric changes than the barometer, as indeed might have been expected—gas being such an attenuated, mobile and highly-elastic fluid. The lesson to be learned from this is, that while we duly appreciate the barometer and go to it continually for its readings, we do not rely solely on it, but continue our examination with extreme caution to the innermost recesses of the mine, ever on the alert for every change.

Fig. 413 shows a cistern barometer which is a common form of the instrument. In its construction the mercury must be absolutely pure; if mixed with zinc or some other metal which makes the mixture lighter, the column will in consequence stand too high. The space left in the tube above the mercurial column is called the Torricellian vacuum, and as the accuracy of the instrument depends to a great extent on the near approach to perfection of this vacuum, great care is necessary in filling the tube to prevent the entry of air with the mercury. If absorbed in the mercury, or if the coating of air adhering to the interior of the tube be allowed to creep up inside, it afterwards acts in opposition to the pressure of the air on the mercury in the cistern, thereby rendering the reading less than it should be. Similarly, if there be moisture in the presumed vacuum, an outward application of heat, such as that of the hand to the tube, will cause the mercury to sink. The usual way of filling the tube is to pour in a small quantity of mercury, so as to fill only a few inches of the tube, and then boil it. This drives out the air and evaporates moisture; on cooling another small quantity of mercury is introduced, and boiled and so on with successive small quantities until the tube is filled. Unless the glass is of excellent quality the heat it is subjected to in this process renders it liable to crack. Other plans are adopted for ensuring the exclusion of the air, the method being determined by the accuracy of observation for which the instrument is intended. It is considered a sufficient test of a good vacuum when, by shaking the tube, the mercury strikes against the closed end with a hard, clear, and instantaneous tap. An absolutely perfect vacuum cannot be obtained, for a small quantity of air will remain, and the mercury itself will rise in vapour into the so-called vacuum. This vapour, however, is not sufficient to cause any perceptible pressure, and is not more than would rise from the mercury in the air. Globules of mercury may often be found condensed on the tubes of barometers.

After the tube for a cistern barometer is filled it is made to dip into a vessel of mercury A, Fig. 413. The accurately graduated scale E F may be engraved on the tube itself, but more frequently is fixed on the case containing it. It is customary in this country, where the variations in height of the column fluctuate between such extremes, to graduate the scale from 27 or 28 inches to 31. A sliding vernier shown in the Fig. is used to read the proportional parts of an inch when the vernier is adjusted level with the top of the mercury column. A mountain barometer has the graduations continued for a considerable distance below the 27 inches, in order that it may be used to ascertain the height of mountains. If a barometer reads 31 inches at sea-level, it indicates less than 18 inches at an altitude of 15,000 feet. A barometer intended for use in shafts sunk to a great depth below sea-level must have the graduations extended a few inches above 31.

The column of mercury in the tube is balanced by the weight of the atmosphere, and its height therefore varies in accordance with the state of the atmosphere, so that if the instrument be graduated with the mercury at a certain level in the cistern, on a change taking place in the atmospheric pressure the mercury in the column is forced up or falls in the tube, and these movements are accompanied by a disturbance in the level of the mercury in the cistern. The exact difference in height between the surfaces of mercury after such disturbance differs from that indicated by an ordinary fixed scale, because the zero mark of the scale is not then at the same height as the mercury in the cistern. If the area of the cistern be very large compared with that of the tube the error in indicated readings is small, and in some barometers for ordinary purposes the inaccuracies, being insignificant, are altogether neglected. Usually, however, some provision is made to remove such inaccuracies. This is effected sometimes by making the inches on the scale of the barometer fractional parts instead of standard measurement inches, and so neutralising the errors. Another plan is to make the scale movable, motion being given to it by a rack and pinion; by this means its lower end can be adjusted to the level of the mercury in the cistern. The best plan is shown in Fig. 413. Here the cistern is provided with a second bottom, which can be raised or lowered by means of the screw B. A pointer, C, is fixed to the side of the cistern, which is partly of glass, so that its lowest point terminates at the height the graduations are measured from. When using the instrument the screw B is turned until the point appears to meet its own reflection in the mercury. After making this adjustment the true height is shown on the scale. A standard barometer of extreme sensitiveness is made on Fortin's plan, which prevents leakage of the mercury in whatever position the instrument may be placed, or however it may be jerked; it is furnished with a certificate after being tested at Kew Observatory, to which place it is sent for verification, and has a glass cistern, an ivory point for adjustment, scales engraved on metal tube, verniers to read to $\frac{1}{500}$th of an inch, and a tube whose bore is 0·6 or 0·5 inch. It is mounted on brackets to a mahogany board, and has a thermometer of extreme sensitiveness, as shown in Fig. 414.

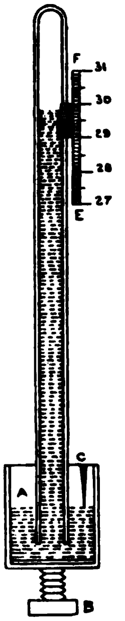


Fig. 413.—The Cistern Barometer.

The mercury in the tube of a barometer does not usually present an even surface owing to its tendency to cling to the sides. When it is rising, the surface is convex, when falling it is concave, so that a rising or falling cistern barometer may be judged by the appearance of the mercury in the tube. In taking the readings the vernier should be adjusted midway between the highest and lowest part of the uneven surface, but the tube should be tapped lightly with the fingers just before the instrument is read.

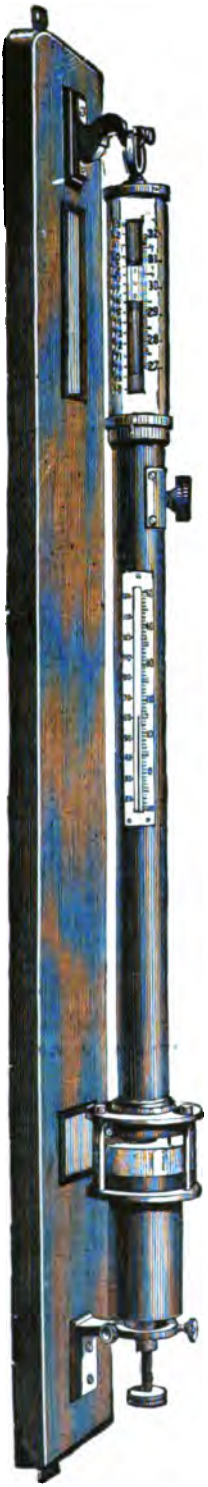


Fig. 414.—Standard Barometer.

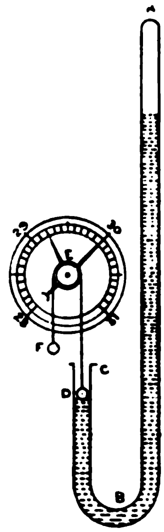


Fig. 415.—The Wheel Barometer.

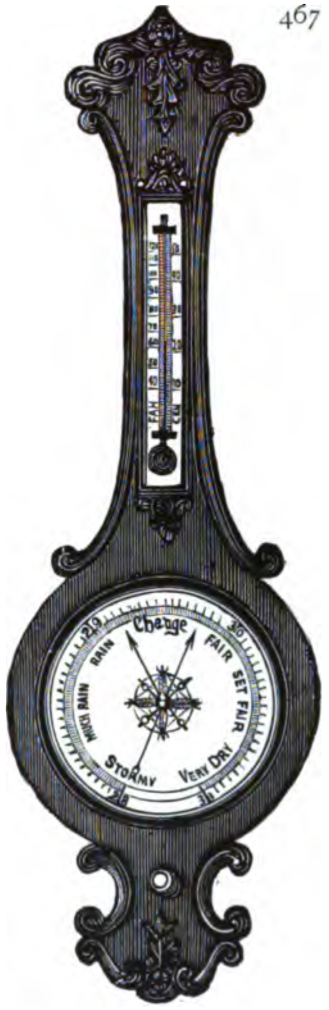


Fig. 416.—The Wheel Barometer.

Fig. 415 shows the wheel form of barometer out of its casing, and Fig. 416 shows its appearance when encased and with the usually accompanying thermometer. In it a large dial-plate is fixed near the lower end of a suitably shaped ornamental case. The circle has graduations from 28 to 31 inches, and also the words Stormy, Much Rain, Rain, Change, Fair, Set-fair, and Very Dry. There is really no useful object in placing these words to correspond with certain heights of the mercury, because the actual height of the barometer, apart from its fluctuations, is no guide in foretelling the weather. The same instrument which indicates "fair" at the sea-level may be carried to an elevation close to, where it then points to "rain" or "stormy." Rapid movement of the barometer, whether rising or falling, shows an uncertain condition of weather, and consequent liability to sudden changes. A steady rise after southerly winds shows an improving tendency, if the wind is from the west or north-west. A steady fall after northerly winds shows a bad tendency, and wind probably shifting southwards. The index-hand of the wheel barometer is caused to move round the face as its axle is acted on by changes in the mercury column. The tube, instead of dipping into a cistern, is bent and turned up 6 or 7 inches as shown in Fig. 415. A float D rests upon the mercury in the short limb. A cord is passed over the wheel E, and is attached at one end to the float D, while a counterpoise F is secured at the other. The hand on the dial-plate is attached to the wheel E and responds to movements of the mercury. If the mercury falls in the limb A B, it rises to an equal extent in the limb B C. The float D is then raised, and the weight F, as it descends, turns the hand into such position on the dial-plate as shows the height of the column. If the mercury is forced up the tube A B, it falls correspondingly in the limb C B, and the weight D is sufficient to overcome that of F, thus again giving motion to the hand, which accordingly indicates the altered height. In this form of barometer the tube is enclosed in a case which hides from view the tube, so that unless the case be opened, a rising or falling barometer cannot be judged from the appearance of the surface of the mercury. An additional hand, therefore, is provided, worked by a small handle below. By turning this handle the additional index-pointer is moved round the dial-plate into any desired position, where it remains stationary until an observer again moves the handle, and it is thus used periodically to mark the height of the mercury. This marker is made unequal in length and different in colour from the pointer moved by the mercury ; each pointer is thus easily distinguished. A glance at any time shows whether the mercury has risen or fallen since the marking-hand was last set to correspond with that subject to movement from atmospheric disturbance, and the amount, if any, is measured by the divergence of the two pointers.

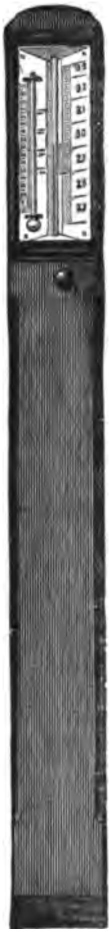


Fig. 417.—Colliery Barometer and Thermometer.

Barometers are sometimes made with an enlargement in the upper part of the tube near the closed end, and have a stop-cock placed just under the enlargement. On the tube being inclined the enlarged limb becomes filled to the top and the mercury is retained by turning the tap. The instrument may then be safely carried about from place to place, as the tube being full is not affected by the vibration. On turning the tap, with the tube placed vertically, as it should always be when in use, the pressure will be shown.

Fig. 417 shows the usual form of colliery barometer, in a substantial oak frame, glass covered face, with thermometer. The scale is graduated from 27 to 33 inches, to be suitable for shafts of ordinary depth. It is fitted with portable tubes and safety travelling screws for safe carriage, and is supplied by Messrs. John Davis & Son, Derby. The barometer and thermometer shown in Fig. 417 meet the requirements of the Mines Act.

The use of mercury is avoided in some forms of barometer, notably the aneroid. In this no liquid is used, but the atmospheric pressure acting on a vacuum chamber pulls at a spring which moves a pointer round a graduated dial-plate by means of a series of levers. The scale on the aneroid barometer is divided from a certain number of points obtained from observations of the mercurial barometer placed simultaneously with the aneroid under an air-pump. It requires occasional adjustment by a standard barometer. If made with great care, it is a reliable instrument, and is very convenient in use for some purposes, but a standard mercurial is the most accurate form of barometer made.

A very delicate glycerine barometer was designed by Mr. James B. Jordan. In this the lighter fluid used rises to a much greater height in the tube than in the mercurial barometer. This renders slighter changes in the atmospheric pressure perceptible. A serious objection to the instrument, however, is its great length, which makes it unfit for general use. In course of time, too, the vacuum becomes impaired owing to infiltration of the air through the glycerine.

In Bartrum's open-scale barometer the tube is between 4 and 5 feet long, and terminates in a cistern. The lower half contains mercury, and the upper a light fluid which forms the manometric index. The tube is made with a bulb about 3 inches long in the middle where the two liquids meet. A rise of mercury in the bulb causes a much greater rise of the lighter fluid in the narrower upper tube, the amount depending on the sectional area of the bulb as compared with that of the upper tube. The diameters of the bulb and cistern are each 1·5 inch, and that of the upper tube is 0·02 inch. With these sizes an increase of atmospheric pressure of 1 inch, as shown by a mercurial barometer, will appear on the scale as 8·2 inches, that is, the scale is rather more than eight times as large as that of an ordinary barometer. Slight alterations in the pressure are therefore easily discernible. The methyl salicylate, which is the liquid in the upper part of the tube, has a specific gravity about $\frac{1}{12}$th that of mercury and a very small vapour tension at ordinary temperatures. Great importance is attached to this, because on it depends the maintenance of a good vacuum and the trustworthiness of the instrument. The maker of this, Mr. J. J. Hicks, of Hatton Garden, London, E.C., claims for this great sensitiveness and accuracy of reading to $\frac{1}{300}$th of an inch. By varying the dimensions it can be made with as open a scale as a glycerine barometer, which is five or six times as long. It is said that Bartrum's instrument does not become impaired in the vacuum through the passage of air into it, or from vapour given off in the upper tube.

If it is necessary to compare the readings of barometers at different places two corrections must be made. The first is for the height above the level of the sea at half tide, such level being taken as giving the standard height. Directly the barometer is taken to a higher level the pressure is less, because the lower strata of air are not resting on it.

The fall of mercury in the column due to ascent amounts to about 1 inch for every 900 feet of elevation, and must be added to observed readings at that level or proportionally for other altitudes before such readings can be compared with observations taken at sea-level. As, however, the air becomes rarefied in ascending, 1 inch of mercury will not invariably balance the same height of air column, and the 900 feet cannot be applicable to great heights.

The second correction is for temperature. Mercury expands when exposed to a higher temperature, and contracts in a lower. An increase of temperature causes the mercury column to lengthen, the elongated column being afterwards sustained by the same pressure of air. It is true that the scale itself on which the heights are measured also expands or contracts, but the material used for scales never expands to the same extent that mercury does: if the scale and the mercury expanded and contracted equally, there would be no occasion for this second correction. The difference of readings due to temperature depends upon the difference in the two expansions, and may be taken at about ·00014 of an inch for every degree above freezing point or 32° F., and this must be deducted from the observed reading. A good barometer is always fitted with a thermometer in order that this second correction may be made.

The expansion of the barometer tube itself needs no correction, because the height of the column is unaltered by the change in the length of the tube. For meteorological observations a quantity should be added to the readings equal to the resistance of the tube in capillary action to the rise of the mercury. The capillary attraction is greater in an unboiled tube than in one in which the mercury is boiled. A greater correction is required for small than for large bores. Altitude measurements made with a single barometer, or with two barometers having tubes with bores of equal size, do not require capillary correction, because the resistance is equal in all parts of the tube.

As an illustration of the foregoing methods of correction, suppose we wish to compare the readings of two barometers, that at A being 450 feet, and that at B 80 feet above sea-level. The barometer at A reads 29·5 inches, and the one at B indicates 29·9 inches, whilst the temperature of the former place is 50° F., and at the latter it is 60° F. The corrections for altitude at A will be $\frac{450}{900}$ = ·5, and for B $\frac{80}{900}$ = ·089. The readings corrected for altitude only then will be A = 29·5 + ·5 = 30 inches, and for B 29·9 + ·089 = 29·989 inches. The correction for temperature at A is (50 — 32) × 29·5 × ·00014 = ·074, and for B (60 — 32) × 29·9 × ·00014 = ·117. Then 30· — ·074 = 29·926 inches as the reading for A fully corrected, and 29·989 – ·117 = 29·872 inches as that of B. Hence the difference in the atmospheric pressure at the two places when reduced for comparison is 29·926 – 29·872 = ·054 inch.

Beside the irregular fluctuations of the barometer, there is a regular daily variation arising from tides in the atmosphere which are apparently produced by the heat of the sun, hence their regularity of appearance and uniformity of height. This is known as the diurnal barometric wave, and its period varies according to the season. An examination of a continued series of hourly observations shows that the daily maximum height is reached in January at 10 o'clock A.M., and again at 9 o'clock P.M., changing slightly until in June the daily maximum heights are reached at 9 o'clock A.M. and 11 o'clock P.M. The minimum daily height is to be seen in January at 5 o'clock A.M. and again at 3 o'clock P.M., and in June at 3 o'clock A.M. and again at 5 o'clock P.M. In the tropics there is but little variation of period, the average daily maximum being reached at 9·30 A.M. and 10 P.M., while the minimum is reached at 3·30 A.M. and again at 4 P.M. The intensity of the wave varies with the latitude. At the level of the sea the average intensity is from ·09 to ·12 inch in tropical regions, diminishing gradually to 0 about 65° or 70° north latitude. The fluctuation in this country is only about ·05 of an inch, and being unimportant is disregarded. The intensity, too, decreases with the elevation above the level of the sea.

Temperate climates are subject to continual changes in the force and direction of the wind and in the amount of moisture in the air. Tropical regions are not subject to such variations, the barometric fluctuations recurring with great regularity. Generally these have a very limited range, being confined to ·2 of an inch. Occasionally, however, there are great sudden falls in the barometer giving warning of an approaching storm or hurricane.

As an instance of such a remarkable fall in the barometer it may be mentioned that during a violent storm at Guadaloupe on the 6th September, 1865, the mercury fell 1·693 inch in one hour and ten minutes. Such sudden falls may be accompanied by a diminished atmospheric pressure on the water over a certain area outside of which a greater pressure prevails. This causes disastrous storm-waves which frequently accompany tropical hurricanes. Such storms are of a rotating character.

At meteorological stations a daily register is kept of the variations of the barometer, the wind, and the temperature, and some attempt is made to investigate the laws governing storms, so that forecasts of their movements may be made, and warning of their expected approach sent in certain directions. At such observatories a record of barometric observations is kept by the aid of self-registering apparatus, which shows at a glance the various fluctuations that have taken place in the barometer. For this purpose a wheel-barometer may be used, the float of which gives motion to a well-balanced lever. The shorter end of the lever is attached to the float, and the longer sometimes carries a pointer which is caused by clockwork to puncture at regular intervals a ruled card slowly moved under the point. A line joining the punctures shows in diagram form the changes in the height of the column.

Instead of a wheel barometer, Messrs. John Davis & Son supply a large and powerful aneroid and an eight-day clock. Between the aneroid and clock a vertically placed cylinder having a paper attached to it ruled to coincide with the barometer scale, revolves. Responsive to the action which takes place in the aneroid, a pencil moves up and down, and every hour is made to mark the paper by mechanism connected with the clock. A weekly barometric chart obtained from one of their self-recording aneroids, reduced to sea-level, is communicated and published in the *Colliery Guardian* by Messrs. John Davis & Son.

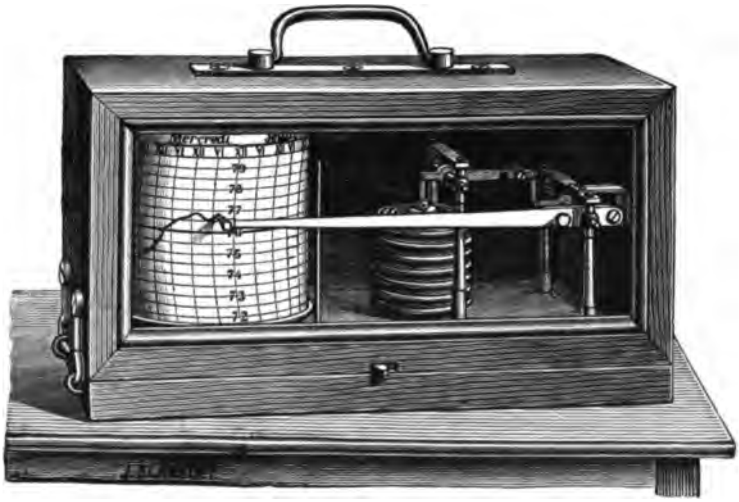


Fig. 418.—Self-recording Aneroid Barometer.

Fig. 418 is an illustration of a Self-Recording Aneroid Barometer or Barograph. The thin metal box of corrugated form is the vacuum chamber which is the prime mover as in the ordinary aneroid, but is of greater height. Its corrugations give the box a certain amount of elasticity which yet is sufficiently rigid not to collapse with the pressure of the atmosphere when the chamber is exhausted. The small movement of the corrugated surface is greatly increased as it is communicated by well-designed and delicately constructed levers to the pen which is held up to the paper-covered drum by the gentle pressure of the lever. These levers are compensated for temperature. The object of multiplying the movements of the vacuum box is to make variations in the atmospheric pressure more apparent in the line traced by the pen at one end of the long lever, which line will be more or less undulating in accordance with the rapidity and range of fluctuations in the atmospheric pressure.

A properly ruled paper chart, supplied with the instrument, is wound round the drum, which is revolved by clock-work. The drum must be of sufficient height to take a 4 or 5-inch diagram card, which allows for movements of the pen over the variations in height of the mercury column in this country. The horizontally ruled lines are marked upward from 27 or 28 inches to 31, or in equivalent numbers of another term of measurement, such as the 72, 73, &c., in the illustration, which are centimetres. The vertically ruled lines numbered XII., VI., XII , VI., &c., are 6-hourly divisional lines, and a point on the circumference of the revolving drum moves through a space equal to that contained between two adjacent vertical lines every six hours, and must not complete a whole revolution in less time than a week. Each day's record is contained on four of the 6-hourly divisional lines, and has the days of the week marked above, those on the illustration being in French.

A continuous ink line is thus automatically traced along the paper chart, which may be left undisturbed for a week. At the expiration of that time the chart should be changed, and the pen be re-filled with a special ink which is an aniline dye supplied with the instrument: if intended to be used for a separate daily record, the instrument is arranged accordingly, and the paper must then be changed every 24 hours. On the weekly chart being withdrawn it is dated and preserved for future reference.

The self-registering thermometer and the self-registering steam pressure gauge are very similarly constructed. In the latter, the movements of the Bourdon tube are multiplied and recorded on a paper chart by similar mechanism to that last described, and correspond with the pointer indications on the dial-plate. In the former a Bourdon tube is also employed. It is fixed outside the case so as to be more accurately influenced by the temperature of the surrounding air. Its movements are transferred to the pen by levers and mechanism similar to that used in the self-recording barometer.

In mechanical arrangements for periodic puncturing the paper the minor variations are lost. By the aid of photography a continuous line showing the fluctuations in a mercurial barometer may be traced on a paper. The barometer is placed in a darkened room and its moving column turns a small mirror. A ray of light concentrated by a lens from the light of a powerful lamp falls on the mirror. A sheet of sensitised paper is so placed as to receive the reflected ray however the mirror may be moved, and as this paper is caused to travel by clockwork, a continuous faint trace is left on it which takes a more or less curved form corresponding to every change in height of the column. The trace on the paper is afterwards developed and fixed when it is in a state for permanent preservation. A continuous line is drawn in ink without the aid of photography by the instrument shown in Fig. 418.

A maximum and a minimum thermometer form parts of some self-recording barometric apparatus.

By permission of the authorities of the Glasgow and Kew Observatories the barometer and thermometer readings, &c., are published as appendices to the *Transactions of the Federated Institution of Mining Engineers* so as to give some idea of the variations of temperature and of atmospheric pressure in the intervening colliery districts in which no reliable record may be kept.

The barometer at Kew being 32 feet, and that at Glasgow 180 feet above sea-level, a correction for the readings is made by reducing those at Glasgow to 32 feet above sea-level, by the addition of 0·15 inch to each observed reading, and the barometer readings at both observatories being reduced to 32° Fahr.

Beside showing these readings for each day throughout the year, diagrams are printed recording the meteorological observations which contain also a list of the fatal explosions in collieries, obtained from the annual reports of H.M. Inspectors of Mines.

The Mines Regulation Act says, "A barometer and thermometer shall be placed above-ground in a conspicuous position near the entrance to the mine," and it is well to have them placed at the pit bottom also and in the furthermost part of the workings. Frequent readings should be taken and recorded by qualified men, who would then be in a position to caution the workmen of repeated falls in the barometer.

It is not suggested that in efficiently ventilated mines where proper precautions are taken changes in the atmospheric pressure are in themselves sufficient to produce an explosion. The ventilating current passing along any face ought, under normal conditions of gas emission, to be ample to carry the gas harmlessly away even when affected by a gradual reduction of pressure. The Royal Commissioners on Accidents in Mines, speaking on this question, say: "Some remarkable instances have appeared to support the view that explosions accompany unusual depressions or specially rapid falls in the barometer, but the evidence which has been adduced to show that this view is of general application seems to us most imperfect and untrustworthy. Indeed, the absence of a general connection between colliery disasters and barometric changes is practically established by the tables compiled by some of your Majesty's Inspectors of Mines, by the Northern Institute of Engineers, and recently by Mr. Thomas Embleton in communications made to the Midland Institute."

Collieries having only feeble currents of air are of course much more seriously affected by a falling barometer, as gas comes off from the goaves much more freely with the reduced pressure then indicated. There is no excuse for neglecting the ventilation, whatever the state of the barometer, but the changes in the weather should be carefully watched, although gas in the mine is more sensitive to atmospheric disturbances than mercury, and abnormal issues are known to have preceded a fall of the barometer.

Even with a continuously high barometer an explosion may readily be brought about by neglecting the usual precautions, such as the opening of doors, or the derangement of brattice or air pipes, which in fiery mines would quickly cause gas to accumulate.

When the barometer is at 28 inches, the pressure of the air per square foot is about 1,979 lbs., and when at 31 inches it is about 2,191 lbs. To find the theoretical quantity of gas that would be given off from each 1,000 cubic feet of space in the gas-charged goaves of a fiery mine due to a fall of the barometer from 31 to 28 inches:—$31-28=3$ inches difference. Then, as $31 : 3 :: 1{,}000 : 96{\cdot}77$ cubic feet. So that if the goaves could be measured and were found to contain 10,000 cubic feet, there would be $96{\cdot}77 \times 10 = 967{\cdot}7$ cubic feet of fire-damp given off by them. The cubic contents of goaves cannot be measured with any degree of certainty, however, and any estimate of goaf contents must be based on the superficial area over which the goaf extends and an assumption of a certain percentage of it being more or less open. The amount of goaf present in some old collieries necessitates the greater caution in observing and noting the changes of the barometer.

Atkinson, in his *General Principles of Ventilation*, says:—"In ordinary states of the weather mercury is about 10,800 times as heavy as the same volume of air near the surface of the earth, and hence about 900 feet of ascent or descent makes a change of 1 inch of mercury in the height of the barometer." Again: "The air at the surface of the earth is generally pressed by the whole of the air above it, to an extent measured by 29·922 inches of mercury (reckoned at the density due to melting ice 32°), as shown by our common barometers; a pressure equal to 2,116·4 lbs. per square foot. To give this pressure we should require the air of the atmosphere to be 26,216 feet high, if it was all as heavy as the air at the earth's surface."

In forming a rule to meet the fluctuations of the barometer, take this 26,216 feet as being the height of the atmosphere at sea level, which gives its appreciable weight or pressure on the earth's surface.

Then for pits on the datum line of sea level will be obtained the following rule to find the height of the mercurial column corresponding to shaft depths:—

$I = \frac{D \times B}{26{,}216}$, where I = inches of mercury due to the shaft depth; D, depth of shaft in feet; and B, height of barometer at the pit top. The barometer reading at the pit bottom will then be the reading at the top + I, or in case where the height of the barometer is given at the top and bottom of a pit whose depth it is desired to know, and where I represents the difference of the two barometer readings, $D = \frac{26{,}216 \times I}{B}$.

Supposing a question like the following has to be answered. The barometer at the top of a shaft is 30·2 inches, the thermometer is 65° F., the depth of the shaft is 1,100 feet, and the thermometer stands at 75° F. at the pit bottom, say what is the difference in the pressure of the air at the top and bottom of the shaft, and the difference in the reading of the barometer.

Here $I = \frac{1{,}100 \times 30{\cdot}2}{26{,}216} = 1{\cdot}267$. Therefore the reading of the barometer at the pit bottom is $30{\cdot}2 + 1{\cdot}267 = 31{\cdot}467$. To get the weight of a cubic foot of air at the shaft top by Atkinson's formula, $\frac{1{\cdot}3253 \times 30{\cdot}2}{459+65} = {\cdot}0763814$ lb. Similarly, $\frac{1{\cdot}3253 \times 31{\cdot}467}{459+75} = {\cdot}078096$ lb. as the weight of a cubic foot of air at the shaft bottom.

Hence, $\cdot078096 - \cdot0763814 = \cdot0017146$ lb. difference in the weight per cubic foot of the air at bottom and top of the shaft.

The *Water-gauge* is a very simple instrument, and consists of a glass tube bent like the letter U, both ends of the tube being open. A little water is placed in the bend of the tube, which forms the bottom part of it. A sliding scale of inches and decimals of an inch is attached to it. At the top of one arm of the tube is placed a nose-piece, by means of which it is passed through a door, and, by so doing, one side of the tube is placed in contact with the air on one side of the door, and the other is exposed to the influence of the atmosphere or air current at the other. Where a difference of atmospheric pressure exists, such as would be between the intake and return currents of air near the shaft, or between a fan drift and the outside air on the surface, the water is depressed in one side of the tube and raised in the other. The scale of inches and decimals shows the difference of level in the tubes. The instrument is used thus to show the force of the air current generated, whether by furnace or fan.

Fig. 419 shows Daglish's water-gauge for use in mines as made by Messrs. John Davis & Son, of Derby. It is fixed by small brackets to a mahogany back for safe carriage, and has a small thumb-screw at the bottom by means of which the scale can be adjusted; a level to ensure its being inserted in an upright position when making an observation; a 6-inch scale divided into inches and tenths; and a brass cap with branch pipe at one end for insertion where required. This form of water-gauge is also made with longer or shorter scales to suit requirements.

Davis' pocket water-gauge is only 6 inches in length, that of the tube being 4½ inches and the scale 3 inches. This leaves a travelling space of 1½ inches for the adjustment of the scale. The gauge is made with a level and with or without a thermometer, the card to which the instrument is attached forming the back of the case in which it is carried. This instrument is very handy, but does not admit of readings higher than 3 inches. Other lengths of tubes and scales are, however, made in the pocket form to suit a greater range if desired.

In some instances it is extremely difficult to obtain accurate readings of the ordinary water-gauge. The variations in pressure which are constantly going on cause a vibratory motion of the water in the glass tube, and this is particularly noticeable where there is a displacement ventilator or with any mechanical ventilator where the upcast shaft is used for winding coal. The running of the cages and the continual opening and shutting of the doors across the pit-top cause fluctuations in the water. Under such circumstances more or less successful attempts are made to watch the oscillations in both legs simultaneously so as to adjust the scale with its zero mark at the mean of the fluctuation in one leg while reading the scale at the mean of the extreme movements in the other. Such difficulties have naturally led to improvements in the instrument. The idea of contracting the bend in the glass tube so as to reduce the passage for water between the tubes is due to Mr. John Daglish.

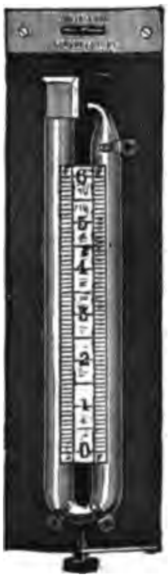


Fig. 419.

The Daglish Water-Gauge.

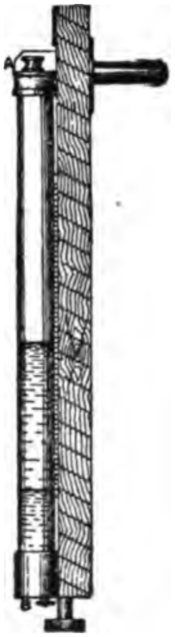


Fig. 420.

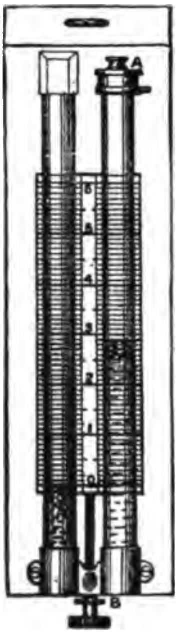


Fig. 421.

The Davis Improved Water Gauge.

An improved form of water-gauge made by Messrs. John Davis & Son is shown in Figs. 420 and 421.

It consists of two parallel glass tubes with their lower extremities sealed into a hollow brass pedestal through which there is a connection or water passage between the tubes. In the centre of the brass pedestal is a tap B pierced by a small hole, making a passage for the water through it when the tap is in a certain position. This tap takes the place of the reduced bend in the Daglish form of water-gauge. There is difficulty in obtaining a continuous U-shaped glass tube with a small hole or contraction of uniform size in the bend, but there is no difficulty in making or retaining the hole in the brass tap of a standard uniform size. The restricted passage for the water through the hole in the tap prevents the oscillation due to varying pressure being so marked as in the Daglish form of water-gauge. The throttling of the water at the tap prevents quick vibratory motions whilst it still affords a sufficiently free connection between the two columns of water.

Two or more persons frequently differ in the readings obtained from the ordinary water-gauge. With the Davis improved gauge the height can be registered by simply turning the brass tap B, thus closing the passage between the two tubes; the height of the water due to the ventilating power is then fixed in the gauge and may remain in that position any length of time until the tap is reopened and the connection re-established. Thus, when the observer wishes to make a water-gauge measurement in a fan-drift or other dark or inconvenient place, he can, at the right moment, close the passage between the tubes, thus keeping awhile in his hand a record of the difference between the two columns of water; on emerging to daylight, anyone may read off the measurement without question as to its accuracy. This means of registering the water-column may be made use of to ascertain the maximum and minimum oscillations of the water-gauge arising from the doors at the top of the upcast pit being opened when winding, and numerous records may be made without haste and afterwards read in a good light.

As seen in Fig. 419 the scale in the Daglish form of water-gauge is placed in front of the glass tubes and partly covers them on either side. This renders it difficult to ascertain the height of water correctly because, owing to capillary attraction, the surface of the liquid in the tube is not even. In the Davis water-gauge the scale is placed at the back and for the full width of both tubes. Every black division line of the scale passes across each tube and so allows the eye to observe the mean height across the whole width of the water surface in the tubes. In ascertaining the line of water-surface in either tube, the gauge should always be held level with the line of sight of the observer. Wherever minute accuracy of reading is required, the observer may use a magnifying glass to ascertain the water level in relation to the divisions upon the scale.

It is usual to charge the Daglish water-gauge by first turning it on one side and then pouring the water through the small hole of the brass branch pipe. As the air issues at the same small opening at the time the water is entering, the operation is somewhat inconvenient. In the Davis water-gauge the top of one tube is closed with a brass cap, having a small hole in the top. When it becomes necessary to charge it, the cap A is unscrewed and water is poured down the glass tube until it reaches the required height, whilst the gauge is held in a vertical position. This operation is easily performed, after which the brass cap is screwed on and the gauge is ready for use.

An advantage in using the Davis water-gauge arises from the fact that in case of breakage only the broken tube will require renewal. As straight boiler gauge-glasses are used the instrument is easily and cheaply repaired. In the older form of water-gauges the tubes are all in one piece, a breakage in one limb necessitating the renewal of both.

Where mechanical appliances are used to produce ventilation, water-gauge observations are taken on the surface, and as near to the ventilator as possible, care being taken to fix the water-gauge out of the influence of any eddying currents, which are apt to produce erroneous results. When the gauge is so fixed its readings indicate the total difference of pressure employed in producing the ventilation.

In furnace ventilation water-gauge observations cannot be taken on the surface, nor underground where the upcast shaft is far distant from the downcast. If, however, they are not far apart and are directly connected by a passage leading from the bottom of one to that of the other, observations are taken by fixing the water-gauge in a perfectly air-tight stopping or door erected in such passage. These readings also show the difference of pressure employed in producing the ventilation.

Where the circumstances admit of it the water-gauge may be permanently fixed in an underground cabin near the downcast shaft, a small pipe being led from it to a point just beyond the separation doors. With mechanical ventilators on the surface the water-gauge is usually fixed in the ventilator engine-house, and the readings, as well as the number of strokes or revolutions of the engine per minute, are recorded by the engineman hourly or every two hours. In some instances water-gauges are fixed in the offices near the collieries.

With regard to the fixing of a water-gauge for accurate observations, the end of the branch pipe should be placed at right angles to the direction of the air current so as to avoid pressure due to velocity acting upon it. Fig. 422 shows a water-gauge fixed in a door or stopping. The arrows indicate the points acted upon by pressure of air on the intake and return sides respectively. Such observations may be taken at suitable places in the mine whatever the means of ventilation.

The weight of a cubic inch of water being ·036 lb., if the water-gauge read 1 inch the pressure is ·036 lb. per square inch, or ·036 × 144 = 5·184 lbs. per square foot; usually taken at 5·2 lbs. per square foot. For any other reading of the water-gauge, multiply 5·2 lbs. by that reading to find the pressure per square foot. Thus, ·5 on the water-gauge = ·5 × 5·2 = 2·6 lbs. pressure per square foot. The water-gauge acts as a check on the state of the air-courses. When they remain in the same state over, say, two days, and the ventilating power is not increased or lessened, the water-gauge on the second day in each position of trial should, under ordinary circumstances, read the same as on the first, and, if not, it would probably be owing to a fall in some aircourse or aircourses which had increased the friction.

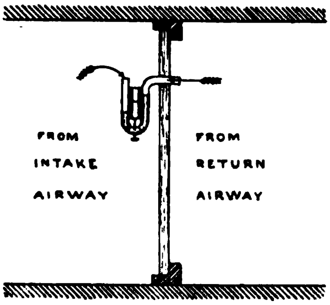


Fig. 422.—WATER-GAUGE PLACED IN A DOOR OR STOPPING.

It will be well here to consider some of the rules for working out problems connected with furnaces and fans as used for motive powers.

The power obtained by furnace ventilation is measured by the difference between the weight of the air in the downcast and upcast shafts. The length of column in the downcast, which would be equal in weight to the difference of the weight of the air in the two shafts, is called the motive column, to find which use the following formula:—

Let M = Motive column in feet.
" T = Average temperature of air in the upcast.
" t = Temperature of the air in the downcast.
" D = Depth of the downcast in feet.

$$\text{Then } M = D\,\frac{T-t}{T+459}.$$

If the depth of upcast and downcast were each 300 feet and were 12 feet in diameter, or had 113 feet superficial area, the temperature of the downcast 60° F. and the average of that in the upcast 150° F., the motive column would be $300 \times \frac{150-60}{150+459} = 44{\cdot}33$ feet, that is, the air in the downcast would balance the air in the upcast with a column 44·33 feet shorter than that in the upcast.

From the formula already given to find the weight of a cubic foot of air at any temperature, and under any pressure, if the barometer read 30 inches, to find the weight of a cubic foot in the downcast, $\frac{1{\cdot}3253 \times 30}{459+60} = {\cdot}0766$ lb., and this multiplied by the depth in feet and by the area thus, ·0766 × 300 × 113 = 2,596·74 lbs.

the total weight in the downcast. For the upcast, $\frac{1·3253 \times 30}{459+150} = ·06529$ lb., the weight of a cubic foot and multiplied by 300 and by 113 = 2,213·34 lbs. as the total weight in the upcast. 2,596·74 − 2,213·34 = 383·4 lbs. as the difference in weight in the two shafts. Therefore to find the motive column, as 2,596·74 : 300 :: 383·4 : 44·3, as given before. If the quantity of air circulating were 150,000 cubic feet per minute, proceed to find the horse-power exercised by the furnace thus:—The weight of a cubic foot of air in the downcast being ·0766 lb. at the bottom of the pit, there would be a pressure of ·0766 × 300 = 22·98 lbs. due to its mere weight. In the upcast the weight would be ·06529 × 300 = 19·587 lbs. and 22·98 − 19·587 = 3·393 lbs. as the difference of pressure on each square foot of area. Therefore $\frac{150,000 \times 3·393}{33,000} = 15·4227$ horse-power.

Some authorities use the formula $M = D \times \frac{T - t}{t + 459}$ which gives the length of motive column in feet of air of the temperature in the upcast, and this formula will give the same pressure in pounds per square foot as $M = D \times \frac{T - t}{T + 459}$ which gives the length of motive column in feet of air of the temperature of the air in the downcast, but the former gives also the theoretical velocity, viz., $v = \sqrt{2gM}$, with which the air would escape from the upcast neglecting friction. Applying this formula to the foregoing example $M = 300 \times \frac{150 - 60}{60 + 459} = 52·02$. Or, taking the total weight as before, worked out in the downcast at 2,596·74 lbs., and in the upcast at 2,213·34 lbs., with a difference between the two of 383·4, the motive column in feet of air of the temperature of the upcast may be found by proportion thus:—As 2,213·34 : 300 :: 383·4 : 51·97, as before, except a slight difference owing to the loss of decimal places.

The simplest way of getting the difference of pressure on each square foot of area between the upcast and downcast is by using the water-gauge. If this had been tested at the separation doors between the upcast and downcast shafts in the case just given, it would have read about ·65″, because 5·2 × ·65 = 3·38 lbs., as will be understood by reference to the remarks on the water-gauge.

This head or motive column may be easily converted into inches of water, as shown in the water-gauge. The motive column is always expressed in feet, and as the weight of a foot of air at 60° F. is 0·0766 lb., and that of an inch of water is 0·036 lb., this gives a pressure of ·036 × 144 = 5·184, but, as before stated, usually taken at 5·2 lbs. per square foot; therefore multiply this motive column by ·0766 (= ·0766 × 44·3 = 3·393 lbs.), the pressure of a foot of air, and divide the product by 5·2 lbs., the pressure of an inch of water, to find the indication of the water-gauge. $\frac{3·393}{5·2} = ·652$. Or, what is the same thing, divide the motive column at once by 68 to find the water-gauge, because $\frac{5·2}{·0766} = 68$ nearly. Taking then the motive column at 44·3 feet in the case being considered, the water-gauge would read $\frac{44·3}{68} = ·6514$.

Or, if the motive column had been expressed in feet of air of the temperature of the air in the upcast, a foot of air at 150° F. as shown, weighs ·06529 lb., and if the motive column of 52·02 feet be multiplied by ·06529 lb., the pressure of a foot of the air, and the product be divided by 5·2, thus $\frac{52·02 \times ·06529}{5·2}$, the result is the water-gauge = ·653, which is almost the same as before, proving that the pressure in pounds per square foot is the same whichever formula is used.

To find the horse power of ventilation then multiply the pressure per square foot (which may be ascertained by multiplying the water-gauge in inches by 5·2) by the cubic feet of air passing per minute, and divide by 33,000.

The ventilating pressure is chiefly required to overcome the resistance due to friction and obstruction, that required to put the air in motion being very slight. If that pressure be expressed as a head of air or motive column, the velocity due to the pressure will be equal to that which a body would acquire when it had fallen through a height equal to the head. A column of air 1 square foot in section and 13·09 feet high weighs 1 lb., and will therefore exert a pressure of 1 lb. to the square foot. This pressure produces a velocity in the air current equal to that which would be attained by a falling body through a height of 13·09 feet. This is usually expressed by the well-known formula for gravity, $V^2 = h\,2\,g$, whence $V = \sqrt{h \times 2\,g}$ in which V is the velocity in feet a second, *h*, the height or space in feet fallen through, and *g*, the velocity in feet acquired by a falling body at the end of one second of time, the value of which is 32·2. Thus $V = \sqrt{h \times 64{\cdot}4} = 8{\cdot}02\sqrt{h}$, and therefore $h = \frac{V^2}{64{\cdot}4}$.

If the air current has a velocity of 4 feet per second, the head required to produce this velocity (omitting all consideration of friction) would be calculated thus, $h = \frac{4^2}{64{\cdot}4} = {\cdot}2484$ foot, or expressed in inches of water-gauge $\frac{{\cdot}2484}{68} =$ ·00365. Strictly speaking, this value should be added to that found for frictional and other resistances in mines, but when it is noted how very small is the pressure required to produce the velocity, and also the fact that the resistances are only estimated approximately, it is not surprising that the pressure to produce the velocity is in practice neglected, and the calculations rendered easier by such neglect.

The economy of a fan is often judged of by what is called its useful effect. This simply means the proportion that the power of ventilation bears to the horse-power exercised by the engine in driving the fan. Thus, suppose the horse-power of a fan-engine to be 100, and the horse-power of ventilation to be 50, we say that the useful effect of the fan is 50 per cent. Where it is desired to work out questions as to the useful effect of ventilating fans, it is necessary to be very exact in all data, or the results are misleading. Thus, the air must be very carefully and accurately measured at a point near the fan inlet, as explained under remarks on the anemometer. If it is desired to calculate the useful effect on the volume of intake air, a correction will have to be made for pressure and temperature, as the volume of air in the fan drift will be increased as compared with its state at the intake owing to different barometer and thermometer readings. That is, supposing in the ventilator drift the barometer reads 30 inches and the temperature shows 70° F., whilst the readings are 31 inches and 40° F. respectively at the intake airway. Then for every 1,000 cubic feet per minute in the intakes, the volume it would occupy in the ventilator drift will be found by the formula already given thus: $1{,}000 \times \frac{31 \times (70 + 460)}{30 \times (40 + 460)} = 1{,}095{\cdot}3$ cubic feet. Then, as 1,095·3 : 1,000 :: quantity in fan-drift : intake quantity.

Again, very accurate diagrams must be taken with the steam indicator (of which more will be said hereafter) on both sides of the piston, and these should be made simultaneously with the air measurements, and in calculating the horse-power of the engine allowance must be made for the area of the piston rod, whether on one or both sides of the piston, as the case may be. Any natural ventilation operating with or against the fan should be carefully ascertained and allowed for. As it is evident that *all* the horse-power of the engine is not used for driving the fan, but a part is required to overcome its own resistances, these should be ascertained by disconnecting the engine from the fan, and diagrams taken of it when running at the same speed as when working the fan. Suppose it be required to find the useful effect of a fan when there are 200,000 cubic feet of air passing per minute with 2 inches of water-gauge. The fan is worked by an engine having a 28-inch cylinder and $4\frac{1}{2}$-foot stroke, there being a piston rod of 4 inches' diameter on either side of the piston, the effective pressure of steam on the piston is 30 lbs., and it has a speed of 270 feet per minute resulting from 30 revolutions. The horse-power to work the engine without the fan has been ascertained to be 18.

The horse-power of the fan is $\frac{200{,}000 \times 5{\cdot}2 \times 2}{33{,}000} = 63$. That of the engine is $28^2 \times {\cdot}7854 = 615{\cdot}753$ area of piston; $4^2 \times {\cdot}7854 = 12{\cdot}566$ area of piston-rod; $615{\cdot}753 - 12{\cdot}566 =$ say 603 effective area of piston $\frac{603 \times 30 \times 270}{33{,}000} = 148$ horse-power of fan engine. As it requires 18 horse-power to work the engine itself, $148 - 18 = 130$ as the useful horse-power of the engine. Therefore, as $130 : 63 :: 100 : 48{\cdot}5$ per cent., which is the useful effect of the fan.

As to the amount of air exhausted by fans of given dimensions, it is almost impossible to say what that might be; very much would depend upon the condition and size of the airways, and these vary, so that the same sized fan of the same make gives different results at different collieries, sometimes being assisted by the natural ventilation and sometimes not.

A centrifugal fan, properly proportioned, and employed merely in displacing air, that is, under no drag, should deliver (at a velocity equal to the tips of its blades) a stream of air having a sectional area equal to the breadth of its blades at their outer ends, multiplied by the circumference of the circle described by those ends.

Also, the greatest water-gauge which any centrifugal fan can afford is dependent upon the speed at which the tips of the blades can safely be driven. Theoretically the depression of water-gauge, due to the velocity of the periphery of a perfect fan is equal to twice the height of column necessary to create such velocity in a falling body.

Take the case of a fan 24 feet in diameter, and allow it to run 64 revolutions per minute, it will be seen from the law $h = \frac{V^2}{64{\cdot}4}$ when $V =$ the speed of the tips of vanes in feet per second, that theoretically the greatest water-gauge it could afford would be 2·954 inches, thus $h = \frac{(24 \times 3{\cdot}1416 \times \frac{64}{60})^2}{64{\cdot}4} = 100{\cdot}43$; twice this head $= 200{\cdot}86$, and $\frac{200{\cdot}86}{68} = 2{\cdot}954$ inches of water-gauge, taking the atmospheric air at a temperature of 60° F. and at 30 inches barometric pressure.

The theoretical water-gauge was calculated on the old system by the formula $h = \frac{v^2}{2g}$, but M. Daniel Murgue's rule proceeds by the formula $h = \frac{v^2}{g}$ which gives exactly double the result. On the continent Murgue's rule is universal for the theoretical water-gauge due to the peripheral velocity, and the same standard should be adopted everywhere.

Experiments seem to show that with a Guibal fan a certain amount of benefit is derived from the shutter and chimney, and that the water-gauge as actually taken during those experiments gave a slight excess over the theoretical water-gauge.

Equivalent Orifice.—If an opening be cut in a thin plate and the latter be then placed at right-angles to the direction of air in motion so that the air in its passage is obstructed by the plate, it meets with resistance in passing through the opening. M. Murgue assimilates the workings of a mine to such an opening in calculations for ventilating purposes, which he calls the *equivalent orifice.*

To find the equivalent orifice for any given mine:—

Let Q = Quantity of air in cubic feet per second passing through the opening (*i.e.* circulating round the mine).
h_a = Ventilating pressure in feet of air column, required to overcome the resistance of the mine.
A = Opening in thin plate in square feet (*i.e. equivalent orifice*).
k = Co-efficient of contraction of orifice (*i.e. vena contracta* = ·65).

Then:—

$$Q = \sqrt{2\, g h_a} \times k\, A = \sqrt{2\, g h_a} \times \cdot 65\ A.$$

$$\therefore A = \frac{Q}{\cdot 65\ \sqrt{2\, g h_a}}$$

It is however often more convenient to use the following units, viz.:—Q in thousands of cubic feet per minute; h_a in inches of water-gauge.

When the formula becomes

$$A = \frac{0{\cdot}388\ Q}{\sqrt{h_a}} \quad \therefore\ Q = \frac{A\ \sqrt{h_a}}{0{\cdot}388} \text{ and } h_a = 0{\cdot}1505 \left(\frac{Q}{A}\right)^2.$$

Thus from the above $Q = \sqrt{2 g h_a} \times \cdot 65$ A we have in thousands of cubic feet per minute

$$Q = \sqrt{64{\cdot}4 \times 68\ h_a} \times \cdot 65\ A \times \cdot 06$$

$$\therefore Q = 66{\cdot}175\ \sqrt{h_a} \times \cdot 039\ A$$

$$25{\cdot}64\ Q = 66{\cdot}175\ \sqrt{h_a} \times A$$

$$\frac{25{\cdot}64\ Q}{66{\cdot}175\ \sqrt{h_a}} = A$$

$$\frac{\cdot 3875\ Q}{\sqrt{h_a}} = A$$

The average value of A for English mines is said to be about 20 and for Belgian 8·6 square feet.

Orifice of Passage.—The air in being exhausted from a mine meets with obstruction at the ventilator, the effect of which is to reduce the duty of the ventilator. The observed depression produced by the ventilator is always higher than where ascertained in the roadways of approach, the increase being to some extent due to overcoming the various resistances in the ventilator. M. Murgue treats this as the *orifice of passage.*

As to the volume of the ventilative current required for any particular colliery, this will depend entirely upon its own circumstances. The volume of air inhaled by a man is 27·8 cubic feet per hour. Smyth, in his *Coal and Coal Mining,* says: "In round numbers 100 cubic feet of air per minute may be required for the health and comfort of each person underground, or for 100 men 10,000 cubic feet; but if fire-damp be given off at, say, the rate of 200 cubic feet per minute, we should need at the very least 30 times that amount of fresh air to dilute it, or 6,000 cubic feet in addition. Increase the number of men and liability to gas and 40,000 or 60,000 cubic feet of air may be indispensable for safety." In André's *Mining Engineering* there is a definite formula for finding the ventilative current necessary in a mine as follows:—

$$V = m\,24 + h\,72 + p\,192 + f\,(q \times 2{,}700 + s).$$

where V = Ventilative current in cubic feet a minute.
m = The number of persons employed.
h = The number of horses in the mine.
p = The weight in pounds of the gunpowder consumed an hour as a maximum.
f = A factor of safety which would vary in different districts as well as in different mines, so as to be 2, 3, 4, 5, or more times: this can only be determined by the judgment of the engineer, but it must not be taken at less than 2 under the most favourable circumstances of a mine worked by longwall (post and stall requiring more) and yielding very little firedamp.
q = The output or average quantity of coal raised a minute in tons.
and s = The exposed surface of the coal in square yards, which can be calculated from the output and the thickness of the seam.

A horse is assumed to breathe 6 times the quantity of air required by a man, and to require 3 times the quantity required by a man and his lamp.

Applying this rule to a mine worked by longwall, in which say 400 tons per day of 8 hours are raised, 200 men employed underground and 10 horses, and 10 lbs. of gunpowder per hour are used, the extent of coal surface exposed being 600 square yards, we have $V = (200 \times 24) + (10 \times 72) + (10 \times 192) + 2\left(\frac{400}{8 \times 60} \times 2{,}700 + 600\right)$.

Then $V = 13{,}140$, that is taking f, the factor of safety, at its lowest figure and assuming the mine to be fairly free of gas, but if it be fiery and the factor 5 be adopted, the quantity would be 21,690 cubic feet.

If Smyth's rule be applied to this example—

$$\begin{array}{rr} \text{For the men } 200 \times 100 = & 20{,}000 \\ \text{,, ,, horses } 10 \times 300 = & 3{,}000 \\ \hline \end{array}$$

it gives $\underline{\underline{23{,}000}}$ cubic feet, besides what may be necessary for the gas, which is not definitely known.

CHAPTER XII.

THE FRICTION OF AIR IN MINES: VENTILATION.

The Pressure necessary to Overcome Friction—Rate of Increase or Decrease of Pressure—Power necessary to produce Ventilation—Rate of Increase or Decrease of Power—Best Form of Airway—The Co-Efficient of Friction—Measurement of Ventilating Pressure—Loss of Pressure in Shafts and Airways—Splitting the Air into an Upper and a Lower Coalseam—Dimensions of Ventilating Fans—Splitting the Air in Three Coalseams—Splitting in One Coalseam in Communication with more than Two Shafts—The Effect of Obstructions and of Regulators in Airways—Bratticing—Natural Ventilating Pressure—Examples on Pressures and Powers of Different Shaped Airways—Questions and Answers on Ventilation.

THE friction of air in mines arises from its rubbing along the top, bottom and sides of the aircourses in its course round the workings. It is not difficult to understand that the more rubbing surface there is presented to the air, the more friction there will be, and that the amount of rubbing surface depends upon the length and perimeter of the road along which the air is taken. It is also obvious that the faster the air is made to travel, the more will be the friction. The late Mr. Atkinson has most ably argued the theory of circulating the air in mines, and he says in his *Practical Treatise on the General Principles of Ventilation*,* that "the pressure required to overcome the friction of air increases and decreases in exactly the same proportion that the area or extent of the rubbing surface exposed to the air increases or decreases, so that when the velocity of the air and the sectional area of the airway, remain the same, the pressure required to overcome the friction is proportional to the area or extent of the rubbing surface exposed to it; and hence if we double or treble the extent of the rubbing surface, we also double or treble the friction, or what is the same, the force or pressure required to overcome it." Again he says "the pressure required to overcome the friction in the same airways varies in the same proportions that the *square* of the velocity of the air increases or decreases, so that a double velocity of air in the same airway meets with a *double double* or fourfold resistance, a treble velocity meets with a *treble treble* or ninefold resistance; and a velocity of four times as great gives rise to a resistance four times four or sixteen times as great. In the same way a half velocity meets with one half of a half or $\frac{1}{4}$th of the resistance; $\frac{1}{3}$rd of the velocity encounters only a third of a third or $\frac{1}{9}$th of the friction, and so on." . . . . . "It seems probable that for every foot of rubbing surface and for a velocity in the air of 1,000 feet per minute, the friction is equal to 0·26881 feet of air column of the same density as the flowing air, which is equal to a pressure, with air at 32°, of 0·0217 lb. per square foot of area of section. Calling this the *co-efficient of friction*, we have the following rules with respect to the friction of air in mines:—

Total pressure $p\,a = k\,s\,v^2$

Rubbing surface $s = \frac{pa}{kv^2}$

Velocity squared $v^2 = \frac{pa}{ks}$

Co-efficient of friction $k = \frac{pa}{sv^2}$

---

* See pp. 34–55, 1st Edition, 1871.

Pressure per foot $p = \frac{ksv^2}{a}$

Area of section $a = \frac{ksv^2}{p}$

where p = pressure per square foot.
a = square feet of sectional area.
s = the area of rubbing surface exposed to the air.
v = the velocity of the air in thousands of feet per minute, 1,000 feet per minute being taken as the unit of velocity.
k = the co-efficient of friction in the same terms or unit as p is taken in."

...... "The quantity of air only varies as the cube root of the *power*, and of the quantity of coals burnt to produce it; so that eight times the coals only double, and twenty-seven times the coals only treble the quantity of air circulating in a mine, whether the ventilation is produced by furnace action, ventilating machines, or otherwise, so long as the airways remain in the same unaltered state."

It may be learned from these quotations that theoretically airways should be as smooth and as free from obstructions as possible, because roughness and inequality such as would be caused by projecting pieces of rock or timber, or from falls in aircourses, produce friction. Again, theoretically, the best form of aircourse is the circular, because a circle whose area is 1 square yard has a perimeter equal to 3·545 yards, whereas a square whose area is 1 square yard has a perimeter of 4 yards. The square form is preferable to the rectangular, for a rectangular airway 6 feet by 1½ feet has an area of 1 square yard, but its perimeter is $2 + 2 + \frac{1}{2} + \frac{1}{2} = 5$ yards. It is not convenient to have circular airways in mines, so the square form should be adopted as far as practicable; also theoretically, large airways are preferable to a number of small ones representing the same area.

THE CO-EFFICIENT OF FRICTION.

The late inspector of mines for the South Durham district, Mr. J. J. Atkinson, fixed the value of the co-efficient of friction in the galleries of a coal mine at 0·00417. In other words, a depression measured by 0·00417 inch of water is required to propel air at a velocity of 1,000 feet per minute through a gallery whose rubbing surface is one square foot. This is equivalent to a pressure of 0·0217 lb. per square foot. Valuable experiments to determine the value of the co-efficient have been made by others, with different results. M. Daniel Murgue, a very able authority on questions of mine-ventilation, made a series of careful experiments upon short lengths of airways, with the object of ascertaining the value of this co-efficient for three well-defined classes of airways, viz., galleries unsupported, galleries lined with masonry, and galleries propped with timber. In these experiments M. Murgue used a delicate form of water-gauge, the readings upon which could be observed, by means of a microscope, to 0·0008 inch. In straight galleries, with unsupported sides, four experiments were made, three of which were in normal or large cross-measure drifts. The mean value of the co-efficient of the loss of pressure in the three airways was equal to 0·00095. The fourth experiment was made in a rising gallery of small area, and a greater air velocity was produced. Here the co-efficient was 0·00123. In galleries lined with masonry, five experiments were made. Of these, two were made in normal straight airways of regular outline, the mean value of the co-efficient of the loss of pressure being 0·00033. One experiment was made in a normal gallery having a long continuous curve, almost a semicircle (see Fig. 422A), but the air measurement was made in the straight portion preceding the curve. Here the value of the co-efficient of the loss of pressure increased to 0·00062, so that the projection of the current of air against the concave side of the unbroken curve, had the effect of nearly doubling the friction in the straight gallery. Two other experiments were made in galleries, one of intermediate, and one of small area, which, although sinuous, had a general straight direction (see Fig. 422B). In the larger of these, the mean value of the co-efficient of friction was 0·00052 and in the smaller, which was only 22·9 square feet area, the co-efficient increased to 0·00055. Three experiments were made in timbered galleries, two of which were in normal airways, having the usual sinuosities of galleries driven in the seam and not set out by marks, one being an old gallery, and the other a newly-repaired one. The mean value of the co-efficient of friction obtained was 0·00158. A final experiment was made in a gallery of small area, the curvatures in the airway being greater than in the preceding cases, and the surface irregularities between the props relatively of greater importance. In this instance the co-efficient of friction reached the highest point obtained, viz., 0·00241.

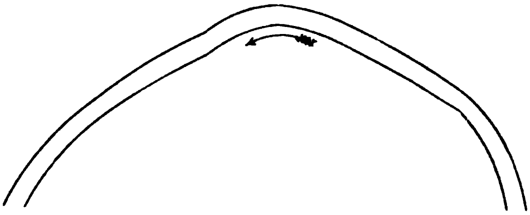


Fig. 422A.—Illustrating the Effect of a Curve in Increasing the Co-efficient of the Loss of Pressure.

From these experiments it is plain that the values of the co-efficient of friction are not only different in different types of galleries, but are influenced by curves in the airways, the sectional area, and their inclination. In galleries of small area, for each of the three types, there is an increase in the co-efficient of friction.

It will be seen then, that there is difficulty in fixing a mean value for the co-efficient of friction for the whole of the galleries in a mine. In conducting any experiment upon the friction of air in mine galleries, there is great difficulty in obtaining an adequate length of gallery possessing a continuous regularity of area and uniformity of outline. Results obtained from experiments in short sections of airways, probably differ from those made in long ones, where the presence of refuge holes, stenton ends, and other openings interrupt the regularity of air-flow and increase the amount of resistance. However desirable it may be to have a correct co-efficient of friction for air-currents, it is not possible to fix one which can be applicable to all mines, but that of 0·00417 inch of water-gauge adopted by the late Mr. J. J. Atkinson, has hitherto been generally accepted. It has therefore been thought advisable to base all calculations which appear in this work, on these figures; if they are wrong, the error is on the safe side, and this is of importance when considering the friction of the mine, with a view to the erection of a ventilating fan.

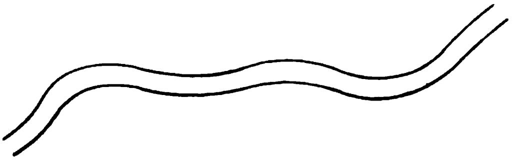


Fig. 422B.—Ordinary Sinuosities in a Mine Gallery.

THE MEASUREMENT OF VENTILATING PRESSURE.

If an ordinary water-gauge be fixed on a stand in a mine gallery (see Fig. 422c), having one limb connected by means of an open-ended pipe with the point A, and the other having fixed to it a longer pipe-connection also with an open end at the point B, more or less distant from A, the stagnant air in the tubes will transmit to the water-gauge the pressures existing in the open ends at the two points. Whatever modification of effect is exercised on the pressure by one orifice is also exerted by the other, so that the difference of level of the water in the gauge, is a

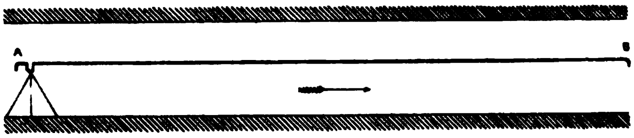


Fig. 422c.—Ascertaining the Difference of Pressures between the points A and B in an Underground Passage.

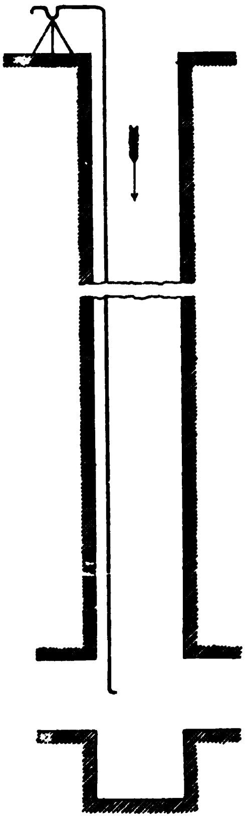


Fig. 422d.—Ascertaining the Difference of Pressures between the Top and Bottom of a Downcast Shaft.

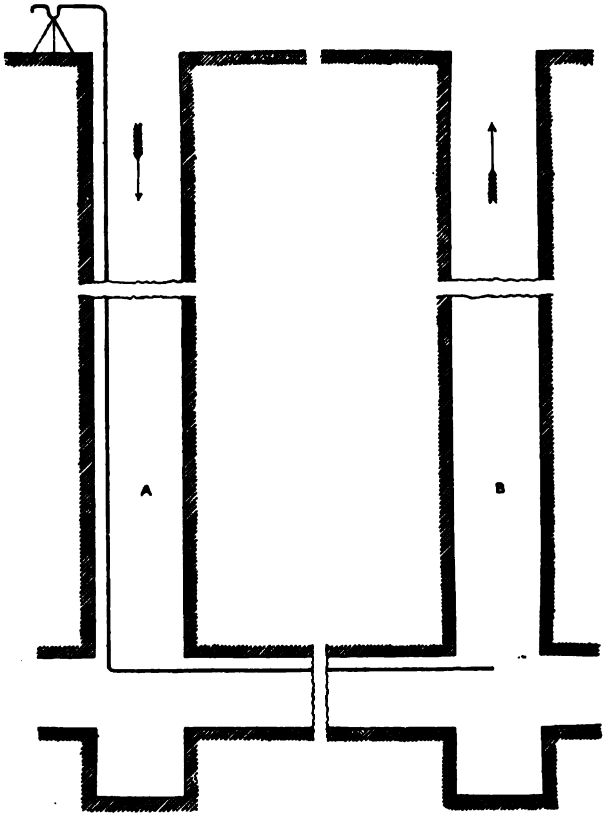


Fig. 422e.—Ascertaining the Difference of Pressures between the Top of a Downcast and the Bottom of an Upcast Shaft.

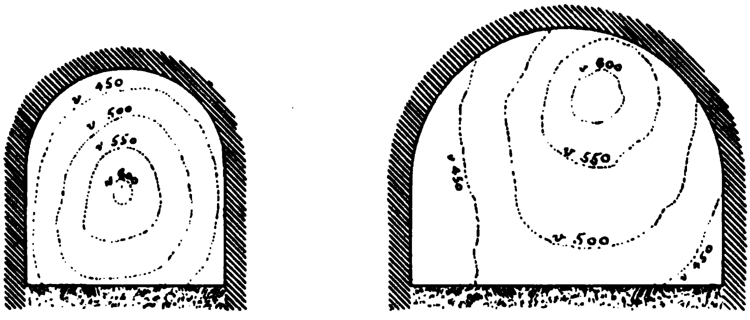
true indication of the loss of pressure due to the circulation of the air in the gallery A B. In the sketch the arrow shows the direction in which the air passes.

Similarly, the exact measure of the difference of the pressures between the top and bottom of a shaft, may be ascertained as in Fig. 422D, in which the water-gauge is shown on the surface with a pipe-connection down the shaft, or the loss of pressure between the top of the downcast and the bottom of the upcast, or *vice versâ*, may be measured as shown in Fig. 442E; or again, if a long pipe be fixed in the downcast shaft from top to bottom, and continued through the separation doors along the gallery to the upcast and to the top of that shaft, the pressure produced by the furnace or fan would be shown if the pipe were broken and a water-gauge attached thereto in the gallery. The difference of level produced in the two columns of water would be registered in the water-gauge. Instead of fixing a long connecting pipe in the mine galleries, the friction of a certain road or section of the mine may often be more easily obtained by inserting the water-gauge at a stopping or door between an intake and its adjacent return airway, as shown in Fig. 422.

If in Fig. 422C, or in Fig. 422D, an increased quantity of air is made to circulate, the ventilating pressure as measured by the water-gauge increases. The increase in the water-gauge is proportional to the square of the mean velocity of the air. If the water-gauge at first read ·5 inch and the velocity of air be doubled, the indication of the water-gauge will then be $\cdot 5 \times 2^2 = 2$ inches Frequent experiments have proved that this is a sound basis for the calculation of the loss of pressure in mine galleries, although some authorities maintain that this ratio ceases to be exact for small velocities.

ASCERTAINING THE VELOCITY OF AIR.

The mean velocity at which air circulates through underground airways is not easily ascertained. It may be thought that the retarding influence of the sides, roof, and floor would lessen the velocity of the air near them, and that a gradual increase towards the centre of the road would take place, the greatest velocity being in the centre, as shown at A, Fig. 422F. Experiments, however, prove that the maximum velocity is seldom found in the centre of the sectional area of the galleries, and sometimes diverges therefrom to a considerable extent. Lines of equal velocities are found in the cross-section of an airway somewhat irregularly. At times the curves of equal velocity may be distributed as shown at B, Fig. 422F, or ranged in various other ways.



A B

Fig. 422F.—Showing two Cross-sections of Arched Airways and the Curves of equal Air Velocity in them.

To ascertain the mean velocity, it has been customary to divide the airway into squares by means of strings stretched across. The velocity at each square is then taken by the anemometer. A single observation of the velocity is not relied on, but several are taken, each being of a certain duration. The arithmetical mean of all these observations has been usually taken to represent the mean velocity in the airway. More complicated calculations for determining the mean velocity of the air were made by M. Murgue in his very delicate experiments on the friction of air-currents. The anemometers used must be tested beforehand with the greatest care, or errors may arise from their use.

LOSS OF VENTILATING PRESSURE AND QUANTITIES OF AIR IN THE WORKINGS.

If, apart from shaft friction, the total ventilating pressure necessary to ventilate an intake and return airway of absolutely uniform sectional area and regularity of surface be, say 10 lbs. per square foot or 1·923 inch depression of water-gauge, then the loss of pressure in proceeding from the downcast shaft becomes less in exact proportion to the distance from such shaft, as shown in Fig. 422G. This proportionate loss, of course, holds good for any other stated pressure. If the intake be, say 228 yards long, the return being the same length with the connection 22 yards the total length is 478 yards, and at regular distances of 47·8 yards the pressure becomes 1 lb. less as shown in the sketch. If we assume a size for the gallery, we may ascertain the volume of air circulating. For an airway 6 feet by 6 under the given conditions, and using the formula $v^2 = \frac{pa}{ks}$ or $v = \sqrt{\frac{pa}{ks}}$ if we substitute values, we have $v = \sqrt{\frac{10 \times 36}{(\cdot 0000000217) \times (6 \times 4 \times 478 \times 3)}} = 694\cdot 3$ feet per minute as the mean velocity, and the quantity $q = 36 \times 694\cdot 3 = 24{,}995$ cubic feet per minute. If a greater or less quantity of air be now produced, the pressure will change according to the square of the mean velocity of the air. Thus, if the velocity be reduced one-half, the volume of air circulating becomes $\frac{24{,}995}{2} = 12{,}497$ cubic feet per minute, but the resistance then is only $\left(\frac{1}{2}\right)^2$ or $\frac{1}{4}$th of what it was previously, and the pressure per square foot becomes $\frac{10}{4} = 2\cdot 5$ lbs.

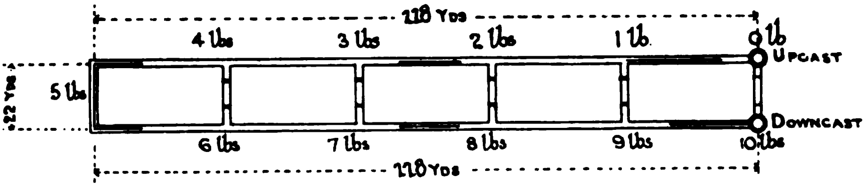


Fig. 422G.—Illustrating the Diminution of Pressure in the Air in Underground Passages between Downcast and Upcast Shafts.

The foregoing is an example of a simple nature with only a single current circulating, such as is only likely to be found in a newly-opened mine.

Let us now take the case of a mine having two splits of air of equal length from a certain distance in, as shown in Fig. 422H. Here the whole volume of air entering the downcast shaft passes to A, where it divides into two currents, one proceeding to B, and then returning to the upcast, the other passing round C and rejoining the first-named split at D, from whence the re-united volumes pass to the upcast shaft. If the sectional areas be exactly the same throughout these air-courses, and they present the same outline of surface to the air, then, being equal in length, they will, if level, have equal quantities of air circulating in each split. A slight modification, however, will ensue if their inclinations differ.

On the assumption that both are level, and that the ventilating pressure necessary to overcome the resistances between the bottom of the downcast and the bottom of the upcast is 10 lbs., we may calculate its diminution inwards as follows: Let us assume that the airway measures 6 feet by 6 feet, and of the length marked on the sketch. The air at A divides into two equal volumes, its velocity from the downcast to A, 78 yards, and from D to the upcast, 72 yards, together making 150 yards, is double that of the split currents. Each split measures 150 + 22 + 156 yards = 328 yards, and is subject to a common pressure. If 1 represent the value of v for the airway A B D, then 2 represents its value in the 150 yards between the splits and the shafts. Then, since $p = \frac{k\,s\,v^2}{a}$, the proportion of the total of 10 lbs. pressure for each of the two sections of roadway may be found. The value of k and a in each section being the same their use may be discarded, and then p is in proportion to $s\,v^2$. Instead of s, however, we may substitute the lengths l, because the perimeters are precisely the same. After substituting values for these we have: As $150 \times 2^2 : 328 \times 1^2 = 600 : 328$. Add 600 and 328 = 928, as the total of the two relative pressures. If 10 lbs. be the actual pressure, that for the 150-yard length of airway is: As 928 : 600 :: 10 : 6·47 lbs., and for the 328-yard length of airway is: As 928 : 328 :: 10 : 3·53 lbs. That is the pressure necessary to overcome the resistances, in the 150-yard length is 6·47 lbs., and for the 328-yard length is 10 − 6·47 = 3·53 lbs. per square foot. The velocity of the air in the outer airway is $v = \sqrt{\frac{6{\cdot}47 \times 36}{({\cdot}0000000217) \times (6 \times 4 \times 150 \times 3)}} = 996{\cdot}92$ feet per minute, and its volume 996·92 × 36 = 35,890 cubic feet per minute, and as a check, v for each inner airway may be worked out thus—

$v = \sqrt{\frac{3{\cdot}53 \times 36}{({\cdot}0000000217) \times (6 \times 4 \times 328 \times 3)}} = 498$ feet per minute or half that of the outer airway.

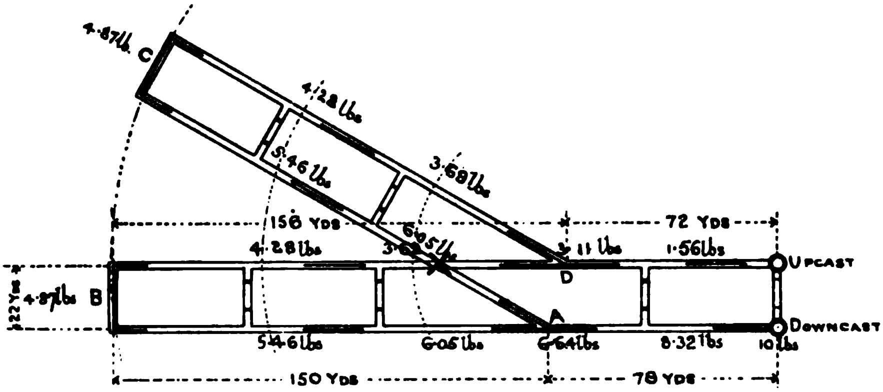


Fig. 422H.—Illustrating the Diminution of Pressure in the Air in Underground Passages between Downcast and Upcast Shafts.

The loss of pressure in the outer airway being 6·47 lbs., the resistance between the downcast and the point A may be found by proportion. Thus: As 150 : 78 :: 6·47 : 3·36 lbs. The diminution of the pressure from the point D to the upcast is as 150 : 72 :: 6·47 : 3·11 lbs., and midway between these points becomes 1·56 lb.

as shown in the sketch. A pressure of 10 lbs. at the downcast shaft becomes $10 - 3{\cdot}36 = 6{\cdot}64$ lbs. at the point A and midway is $\frac{10 + 6{\cdot}64}{2} = 8{\cdot}32$ lbs., as shown on the sketch. Any other points may be ascertained by proportion. The diminution of pressure in the airway, A B D, and in the airway A C D, being $3{\cdot}53$ lbs., if six equidistant points along their courses be chosen at 54⅔-yard intervals, there will be a regular loss of $\frac{3{\cdot}53}{6} = {\cdot}59$ lb. nearly at each, and the pressures are as marked in the sketch for those points.

In the case of a mine with six splits of unequal length of the same perimeter and sectional area dividing at the downcast, and re-uniting in the upcast, and therefore subject to a common pressure, it will be found that the shortest split gets the greatest and the longest the smallest proportion of the total quantity of air circulating. Assume these airways to be 6 feet square and of the following lengths:

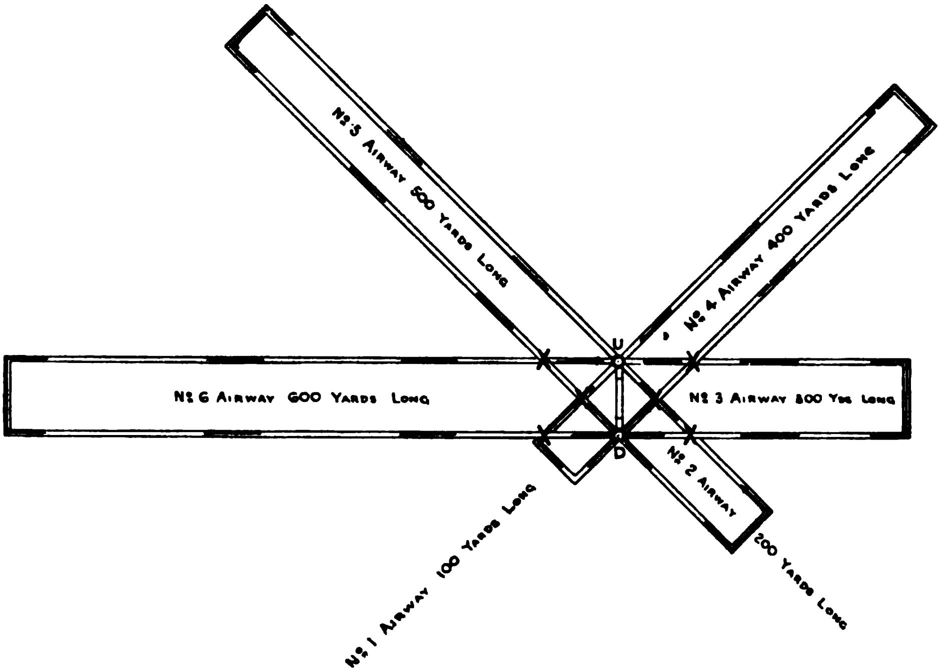

Nº 5 Airway 500 Yards Long
Nº 4 Airway 400 Yards Long
U
Nº 6 Airway 600 Yards Long
Nº 3 Airway 300 Yds Long
D
Nº 2 Airway 200 Yards Long
Nº 1 Airway 100 Yards Long

Fig. 422J.—Plan of Six Airways of Different Lengths.

No. 1 airway	100 yards.
2 „	200 „
3 „	300 „
4 „	400 „
5 „	500 „
6 „	600 „

as shown in Fig. 422J. The relative quantities in the airways will be the same as the relative velocities in them, because they are all of the same area. The velocities may be found by the formula $v = \sqrt{\frac{pa}{ks}}$. But since p and a and k

are the same in each airway, if we set a value of 1 on each of these, $v = \sqrt{\frac{1 \times 1}{1 \times s}} = \sqrt{\frac{1}{s}}$. The perimeters of the airways being alike in all cases, the result will not be affected if we substitute the length l for s, and it may be in yards, feet, or any other standard of measurement so long as that standard is used in all the airways. v then $= \sqrt{\frac{1}{l}}$, and the relative velocities in the six airways are:

For No. 1 airway $v = \sqrt{\frac{1}{100}} = \cdot 1$

" " 2 " $v = \sqrt{\frac{1}{200}} = \cdot 07071$

" " 3 " $v = \sqrt{\frac{1}{300}} = \cdot 057735$

" " 4 " $v = \sqrt{\frac{1}{400}} = \cdot 05$

" " 5 " $v = \sqrt{\frac{1}{500}} = \cdot 044721$

" " 6 " $v = \sqrt{\frac{1}{600}} = \cdot 040825$

Total ... ·363991

If we assume some quantity, say 100,000 cubic feet per minute as descending the downcast shaft, then we may use the above relative velocities to ascertain the distribution in the six airways, thus:—

	Cubic feet per minute.
Quantity in No. 1 airway, as ·363991 : ·1 :: 100,000 :	27,473
" " 2 " as ·363991 : ·07071 :: 100,000 :	19,427
" " 3 " as ·363991 : ·057735 :: 100,000 :	15,862
" " 4 " as ·363991 : ·05 :: 100,000 :	13,736
" " 5 " as ·363991 : ·044721 :: 100,000 :	12,286
" " 6 " as ·363991 : ·040825 :: 100,000 :	11,216
Total volume circulating ...	100,000

The value of p for the six airways may be found by applying the formula $p = \frac{ksv^2}{a}$ to either of them. But $p = \frac{ksv^2}{a}$ being equivalent to $p = \frac{ks\left(\frac{q}{a}\right)^2}{a}$ we have $p = \frac{ksq^2}{a^3}$. Applied to No. 1, and using the high co-efficient of friction of Atkinson, we have

$$p = \frac{(0\cdot 0000000217) \times (100 \times 3 \times 6 \times 4) \times (27{,}473)^2}{36^3}$$

$$p = \frac{0\cdot 0000000217 \times 7{,}200 \times 27{,}473^2}{36^3} = 2\cdot 5275 \text{ lbs. per sq. foot.}$$

If we assume each shaft to be 15 feet in diameter, lined with masonry throughout, and having a smooth surface, and say 300 yards deep, we may estimate the pressure in overcoming the shaft resistances. M. Murgue's value for the co-efficient of friction for galleries lined with masonry is as low as ·00033 inch of water, or ·001716 lb. per square foot. We will, however, to be on the safe side, take the co-efficient of friction for shafts at ·01 lb. per square foot. Then for the two shafts we get

$$p = \frac{(\cdot 00000001) \times (15 \times 3{\cdot}1416 \times 300 \times 2 \times 3) \times (100{,}000)^2}{176{\cdot}715^3}$$

$$p = \frac{\cdot 00000001 \times 84{,}823 \times 100{,}000^2}{176{\cdot}715^3}$$

$$p = 1{\cdot}537 \text{ lb. per square foot.}$$

The pressure, then, required to circulate 100,000 cubic feet of air per minute through the shafts and six airways is 2·5275 + 1·537 = 4·0645 lbs. per square foot. That is the pressure to overcome the friction; that necessary to create the velocity being so slight that its consideration may be left out altogether. If, now, we double the mean velocity of the air in the shafts, there will be a corresponding increase in the mean velocity of the air in each split, and the total quantity of air circulating becomes 200,000 cubic feet per minute. As p varies according to v^2, we should have 2^2, or four times the pressure, or 4·0645 × 4 = 16·258 lbs.

It is very rarely, if ever, that there is the uniformity of sectional area in our underground galleries shown in Figs. 422G, 422H, 422J, and the diminution in pressure is more or less irregular over equal distances, in accordance with the changes in area. Usually the air would not be allowed to split freely, as shown in Fig. 422J; if the circumstances of the mine were such as to make it desirable to have equal quantities going into the splits, it would be necessary to fix regulators in the five shorter splits. The effect of regulators will be dealt with later.

If we take a case of four splits of air, of which two are in an upper and two in a lower seam of coal, as shown in Fig. 422K, the air entering the downcast shaft may, of course, be regulated in each split to suit its requirements. If there are no regulators placed in the airways, and the air is free to flow unobstructedly in each, the relative quantities passing along each split will depend upon circumstances. If the upper seam be very near the surface, and the lower be at a great depth, nearly all the air entering the downcast may divide into the two splits of the upper seam, leaving the two splits in the lower seam insufficiently ventilated. If the airways in the upper seam be short and of large sectional area, and those in the lower be long, small, and rugged, the same result would follow. If, however, the two seams are only separated by a few yards of intervening strata, and both are at a considerable depth from the surface, it is probable that measurable currents of air will circulate in each split. But it is possible, if the lower seam airways are large and short, compared with those of the upper, for the greater quantity of air then to flow in the lower seam.

Supposing the downcast A, and upcast B, in Fig. 422K, to be each 15 feet in diameter and 300 yards deep to the upper seam, the lower being 100 yards deeper, or 400 yards from the surface, and that 100,000 cubic feet of air per minute enter the downcast shaft, as in the last example. The pressure necessary merely to overcome the shaft resistances to the upper seam is 1·537 lb. as calculated in that example. Now, if air be allowed to flow unrestrainedly into the upper and lower seams, then the quantity in each will depend upon the pressure, and this will be influenced by the relative lengths of splits in each seam, their sectional area, etc. The pressure of the upper seam will be that of the lower, plus that necessary to overcome the shaft resistances between the seams.

Let us assume that there are two splits, each 500 yards long, in the upper seam, while the two splits in the lower seam are each 600 yards long, all being 7 feet by 7 in size. We cannot calculate the pressure necessary to overcome the shaft friction between the two seams, because we do not know the volume of air which would flow to the lower seam. We may, however, be sure it will be less than half of the

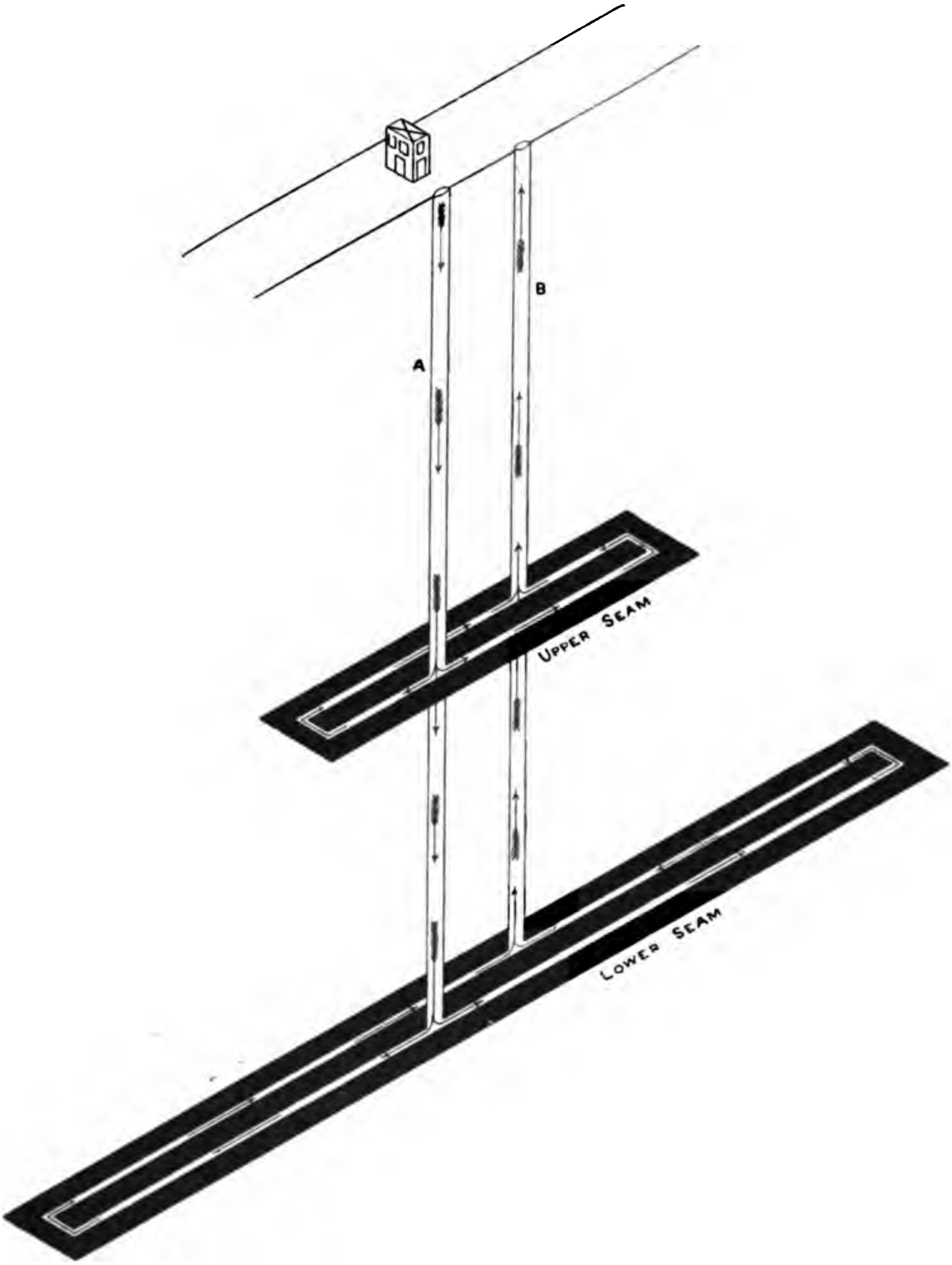


Fig. 422K.—Illustrating the Diminution of Pressure in the Air in the Shafts, and the effect of Splitting into an Upper and a Lower Seam.

volume in the upper portion of the shafts. Let us assume that a volume of 44,180 cubic feet per minute passes through the two splits in the bottom seam, that is 22,090 cubic feet in each, and if this quantity is not the proper proportion we can afterwards adjust it. Then the value of p for the split airways is

$$p = \frac{({\cdot}0000000217) \times (4 \times 7 \times 3 \times 600) \times (22{,}090)^2}{49^3} = 4{\cdot}5362 \text{ lbs.};$$

that for the two shafts between the two seams is

$$p = \frac{({\cdot}00000001) \times (47{\cdot}124 \times 100 \times 3 \times 2) \times (44{,}180)^2}{176{\cdot}715^3} = {\cdot}1 \text{ lb.}$$

The pressure then influencing the flow of air into the upper seam is

4·5362 lbs.
·1
4·6362 lbs. per square foot.

The velocity in each split of the upper seam arising from this pressure is

$$v = \sqrt{\frac{4{\cdot}6362 \times 49}{({\cdot}0000000217) \times (7 \times 4 \times 500 \times 3)}} = 499{\cdot}25$$ feet per minute, and therefore the volume in each split is $499{\cdot}25 \times 49 = 24{,}463$, or, together, 48,926 cubic feet per minute.

The total volume of air in motion in the shafts above the level of the upper seam therefore is

44,180
48,926
93,106 cubic feet per minute.

The pressure necessary to overcome the shaft friction above the level of the upper seam for this quantity is

$$p = \frac{({\cdot}00000001) \times (47{\cdot}124 \times 300 \times 3 \times 2) \times (93{,}106)^2}{176{\cdot}715^3} = 1{\cdot}3324 \text{ lb.}$$

Or as $100{,}000^2 : 93{,}106^2 :: 1{\cdot}537 : 1{\cdot}3324$ lb.

It has been shown that for 100,000 cubic feet of air per minute to circulate in the upper portion of the shafts it requires a pressure of 1·537 lb., or for 93,106 cubic feet it requires a pressure of 1·3324 lb. In the former case the pressures influencing the flow of air in the two seams must be correspondingly increased.

The pressure for the whole of the mine when 93,106 cubic feet per minute circulate is

4·6362
1·3324
5·9686 lbs. per square foot.

Since p varies as v^2 or q^2, the pressure for the whole mine for the increased volume is as $93{,}106^2 : 100{,}000^2 :: 5{\cdot}9686 : 6{\cdot}8852$ lbs.

The water-gauge of the whole mine, including the shafts, is $\frac{6{\cdot}8852}{5{\cdot}2} = 1{\cdot}3241$ inch.

The pressure influencing
the top seam is as 5·9686 : 6·8852 :: 4·6362 5·348 lbs.
„ bottom „ „ 5·9686 : 6·8852 :: 4·5362 5·2326 lbs.

and the quantities of air — Cubic feet per minute.

in the top seam $= \sqrt{4\cdot6362} : \sqrt{5\cdot348} :: 48{,}926 : 52{,}549$

„ bottom „ $= \sqrt{4\cdot5362} : \sqrt{5\cdot2326} :: 44{,}180 : 47{,}451$

or the increased volumes may be more simply obtained by direct proportion thus:

as 93,106 : 100,000 :: 48,926 : 52,549 top seam air

93,106 : 100,000 :: 44,180 : 47,451 bottom „

Total 100,000 cubic feet per minute.

It is impossible to calculate except approximately the ventilating pressure required for a large mine, because the galleries have frequent contractions and enlargements, instead of the uniformity of sectional area in the example just worked. Even where the area keeps approximately the same, there is great irregularity in the ruggedness of the sides, owing to greater or less intervals between posts, and the unevenness of floor and roof, all of which exercise their influence on the friction of a passing current. There are, moreover, side openings with leakages of air, and constant changes owing to the advance of the workings, the faces of such workings having irregular shapes.

In considering the design and size of fan, however, for a given mine, it is important that the expected volume of air to be circulated at a certain water-gauge be stated, the function of the fan being to pass air under the circumstances existing at that mine. As these, at best, can only be known approximately, allowance must be made for the possible contingencies of a higher water-gauge and a greater volume.

Assuming the requirement of 100,000 cubic feet of air per minute, shown in Fig. 422K, to be the utmost ever likely to be necessary, with 2·35 inches of water-gauge, we may proceed to ascertain the dimensions of the fan. The weight of a cubic foot of air at 60° Fahr., being ·0766 lb. and that of a cubic foot of water being 62·425 lbs., the one is $\frac{62\cdot425}{\cdot0766} = 815$ times heavier than the other, and, therefore, an inch of water-gauge is equal to 815 inches of ventilating column. $815 \times 2\cdot35 = 1{,}915$ inches and $\frac{1{,}915}{12} = 159\frac{1}{2}$ feet, the ventilating column. The theoretical water-gauge due to a perfect fan, is double the height due to the tangential velocity. The correct rule for ascertaining the pressure in head of air of a fan is $h = \frac{v^2}{g}$.

Substituting for h in the example we have:—

$159\frac{1}{2} = \frac{v^2}{32\cdot2}$ $\therefore v = \sqrt{159\frac{1}{2} \times 32\cdot2} = 71\cdot66$ as the velocity in feet per second of the circumference of the fan, or $71\cdot66 \times 60 =$ say 4,300 feet per minute. The next point is to decide on a suitable speed for the fan to work at. As it must work continuously, a moderate speed should be aimed at, as this will allow a margin for increase of speed in case of emergency. The speed for this may safely be fixed at 100 revolutions per minute, $\frac{4{,}300}{100} = 43$ feet circumference of the fan, and $\frac{43}{3\cdot1416} = 13\cdot69$ feet diameter of fan. The width of the fan to give the volume of 100,000 cubic feet of air per minute must now be considered. The effective capacity of the fan must be such as to give the required volume of air, and to obtain this allowance must be made by a deduction of the inlet area. The law of the *equivalent orifice* is used to determine the dimensions of the inlet of a fan. By using the formula for the equivalent orifice we can ascertain the size of opening in a plate of thin metal to pass a given quantity of air with a given water-gauge. The volume of air passing through this opening depends upon the amount of water-gauge. The equivalent orifice is the measure of resistance of the whole mine, corresponding to the measure obtained by using the water-gauge; the greater the water-gauge, the less will be the area of the opening.

The formula for this area of orifice is

$$a = \frac{0{\cdot}388q}{\sqrt{h}} \therefore q = \frac{a\sqrt{h}}{{\cdot}388} \text{ and } h = {\cdot}1505\left(\frac{q}{a}\right)^2$$

where a = opening in thin plate in square feet;
q = quantity of air in thousands of cubic feet per minute, passing through the opening;
h = the ventilating pressure in inches of water-gauge, required to overcome the resistance of the mine.

This formula, with slight modification, is as deduced by M. Daniel Murgue, and assumes that the co-efficient for the *vena contracta* = 0·65, and that the weight of a cubic foot of air is ·0766 lb.

It is necessary to understand clearly what the *vena contracta* is, to assist in which the following explanation is offered.

If an open-topped vessel be filled with water, and an aperture be made in its side, through which the water may be discharged, the quantity issuing depends upon the velocity of the efflux multiplied by the area of the aperture in square feet. But the velocity of efflux is obtained by multiplying the theoretical velocity due to the head of water, from surface of water to centre of orifice, by a certain co-efficient which differs for different orifices. Fig. 422L, shows vessels A, B, C, D, and E, all having different orifices, each exercising an influence on the discharge of the water. In some orifices the water nearly fills the side tubes when being discharged, and there is little loss from what is called the *vena contracta*, or contracted vein. In the case of an orifice in a thin plate, the actual current through the opening does not occupy the whole sectional area of the aperture, because the currents converge at the opening and impinge upon each other. The issuing water is thus kept to a central stream within the aperture surrounded by an air space between it and the perimeter of the aperture. The ratio of this sectional area of the central stream to the sectional area of the orifice through which it passes, has been proved by some experiments to be as 65 is to 100, though slightly different in others. These figures apply only to an orifice in a thin plate.

The same physical laws which control the flow of fluids through openings are applied to air. The pressure of a certain head causes the flow of air through the aperture with a velocity of efflux depending upon the different orifices. Where the discharge is through an orifice in a thin plate the co-efficient for the vena contracta is taken at ·65. In other words, the quantity of air passing through the opening is not equal to $a \times v$, but is taken as being equal to ${\cdot}65a \times v$, where a is the area of opening and v the velocity of the current through it.

If, now, we apply the formula for the determination of the equivalent orifice to the case under consideration, we have

$$a = \frac{{\cdot}388 \times 100}{\sqrt{2{\cdot}35}} = 25{\cdot}31 \text{ square feet area}$$

$$\text{and } \sqrt{\frac{25{\cdot}31}{{\cdot}7854}} = 5{\cdot}68 \text{ feet diameter of opening in the plate.}$$

We may test the accuracy of the formula for the equivalent orifice by this example. Here a water-gauge of 2·35 inches is equivalent to a ventilating column of 159½ feet as shown before, because $\frac{815 \times 2{\cdot}35}{12} = 159\frac{1}{2}$. The velocity in feet per second due to such a column is $8·02\sqrt{159·5} = 101·28$, or per minute it is $101·28 \times 60 = 6,077$. Now if the area through which air passes at this velocity is 25·31 square feet, then the total quantity passing would be $6,077 \times 25·31$. Inasmuch, however, as it has been shown that only 65 per cent. of the whole area of the orifice is taken up by the flowing air, we have $\frac{25·31 \times 65}{100} = 16·452$ square feet as the area of that portion of the orifice through which air issues at the rate of 6,077 feet per minute; $6,077 \times 16·452 = 99,979$ cubic feet per minute as against the 100,000 actually passing and from which the equivalent orifice, 25·31 square feet, is calculated.

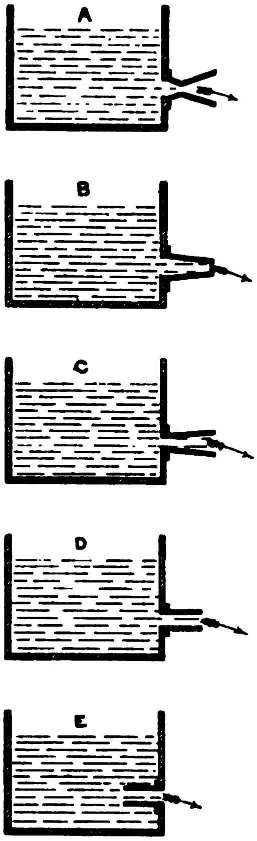


Fig. 422L.—Discharge of Water from Orifices.

It is customary to allow twice the inlet area as calculated from the rule for the equivalent orifice, because the inlet to a fan is not a hole in a thin plate, and considerable allowance is made for additional frictional resistances within the fan. The inlet area may thus be a single one of $2 \times 25·31$ square feet area, or there may be a double inlet, one on each side of the fan, and each being 5·68 feet in diameter. Certain types of large fans run more evenly with a double inlet because a more regular supply is thus ensured. Assuming we have the double inlet, then to find the effective area, that is the area of the fan, 13·69 feet in diameter less the area of the inlet 5·68 in diameter or $147·14 - 25·31$ = say 122 feet effective area. Each foot in width of this fan then will produce an effective volume of 122 cubic feet. The quantity to be discharged per minute is 100,000 cubic feet and the number of revolutions 100, $\frac{100,000}{100}$ = say 1,000 cubic feet per revolution $\frac{1,000}{122}$ = fully 8 feet. It is generally considered that a good fan will discharge one-half its effective capacity each revolution. Experiments have, however, proved that many good fans discharge much less, and this subject needs careful investigation. In practice fans are usually made having a certain ratio between their diameters and widths, and the result here obtained may be slightly modified. In the Waddle fan, which receives air only on one side, the diameter of the inlet is about one-third the diameter of the whole fan.

If the speed of the fan be increased from 100 to 150 revolutions per minute the water-gauge will be increased 2·25 times, and will then read $2·35 \times 2·25 = 5·29$ inches, but the volume then circulating will be 150,000 cubic feet per minute.

As another instance to show how we may ascertain approximately the dimensions of a large ventilating fan, let us assume that the volume of air to be circulated is 300,000 cubic feet per minute with a water-gauge of 4·5 inches.

For a water-gauge of 4·5 inches we should have a ventilating column of

$$\frac{815 \times 4·5}{12} = 305·62 \text{ feet.}$$

The tangential velocity of the fan due to this water-gauge is $v = \sqrt{305·62 \times 32·2}$ = 99·2 feet per second, or $99·2 \times 60 = 5,952$ feet per minute.

A speed of anything under 100 revolutions is considered within moderate limits for a large fan; if we fix it at 60 we are allowing a margin for increase. Then $\frac{5{,}952}{60} = 99{\cdot}2$ feet circumference of fan, and $\frac{99{\cdot}2}{3{\cdot}1416} = 31{\cdot}58$ feet diameter of fan. To ascertain the width we will first see what the dimensions of the inlet must be. The area of the equivalent orifice $a = \frac{{\cdot}388 \times 300}{\sqrt{4{\cdot}5}} = 54{\cdot}87$ square feet.

$$\sqrt{\frac{54{\cdot}87}{{\cdot}7854}} = 8{\cdot}36 \text{ feet diameter of opening in the plate.}$$

The effective area of the fan then is $(31{\cdot}58^2 \times {\cdot}7854) - 54{\cdot}87 = 783{\cdot}14 - 54{\cdot}87$ say 728 square feet. Each foot in width of the fan has an effective volume of 728 cubic feet. The quantity per minute, being 300,000 cubic feet, and the number of revolutions 60, we have $\frac{300{,}000}{60} = 5{,}000$ cubic feet discharged at each revolution of the fan, $\therefore \frac{5{,}000}{728} = 6{\cdot}868$, or say 7 feet. This fan should have a double inlet, if of a type to admit it, to allow for frictional resistances within the fan, and it would probably be made somewhat wider than 7 feet, because even a good fan only discharges about half of its effective capacity at each revolution.

The equivalent orifice of a mine to pass 300,000 cubic feet per minute with a water-gauge of 4·5 inches has been shown to be 54·87 square feet. This probably is the extreme equivalent orifice to be found at any mine in the United Kingdom, and is never reached on the continent of Europe, where it is oftener under 30 square feet than over. The average equivalent orifice for English mines is said to be only about 20 square feet. It differs in different mines, and varies from time to time in the same mine.

It can readily be seen, however, that it is absurd to ask a maker to supply a fan to produce an extremely large volume of air at a low water-gauge. If, for instance, he were asked for a fan capable of producing 350,000 cubic feet per minute at a water-gauge of 3 inches. Then if the equivalent orifice be worked out according to these particulars, it will be found that the result does not represent the condition of any coal-mine whatever. It is very rarely that the equivalent orifice of a mine is so great as 55 square feet. The condition of the mine would prevent 350,000 cubic feet of air per minute from circulating, when creating a vacuum of 3 inches. The conditions most favourable to good results from a fan are airways of large sectional area, and where the mine is so divided into districts that each separate split receives its required quantity of air without the use of regulators. The effect of placing regulators will be shown hereafter. Where the mine condition is known and a fan supplied with a guarantee to produce a certain volume with a certain water-gauge, and after its erection it is found to give the stipulated quantity only at a higher water-gauge, it is likely that the dimensions of the fan make it impossible for so much air to pass through it at the stipulated water-gauge, because its orifice of passage is too small.

Again, if a properly proportioned fan be set to work where there is a leakage in the fan drift near the fan, which thus gets an unlimited supply of fresh air, the efficiency of the fan will be greatly impaired, and a lower water-gauge and reduced quantity result with the same fan speed as that necessary for the ventilation of the mine.

What is called the manometrical efficiency is the proportion between the theoretical water-gauge of a fan and the actual water-gauge. Suppose, for instance, a fan of 30 feet in diameter, running at 60 revolutions per minute. The velocity of its circumference in feet per second is

$$\frac{30 \times 3{\cdot}1416 \times 60}{60} = 94{\cdot}25.$$

Then by the formula $\frac{v^2}{g}$ we have $\frac{94{\cdot}25^2}{32{\cdot}2} = 275{\cdot}86$ feet of ventilating column, $\frac{275{\cdot}86}{68} = 4$ inches, the theoretical water-gauge. If the actual water-gauge is 2 inches, then the manometrical efficiency is $\frac{2}{4} = {\cdot}5$.

Let us now take a case in which three seams of coal are being worked by means of three shafts shown in Fig. 422M. Here, A and C are downcast shafts, of which A is 300 yards deep to the top seam, while C passes through the middle seam at a depth of 400 yards, and reaches the bottom seam at a depth of 500 yards. B is the upcast for all the workings of the different seams, and is 20 feet in diameter, while A and C are each 15 feet in diameter. There are two splits from the bottom of A, one being 300 and the other 400 yards in length. None of the air entering shaft C is used to ventilate the top seam, but the first two splits are taken from it into the middle seam, one of which is 300 yards long and the other 200. The remainder of the air flows through shaft C to the bottom seam and is there split into two currents, one of which is 300 yards and the others 200 yards long to the upcast shaft. If all the airways are of a uniform sectional area and perimeter, being say 7 feet by 7, and no regulators be used, so that the air flows unrestrainedly through the mine, let us see what proportionate volumes will dispose themselves in the various splits.

Let us assume that a certain volume, say 43,508 cubic feet of air per minute, ventilates the two splits in the bottom seam. The relative quantities of air coursing airways subject to the same pressure may be found by the formula $R = \sqrt{\frac{a^3}{s}}$. Where the airways have the same perimeters and only differ in length, the value of the length l may be substituted for s. The lengths of airways in the bottom seam are 300 and 200 yards.

The relative quantities for the splits then are

$$\sqrt{\frac{49^3}{300}} = 19{\cdot}803, \text{ and}$$

$$\sqrt{\frac{49^3}{200}} = 24{\cdot}254$$

$$\overline{\underline{44{\cdot}057}}$$

The actual volumes going into the bottom splits then are—

As $44{\cdot}057 : 19{\cdot}803 :: 43{\cdot}508 : 19{,}556$
and „ $44{\cdot}057 : 24{\cdot}254 :: 43{,}508 : 23{,}952$

$\overline{43{,}508}$ cubic feet of air per minute.

The pressure necessary to overcome the friction of these airways may be found for either split, since it is the same in each, thus:—

$$\text{For the longer } p = \frac{({\cdot}0000000217) \times (7 \times 4 \times 300 \times 3) \times (19{,}556)^2}{49^3} = 1{\cdot}778 \text{ lb.,}$$

$$\text{or for the shorter } p = \frac{({\cdot}0000000217) \times (7 \times 4 \times 200 \times 3) \times (23{,}952)^2}{49^3} = 1{\cdot}778 \text{ lb.}$$

The pressure necessary to overcome the frictional resistances in passing 43,508 cubic feet per minute down the portion of shaft C between the middle and bottom

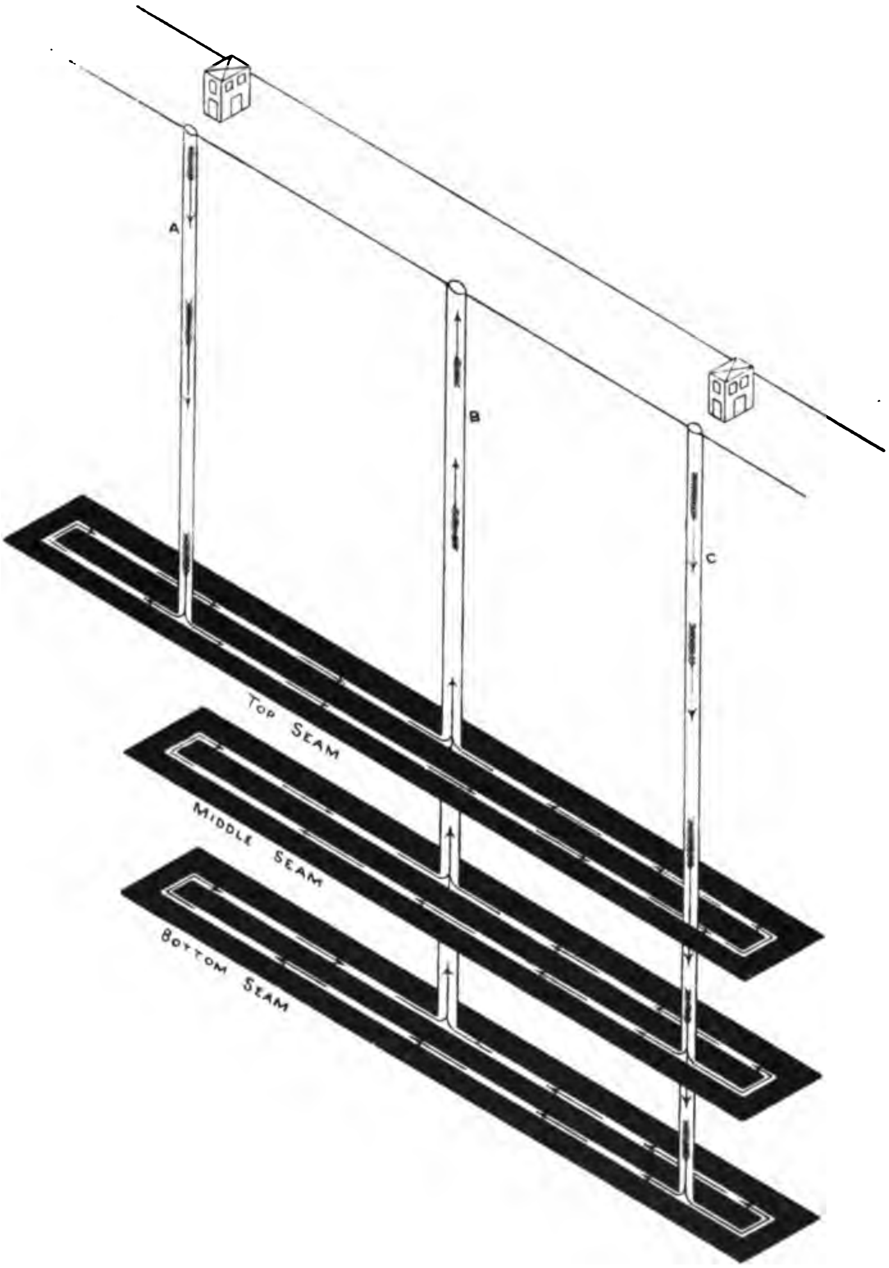


Fig. 422M.—Illustrating the Diminution of Pressure in the Air in the Shafts and the Effect of Splitting into Different Seams.

seams, the shaft being 47·124 feet in circumference and 176·715 square feet area, is

$$p = \frac{(\cdot 00000001) \times (47\cdot 124 \times 100 \times 3) \times (43{,}508)^2}{176\cdot 715^3} = \cdot 048493 \text{ lb.}$$

The area of a 15-foot diameter shaft being 176·715 square feet, and that of a 20-foot diameter shaft being 314·16, the relative velocities of the air in the two shafts between the middle and bottom seams is as 176·715 : 314·16 or as 1 : 1·778, so that if the velocity of the air in the 20-foot shaft be 1, that in the 15-foot shaft will be 1·778. Since $p = \frac{ksv^2}{a}$ and k is common to both shafts, the relative pressures are as $\frac{sv^2}{a}$, or as $\frac{47\cdot 124 \times 1\cdot 778^2}{176\cdot 715} : \frac{62\cdot 832 \times 1^2}{314\cdot 16}$ = as ·8428 : ·2 or as 1 : ·2373. The actual pressure for the 15-foot shaft has been found to be ·048493 lb., and therefore that for the 20-foot shaft is as 1 : ·2373 :: ·048493 : ·011507 lb. The pressure absorbed in overcoming the frictional resistances of shafts B and C between the bottom and middle seams is

·048493
·011507

·06 lb. per square foot.

The pressure then influencing the flow of air in the middle seam is this ·06 lb. absorbed in the shafts plus that in overcoming the bottom seam or 1·778 lb.

Thus ·06
1·778

1·838 lb. per square foot as the pressure in the middle seam splits, and we have to ascertain the air currents such pressure would be capable of producing there. We may now apply the formula $v = \sqrt{\frac{pa}{ks}}$ to ascertain the velocities of the air in the two middle seam splits, thus:—

For the longer $v = \sqrt{\frac{1\cdot 838 \times 49}{(\cdot 0000000217) \times (7 \times 4 \times 300 \times 3)}} = 405\cdot 82$ feet per min.

and for the shorter $v = \sqrt{\frac{1\cdot 838 \times 49}{(\cdot 0000000217) \times (7 \times 4 \times 200 \times 3)}} = 497\cdot 03$ or as $\sqrt{200} : \sqrt{300} :: 405\cdot 82 : 497\cdot 03$ feet per minute, and the respective volumes of air flowing are 405·82 × 49 = 19,885, and 497·03 × 49 = 24,355 cubic feet per minute. The air flowing in the middle seam then is

In the longer split 19,885 cubic feet per minute.
„ „ shorter „ 24,355 „ „ „ „

Total 44,240 „ „ „ „

This result may be tested by the relative or proportional quantities for the two splits in the middle seam. The proportion is the same as in the case of the two splits in the bottom seam.

Thus for the 300-yard airway as 44·057 : 19·803 :: 44,240 : 19,885 cubic feet per minute, and for the 200-yard airway as 44·057 : 24·254 :: 44,240 : 24,355 cubic feet per minute.

The total volume of air then entering shaft C, and also passing up shaft B from

the middle to the level of the top seam, is 43,508 cubic feet in the bottom seam,
and 44,240 ,, ,, ,, middle ,,

Making together 87,748 ,, ,, per minute.

The pressure necessary to overcome the frictional resistances in shaft C from the surface to the middle seam is

$$p = \frac{({\cdot}00000001) \times (47{\cdot}124 \times 400 \times 3) \times (87{,}748)^2}{176{\cdot}715^3} = {\cdot}789 \text{ lb.},$$

and for the shaft B between the middle and top seams

$$p = \frac{({\cdot}00000001) \times (62{\cdot}832 \times 100 \times 3) \times (87{,}748)^2}{314{\cdot}16^3} = {\cdot}0468 \text{ lb.}$$

The sum of the frictional resistances so far ascertained is as follows :—

	per sq. foot.
Shaft C, from the surface to the middle seam	·789 lb.
,, B, ,, the middle to the top seam	·0468 ,,
For overcoming shaft friction between bottom and middle seams and that of the different splits	1·838 ,,
	2·6738 lbs.

or a total of 2·6738 lbs. per square foot to the junction of the top seam air coming from shaft A, and which is influenced by this pressure.

Now a difficulty arises in determining the quantity of air which enters shaft A, because we do not know the proportion of the frictional resistances in the top seam splits and those in the shaft itself, the total of which amount to 2·6738 lbs. per square foot.

The relative quantities, however, which would pass into the top seam splits, may be found thus :—

$$R = \sqrt{\frac{a^3}{s}} \text{ or } R = \sqrt{\frac{a^3}{l}} \text{ as the airways are all 7 feet square,}$$

$$\therefore R = \sqrt{\frac{49^3}{400}} = 17{\cdot}15 \text{ for the longer,}$$

$$\text{and } R = \sqrt{\frac{49^3}{300}} = \underline{19{\cdot}803} \text{ for the shorter split.}$$

$$36{\cdot}953$$

Since the areas of the two airways are the same, the relative velocities are as 17·15 is to 19·803, or as 1 : $\frac{19{\cdot}803}{17{\cdot}15} = 1{\cdot}1547$. If the velocity in the longer split is 1, then that in the shorter is 1·1547. The quantity of air passing then is $(1 \times 49) + (1{\cdot}1547 \times 49) = 49 + 56{\cdot}58 = 105{\cdot}58$. If v = the velocity of the air in shaft A, then $176{\cdot}715\,v = 105{\cdot}58$, and $v = \frac{105{\cdot}58}{176{\cdot}715} = {\cdot}5975$. We have thus as the relative velocities in shaft A and the two top seam splits ·5975, 1, and 1·1547.

The relative pressures necessary for the shaft A and for the two splits in the top seam may now be found

$$\text{For the shaft, } p = \frac{({\cdot}00000001) \times (47{\cdot}124 \times 300 \times 3) \times ({\cdot}5975)^2}{176{\cdot}715} = {\cdot}0000008567 \text{ lb.}$$

For the longer airway,

$$p = \frac{({\cdot}0000000217) \times (7 \times 4 \times 400 \times 3) \times (1)^2}{49} = {\cdot}00001488 \text{ lb.}$$

or tested for the shorter—

$$p = \frac{({\cdot}0000000217) \times (7 \times 4 \times 300 \times 3) \times (1{\cdot}1547)^2}{49} = {\cdot}00001488 \text{ lb.}$$

The total relative pressure then is—

·0000008567
·00001488

·0000157367 lb. per square foot.

But the actual total pressure for shaft C, part of shaft B, and the associated splits has been shown to be 2·6738 lbs. when there is a volume of 87,748 cubic feet of air per minute flowing down the upper portion of shaft C.

Therefore the actual pressure in shaft A is—

As ·0000157367 : ·0000008567 :: 2·6738 : ·14557 lb.
and for the airways as ·0000157367 : ·00001488 :: 2·6738 : 2·52823 lbs.

The actual velocity of the air entering shaft A then is $\sqrt{\dfrac{{\cdot}14557 \times 176{\cdot}715}{({\cdot}00000001) \times (47{\cdot}124 \times 300 \times 3)}} = 246{\cdot}278$ feet per minute, and therefore the quantity of air in the shaft is 246·278 × 176·715 = 43,521 cubic feet per minute. The actual velocity in the longer airway is as ·5975 : 1 :: 246·278 : 412·2 feet per minute, and the quantity passing is 412·2 × 49 = 20,198 cubic feet per minute. The actual velocity in the longer airway may also be tested thus—

$$v = \sqrt{\frac{2{\cdot}52823 \times 49}{({\cdot}0000000217) \times (4 \times 7 \times 400 \times 3)}} = 412{\cdot}2 \text{ feet per minute as before.}$$

The actual velocity in the shorter airway is—

As ·5975 : 1·1547 :: 246·278 : 475·97 feet per minute, and the quantity passing therefore is 475·97 × 49 = 23,323 cubic feet per minute. The actual velocity in the shorter airway may also be tested thus—

$$v = \sqrt{\frac{2{\cdot}52823 \times 49}{({\cdot}0000000217) \times (4 \times 7 \times 300 \times 3)}} = 475{\cdot}97 \text{ feet per minute}$$

as obtained before. The addition of the quantities found to be passing in the splits should together be the same as that found to be the whole volume passing through the shaft, thus:—

Quantity in the longer split 20,198
„ „ shorter „ 23,323

Total 43,521 cubic feet per minute, which agrees exactly with the volume found to be passing in the shaft.

This quantity of 43,521 cubic feet joins the 87,748 cubic feet of air per minute from the middle and bottom seams in the shaft B, the united volumes from the top seam passing up the upcast. The pressure necessary to overcome the frictional resistances in shaft B to the passage of the enlarged volume of air from the top seam to the surface may now be ascertained.

The total quantity of air is 43,521 top seam air
87,748 middle and bottom seam air

131,269 cubic feet per minute.

$$\text{then } p = \frac{(\cdot00000001) \times (62\cdot832 \times 300 \times 3) \times (131{,}269)^2}{314\cdot16^3} = \cdot31426 \text{ lb.}$$

The pressure necessary to overcome the whole of the mine and shaft resistances then is—

2·6738 lbs.
and ·31426

2·98806 lbs. per square foot, and $\frac{2\cdot98806}{5\cdot2} = \cdot5746$ inch of water-gauge.

It would appear thus that under the conditions given the air would flow into each seam in nearly equal volumes. The shaft resistances between the two bottom seams are slight, and the airways being of similar lengths only a little less air would pass into the bottom seam than into the middle seam, the difference being a measure of the shaft resistances. Then in shaft A, the lessened shaft resistance compared with shaft C is near about balanced by the increased length of the top seam airways, which thus between them get nearly the same quantity of air as is circulating in the bottom seam.

Airways and shafts of large sectional area greatly assist the ventilation of the mine, as may be strikingly shown in the last example, if we take the diameter of shaft A to be only 10 feet in diameter and the top seam airways to be only 5 feet by 5.

The circumference of a 10-foot shaft is 31·416 feet and its area is 78·54 square feet.

The pressure influencing the circulation of the air in the top seam would be 2·6738 lbs. as before, but a great falling off in the quantity follows.

Thus the relative quantities which under the new conditions would flow in the top seam splits are:—

$$R = \sqrt{\frac{25^3}{400}} = 6\cdot25 \text{ for the longer}$$

$$\text{and } R = \sqrt{\frac{25^3}{300}} = 7\cdot2168 \text{ for the shorter.}$$

Since the airways have equal areas the relative velocities of air in them are as 6·25 : 7·2168, or as $1 : \frac{7\cdot2168}{6\cdot25} = 1\cdot1547$, being the same as those found for 7-foot square airways. The quantity of air passing then is $(1 \times 25) + (1\cdot1547 \times 25) = 25 + 28\cdot867 = 53\cdot867$. If $v =$ the velocity of the air in shaft A, then $78\cdot54\ v = 53\cdot867$ and $v = \frac{53\cdot867}{78\cdot54} = \cdot68585$. The relative velocities in shaft A and the two top seam splits are ·68585, 1, and 1·1547.

The relative pressures then must be—

For the shaft—

$$p = \frac{(\cdot00000001) \times (31\cdot416 \times 300 \times 3) \times (\cdot68585)^2}{78\cdot54} = \cdot0000016934 \text{ lb.}$$

For the longer airway—

$$p = \frac{(\cdot0000000217) \times (5 \times 4 \times 400 \times 3) \times (1)^2}{25} = \cdot000020832 \text{ lb.}$$

Or tested for the shorter—

$$p = \frac{(\cdot0000000217) \times (5 \times 4 \times 300 \times 3) \times (1\cdot1547)^2}{25} = \cdot000020832 \text{ lb.}$$

The total relative pressure then is—

·0000016934
·0000020832

·0000225254 lb. per square foot.

As the actual pressure for shaft C, part of shaft B, and the two lower seams splits is 2·6738 lbs., the actual pressure in shaft A is—

As ·0000225254 : ·0000016934 :: 2·6738 : ·201 lb.
and for the airways As ·0000225254 : ·0000020832 :: 2·6738 : 2·4728 lbs.

The velocity of air in shaft A, when the pressure necessary to overcome its frictional resistance is ·201 lb., is $\sqrt{\dfrac{{\cdot}201 \times 78{\cdot}54}{({\cdot}00000001) \times (31{\cdot}416 \times 300 \times 3)}} = 236{\cdot}3$ feet per minute, and the quantity in motion is 236·3 × 78·54 = 18,559 cubic feet per minute.

The velocities in the two splits now become—

In the longer, As ·68585 : 1 :: 236·3 : 344·53 feet per minute.
In the shorter, „ ·68585 : 1·1547 :: 236·3 : 397·83 feet per minute.

and the quantity of air in circulation is

In the longer airway, 344·53 × 25 = 8,613 cubic feet per minute
In the shorter „ 397·83 × 25 = 9,946 „ „ „ „

Total 18,559 „ „ „ „

If tested by the formula the velocities in the airways are

In the longer, $v = \sqrt{\dfrac{2{\cdot}4728 \times 25}{({\cdot}0000000217) \times (4 \times 5 \times 400 \times 3)}} = 344{\cdot}53$ ft. per min.

In the shorter, $v = \sqrt{\dfrac{2{\cdot}4728 \times 25}{({\cdot}0000000217) \times (4 \times 5 \times 300 \times 3)}} = 397{\cdot}83$ ft. per min.

It is true that with shaft A and the airways reduced in size there would now be a slight reduction of the pressure necessary in shaft B, due to the diminished volume of air in its upper portion, but it may be shown how comparatively unimportant this is, and how little it affects the water-gauge of the whole mine.

The total quantity of air in shaft B from the top seam upwards is

Top seam air 18,559 cubic feet per minute.
Middle and bottom seam 87,748 „ „ „ „

Making together 106,307 „ „ „ „

$$\text{then } p = \frac{({\cdot}00000001) \times (62{\cdot}832 \times 300 \times 3) \times (106{,}307)^2}{314{\cdot}16^3} = {\cdot}2061 \text{ lb.}$$

The pressure now produced to overcome the whole of the mine and shaft resistances becomes

2·6738 lbs.
·2061 „

2·8799 „ per square foot, and $\dfrac{2{\cdot}8799}{5{\cdot}2} = {\cdot}5538$ inch of water-gauge, or a difference of only ·5746 − ·5538 = ·0208 or say $\frac{1}{50}$th of an inch as compared with the water-gauge under the previous conditions. This decrease in the water-gauge is small and of little consequence compared to the falling off of the top seam air from the

43,521 cubic feet per minute previously calculated to the
18,559 „ „ „ „ for the later calculation or a

Difference of 24,962 „ „ „ „ being a decrease when the calculation

is made for a 10-foot shaft and 5-foot airways.

As a further illustration of the benefit to mine ventilation derived from making the shafts of large sectional area, let us assume that it is desired to abandon shaft A, and that the situation of shafts B and C are such that when a communication in the top seam is made with shaft C the split airways in this seam are of the same length as before.

An opening made from shaft C in the top seam would cause a general disturbance of the ventilating pressures in the various workings. Let us assume also that shaft C is enlarged to 20 feet in diameter to correspond with the size of the upcast and to compensate for the loss of shaft A.

If the same volume of air be taken to ventilate the bottom seam as previously, viz., 43,508 cubic feet per minute, the pressure in overcoming the friction in the splits is 1·778 lb. as before, and that in the upcast between the middle and bottom seams is also the same, viz., ·011507 lb. Instead of the ·048493 lb. in the downcast we should now have ·011507 lb., the same as in the upcast. These different pressures together make 1·778

·011507
·011507

1·801014 lb. per square foot as the pressure in the middle seam splits.

Previously it was 1·838, and as p varies as v^2, the velocity

In the longer airway is,

As $\sqrt{1\cdot838} : \sqrt{1\cdot801014} :: 405\cdot82 : 401\cdot72$ feet per minute, and

In the shorter airway is,

As $\sqrt{1\cdot838} : \sqrt{1\cdot801014} :: 497\cdot03 : 492\cdot0$ feet per minute,

and the respective volumes of air flowing are $401\cdot72 \times 49 = 19{,}684$ and $492 \times 49 = 24{,}108$ cubic feet per minute. The air flowing in the middle seam then is

In the longer split 19,684 cubic feet per minute.
„ „ shorter „ 24,108 „ „

Total 43,792 „ „

The air passing down shaft C from the top seam to the middle seam, and up shaft B from the middle to the top seam, is

43,508 cubic feet in the bottom seam.
43,792 „ „ middle „

Making together 87,300 „ per minute.

The pressure necessary to overcome the frictional resistances in shaft B between these two seams is

$$p = \frac{(\cdot00000001) \times (62\cdot832 \times 100 \times 3) \times (87{,}300)^2}{314\cdot16^3} = \cdot04633 \text{ lb.}$$

That for shaft C for a similar quantity of air for the same distance is the same, the areas and perimeters of the shafts being equal.

The sum of the frictional resistances now ascertained is as follows:—

In shaft C between the bottom and middle seams	·011507
„ „ „ „ „ middle and top „	·04633
„ „ B „ „ bottom and middle „	·011507
„ „ „ „ „ middle and top „	·04633
„ bottom seam airways	1·778
	1·893674 lbs. per sq. ft.

as the pressure in the top seam splits. Previously it was 2·52823 lbs. when the velocity of the air in the longer airway was 412·2 feet per minute and in the shorter 475·97. The pressure p varies in accordance with v^2, so that we now have as the velocities of the air

In the longer split As $\sqrt{2\cdot52823} : \sqrt{1\cdot893674} :: 412\cdot2 : 356\cdot74$ feet per minute.
„ „ shorter „ „ $\sqrt{2\cdot52823} : \sqrt{1\cdot893674} :: 475\cdot97 : 411\cdot92$ „ „ „

and the quantities passing therefore are

In the longer split	356·74 × 49 = 17,480	cubic feet per minute.
„ „ shorter „	411·92 × 49 = 20,184	„ „ „ „
	37,664	„ „ „ „

The total quantity of air entering shaft C and supplying the three seams is

In the top seam	37,664	cubic feet per minute.
„ „ middle „	43,792	„ „ „ „
„ „ bottom „	43,508	„ „ „ „
Total	124,964	„ „ „ „

The pressure necessary to overcome the frictional resistances in shaft C from the surface to the top seam is

$$p = \frac{(\cdot00000001) \times (62\cdot832 \times 300 \times 3) \times (124{,}964)^2}{314\cdot16^3} = \cdot2848 \text{ lb.}$$

The pressure in shaft B from the top seam to the surface is the same as in the case of shaft C over the same depth. The total of the mine pressure then is

In shaft C between the surface and top seam	·2848 lb.
„ „ „ „ „ top and middle seams	·04633
„ „ „ „ „ middle and bottom „	·011507
„ „ B „ „ bottom and middle „	·011507
„ „ „ „ „ middle and top „	·04633
„ „ „ „ top seam and the surface	·2848
„ bottom seam airways	1·778
Total	2·463274 lbs. per sq. ft.

and $\frac{2\cdot463274}{5\cdot2} = \cdot4737$ inch of water-gauge for a total volume of 124,964 cubic feet per minute as compared with the 131,269 cubic feet previously calculated for the three shafts when the water-gauge was ·5746 inch or equivalent to a pressure of 2·98806 lbs. per square foot.

In order to fairly compare the two cases let us increase the ventilation in the second case until the water-gauge reads the same as in the first. Then since p varies as v^2 or as q^2, we have

As $\sqrt{2\cdot463274} : \sqrt{2\cdot98806} :: 124{,}964 : 137{,}633$ cubic feet per minute.

so that with the same water-gauge, there would be an increased ventilation from a single 20-foot downcast compared with two 15-foot downcasts. It may be of interest to see how the total quantity of 137,633 cubic feet of air per minute would be divided in the different seams.

The increased pressure
In the bottom seam would be, As 2·463274 : 2·98806 :: 1·778 : 2·1568 lbs.
" " middle " " " " 2·463274 : 2·98806 :: 1·801014 : 2·1847 lbs.
" " top " " " " 2·463274 : 2·98806 :: 1·893674 : 2·2971 lbs.
and the consequent increased volumes of air in these seams are
In the bottom seam, As $\sqrt{1\cdot778}$: $\sqrt{2\cdot1568}$ :: 43,508 : 47,919
" " middle " " $\sqrt{1\cdot801014}$: $\sqrt{2\cdot1847}$ :: 43,792 : 48,232
" " top " " $\sqrt{1\cdot893674}$: $\sqrt{2\cdot2971}$ :: 37,664 : 41,482

Total 137,633 cub. ft. per min.

or the increased volumes may be more simply obtained by proportion, thus:—
In the bottom seam, As 124,964 : 137,623 :: 43,508 : 47,919
" " middle " " 124,964 : 137,623 :: 43,792 : 48,232
" " top " " 124,964 : 137,623 :: 37,664 : 41,482

Total 137,633 cub. ft. per min.

We may now show the result of the two calculations side by side for comparison.

Name of Seam.	With two 15-foot Downcasts, one 20-foot Upcast, and water-gauge of the whole mine and shafts, ·5746 inch.		With an Upcast and Downcast each 20 feet in diameter, airways and total water-gauge being the same as in the case of the three shafts, viz., ·5746 inch.	
	Pressure in lbs. per square foot.	Quantity of Air in cubic feet per minute.	Pressure in lbs. per square foot.	Quantity of Air in cubic feet per minute.
Top seam	2·6738	43,521	2·2971	41,482
Middle "	1·838	44,240	2·1847	48,232
Bottom "	1·778	43,508	2·1568	47,919
Total.........	2·98806	131,269	2·98806	137,633

It will thus be seen that so far as the total ventilation of the mine is concerned there would be an advantage under the given conditions, of having a single downcast 20 feet in diameter as compared with two such shafts each 15 feet in diameter. With the same total pressure, however, there would be a falling off in the quantity of air circulating in the top seam, if no regulators were placed, and larger volumes in the other seams. This is due to the re-distribution of pressures which must ensue if any such change in the shafts be carried out.

Instead of a colliery consisting of two downcast shafts and one upcast as shown in Fig. 422M, there may be two or three seams being worked with a single downcast and two upcast shafts. Fig. 422N shows such a colliery. In addition to these shafts there is a staple or blind pit used for ventilation between the No. 1 and No. 2 seams and marked with the letter D on the plan.

Supposing the downcast shaft A to be 20 feet in diameter and each upcast 15 feet. Shaft C is only sunk 300 yards to the No. 1 seam, while shafts B and A

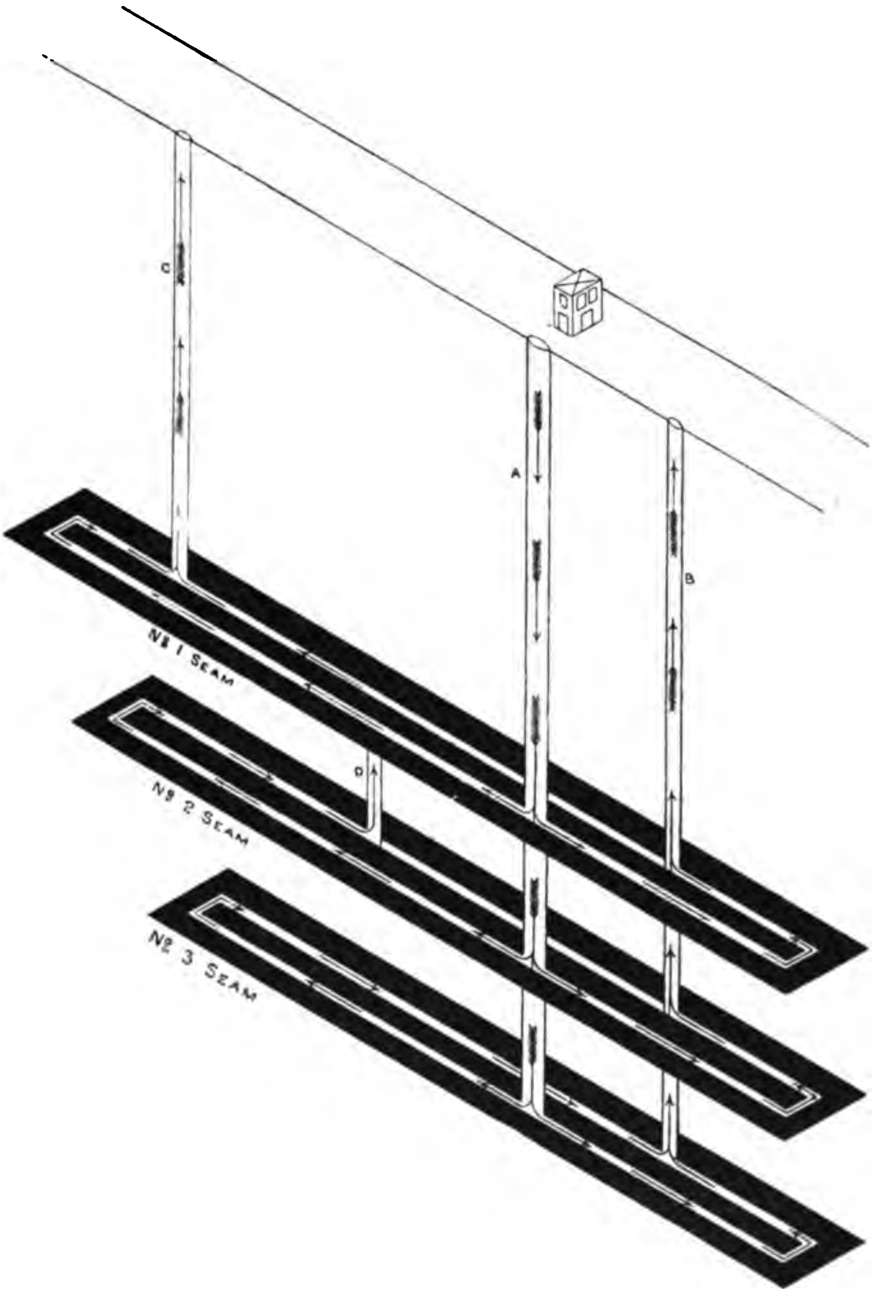


Fig. 422N.—Illustrating the Diminution of Pressure in the Air in the Shafts and the effect of Splitting into different Seams.

reach the No. 3 seam at a depth of 500 yards after passing No. 1 seam at 300 yards and No. 2 at 400 yards. There are two splits of air from shaft A into each seam. Of the two splits in the No. 1 seam, that towards shaft B, which we will call the east split, is 800 yards long to its junction with the upcast B. The other, or west split, is 1,000 yards long to the shaft C, where it joins the return air from a No. 2 seam split. The east split in the No. 2 seam is, say, 700 yards long to its junction with the upcast B, while the west split, after a course of 600 yards, in No. 2 seam, returns up the staple D to the No. 1 seam 100 yards above; the continuation of the return airway in that seam being 300 yards to the upcast C. The east split in No. 3 seam is 600 yards long and the west split 800 yards long to their junction with upcast B. All the airways are 6 feet high and 6 feet wide and the staple D is 8 feet in diameter, no regulators or checks of any description being placed in the airways.

It is plain that the distribution of the air will be affected by the relative ventilating pressures maintained in shafts C and B. The ventilating power at shaft C may be slackened at the same time that that at B is increased, or *vice versâ*, so that the relative volumes of air passing up the two shafts may be subject to continual change.

Let us assume that 30,000 cubic feet of air pass down the shaft A below No. 2 seam. Then the relative quantities in each split of No. 3 seam are

$$\sqrt{\frac{36^3}{600}} = 8{\cdot}8182$$

$$\sqrt{\frac{36^3}{800}} = 7{\cdot}6368$$

$$\underline{16{\cdot}455}$$

and the actual volumes are

In the east split, As 16·455 : 8·8182 :: 30,000 : 16,077
„ west „ „ 16·455 : 7·6368 :: 30,000 : 13,923
30,000

cubic feet per minute.

The value of p is

For the east split $\dfrac{({\cdot}0000000217) \times (6 \times 4 \times 600 \times 3) \times (16{,}077)^2}{36^3} = 5{\cdot}1932$ lbs.,

or for the west split $\dfrac{({\cdot}0000000217) \times (6 \times 4 \times 800 \times 3) \times (13{,}923)^2}{36^3} = 5{\cdot}1932$ lbs.

In shaft A, from No. 2 to No. 3 seam

$$p = \frac{({\cdot}00000001) \times (62{\cdot}832 \times 100 \times 3) \times (30{,}000)^2}{314{\cdot}16^3} = {\cdot}005471 \text{ lb.}$$

In shaft B, between No. 2 and No. 3 seams

$$p = \frac{({\cdot}00000001) \times (47{\cdot}124 \times 100 \times 3) \times (30{,}000)^2}{176{\cdot}715^3} = {\cdot}023057 \text{ lb.}$$

The pressure influencing the flow of air in the east split of the No. 2 seam is

$$\begin{array}{r} {\cdot}005471 \\ {\cdot}023057 \\ \underline{5{\cdot}1932} \\ \underline{5{\cdot}221728} \text{ lbs. per square foot} \end{array}$$

and the velocity in it therefore is

$$\sqrt{\frac{5{\cdot}2217 \times 36}{({\cdot}0000000217) \times (6 \times 4 \times 700 \times 3)}} = 414{\cdot}58 \text{ feet per minute}$$

and the quantity of air is 414·58 × 36 = 14,925 cubic feet per minute.

If we assume that a similar volume of air ventilates the west split in No. 2 seam, then the length of this aircourse being 600 yards plus 300 yards in the No. 1 seam, or 900 yards in all, besides its passage up the staple D,

$$p = \frac{(\cdot 0000000217) \times (6 \times 4 \times 900 \times 3) \times (414\cdot 58)^2}{36} = 6\cdot 7136 \text{ lbs.}$$

or, As 700 : 900 :: 5·2217 : 6·7136 lbs.

If the staple is lined with brickwork and presents a smooth surface to the passage of air, we may use the same co-efficient of friction as for shafts. Assume this is so, and p for the staple is

$$\frac{(\cdot 00000001) \times (25\cdot 132 \times 100 \times 3) \times (14{,}925)^2}{50\cdot 265^3} = \cdot 13225 \text{ lb.}$$

The pressure for the whole of the west split in No. 2 seam is

6·7136 lbs.
·13225 lb.
6·84585 lbs. per square foot.

The pressure necessary to overcome the frictional resistances in shaft A between the No. 1 and No. 2 seams, where the quantity passing is

14,925
14,925
30,000
59,850 cubic feet per minute is

$$p = \frac{(\cdot 00000001) \times (62\cdot 832 \times 100 \times 3) \times (59{,}850)^2}{314\cdot 16^3} = \cdot 021777 \text{ lb.,}$$

but the proportion of this for the west split on No. 2 seam only is, As 59,850 : 14,925 :: ·021777 : ·0054307 lb.

The pressure determining the quantity of air flowing in the west split in No. 1 seam then is

6·84585
·00543
6·85128 lbs. per square foot,

and the velocity this gives in the airway is,

$$\sqrt{\frac{6\cdot 85128 \times 36}{(\cdot 0000000217) \times (6 \times 4 \times 1{,}000 \times 3)}}, = 397\cdot 32, \text{ feet per minute.}$$

and the volume of air is 397·32 × 36 = 14,303 cubic feet.

We have then this 14,303 cubic feet of air per minute, and the 14,925 which returns up the staple D, uniting in shaft C, making a total of 29,228 cubic feet passing up shaft C. The pressure necessary to overcome the shaft resistance is

$$p = \frac{(\cdot 00000001) \times (47\cdot 124 \times 300 \times 3) \times (29{,}228)^2}{176\cdot 715^3} = 065654 \text{ lb.}$$

The volume of air in the east split of No. 2 seam is 14,925, which joins the 30,000 cubic feet from the No. 3 seam, the united volume of 44,925 cubic feet passing up shaft B between the No. 2 and No. 1 seam. The shaft pressure due to this quantity is

$$p = \frac{(\cdot 00000001) \times (47\cdot 124 \times 100 \times 3) \times (44{,}925)^2}{176\cdot 715^3} = \cdot 0517 \text{ lb.}$$

The pressure in shaft A, between No. 1 and No. 2 seams, was shown to be ·021777 lb., the proportion for the west split in No. 2 seam being ·0054307 lb., that for the east split is

$$\begin{array}{rl} & \cdot 021777 \\ \text{less} & \cdot 0054307 \\ \hline & \cdot 0163463 \text{ lb. per square foot.} \end{array}$$

The pressure affecting the flow of air in the east split of No. 1 seam is

$$\begin{array}{l} \cdot 0163463 \\ \cdot 0517 \\ 5\cdot 221728 \\ \hline 5\cdot 2897743 \text{ lbs. per square foot,} \end{array}$$

and the velocity of the air in this airway would be

$$v = \sqrt{\frac{5\cdot 2898 \times 36}{(\cdot 0000000217) \times (6 \times 4 \times 800 \times 3)}} = 390\cdot 33 \text{ feet per minute,}$$

and the quantity circulating in it is 390·33 × 36 = 14,052 cubic feet per minute.

The total volume of air in shaft B is

$$\begin{array}{l} 44{,}925 \\ 14{,}052 \\ \hline 58{,}977 \text{ cubic feet per minute passing up} \end{array}$$

from the No. 1 seam to the surface.

The value of p for this portion of the shaft is

$$\frac{(\cdot 00000001) \times (47\cdot 124 \times 300 \times 3) \times (58{,}977)^2}{176\cdot 715^3} = \cdot 26732 \text{ lb.}$$

The whole volume of air entering shaft A is the sum of the two volumes discharged from shafts B and C, viz., in shaft B 58,977

$$\begin{array}{rl} \text{in shaft B} & 58{,}977 \\ \text{,, ,, C} & 29{,}228 \\ \hline & 88{,}205 \text{ cubic feet per minute,} \end{array}$$

and the pressure necessary to overcome the friction in shaft A for such quantity from the surface to No. 1 seam is

$$p = \frac{(\cdot 00000001) \times (62\cdot 832 \times 300 \times 3) \times (88{,}205)^2}{314\cdot 16^3} = \cdot 14189 \text{ lb.}$$

The proportion of this pressure due to the ventilator at shaft C is

As 88,205 : 29,228 :: ·14189 : ·04702 lb.,

and to that at shaft B, ,, 88,205 : 58,977 :: ·14189 : ·09487 lb.

The total pressure exerted by the ventilating power at shaft C then is

$$\begin{array}{l} 6\cdot 85128 \\ \cdot 06565 \\ \cdot 04702 \\ \hline 6\cdot 96395 \text{ lbs. per square foot,} \end{array}$$

causing a circulation of 14,925 cubic feet per minute through the west split of No. 2 seam, and 14,303 cubic feet through the west split of No. 1 seam, with a total volume of 29,228 cubic feet in the shaft.

The total pressure exerted by the ventilating power at shaft B is

$$\begin{array}{l} 5\cdot 28977 \\ \cdot 26732 \\ \cdot 09487 \\ \hline 5\cdot 65196 \text{ lbs. per square foot,} \end{array}$$

producing a circulation of 30,000 cubic feet per minute in No. 3 seam, 14,925 cubic feet through the east split of No. 2 seam, and 14,052 cubic feet through the east split of No. 1 seam, with a total volume of 58,977 cubic feet per minute in the shaft.

It will be noticed that although the pressure for the workings ventilated to shaft B, inclusive of shaft friction, is less than that required for the workings ventilated to shaft C, inclusive of shaft friction, it produces a greater volume of air. This is partly due to the greater number of splits, and partly to their being shorter on the east side of shaft A. Splitting the air into different seams from the downcast shaft has the same beneficial effect on the ventilation of a mine as splitting into airways of one seam direct from the downcast shaft.

We may now increase the ventilating power in shaft B so as to make the ventilating pressure correspond with that in shaft C, that is, raise it from 5·65196 lbs. to 6·96395 lbs. By so doing we should increase the quantity of air in shaft B

As $\sqrt{5{\cdot}65196} : \sqrt{6{\cdot}96395} :: 58{,}977 : 65{,}465$ cubic feet per minute.

The air in No. 3 seam and in the east splits of the other seams would then be increased in proportion to the total increase thus:—

In No. 3 seam, As 58,977 : 65,465 :: 30,000 : 33,300 cubic feet per minute.
„ „ 2 „ „ 58,977 : 65,465 :: 14,925 : 16,567 „ „ „
„ „ 1 „ „ 58,977 : 65,465 :: 14,052 : 15,598 „ „ „

In No. 3 seam the new volume would be proportioned between the two splits thus:—

In the east split, As 16·455 : 8·8182 :: 33,300 : 17,845
„ „ west „ „ 16·455 : 7·6368 :: 33,300 : 15,455

33,300 cubic feet per minute.

The increased pressure here would be

For the east split, As $16{,}077^2 : 17{,}846^2 :: 5{\cdot}1932 : 6{\cdot}3993$ lbs,
or tested for the west „ „ $13{,}923^2 : 15{,}455^2 :: 5{\cdot}1932 : 6{\cdot}3993$ lbs.

In the east split of the No. 2 seam it would be

As $14{,}925^2 : 16{,}567^2 :: 5{\cdot}2217 : 6{\cdot}433$ lbs.,

and in the east split of No. 1 seam

As $14{,}052^2 : 15{,}598^2 :: 5{\cdot}2898 : 6{\cdot}5177$ lbs.

The ventilation of the colliery as a whole has now been increased thus:—

	At first. Cubic feet per minute.		Afterwards. Cubic feet per minute.
Discharged from shaft B	58,977		65,465
„ „ „ C	29,228		29,228
Total	88,205	Total	94,693

and with the higher quantity the pressure is 6·96395 lbs. per square foot as measured at the top of each upcast shaft, and inclusive of shaft friction.

Let us now assume that it is considered desirable to do away with shaft C entirely, to enlarge the shaft B to 20 feet in diameter, make a connection with it for the return airway of the west split in the No. 1 seam, and another for that of the west split in No. 2 seam, thus dispensing with the further use of the staple D. For simplicity of comparison let us assume that all airways remain of their former length, that of the west split in No. 2 seam being 900 yards.

If now of the air entering shaft A, say 30,000 cubic feet per minute, ventilate the No. 3 seam, we shall have in the east split 16,077
„ „ west „ 13,923

30,000 cubic feet,

being the same before and after altering the shafts.

The loss of pressure in shaft A from the No. 2 to the No. 3 seam will be ·005471 lb. as before, and in shaft B the pressure between those seams will now be the same as that of shaft A, viz., ·005471 lb. The pressure for the airways is 5·1932 lbs. as before, so that the pressure influencing the flow of air in the two splits of the No. 2 seam is

·005471
·005471
5·1932
5·204142 lbs. per square foot.

The velocity of the air in the east split of No. 2 seam due to this pressure is

$$v = \sqrt{\frac{5{\cdot}20414 \times 36}{({\cdot}0000000217) \times (6 \times 4 \times 700 \times 3)}} = 413{\cdot}88 \text{ feet per minute}$$

and the quantity of air is 413·88 × 36 = 14,900 cubic feet per minute.

The relative quantities for the No. 2 seam splits are

$$\text{For the east split } \sqrt{\frac{36^3}{700}} = 8{\cdot}164$$

$$\text{,, ,, west ,, } \sqrt{\frac{36^3}{900}} = 7{\cdot}2$$

15·364

Therefore the quantity passing in the west split is

As 8·164 : 7·2 :: 14,900 : 13,140 cubic feet per minute

and the whole ventilation of this seam amounts to

In the east split 14,900
,, west ,, 13,140
28,040 cubic feet per minute.

30,000 cubic feet in No. 3 seam + 28,040 in No. 2 make 58,040 in the two shafts between No. 1 and No. 2 seams.

The pressure for each shaft is

$$p = \frac{({\cdot}00000001) \times (62{\cdot}832 \times 100 \times 3) \times (58{,}040)^2}{314{\cdot}16^3} = {\cdot}02048 \text{ lb.}$$

or as $30{,}000^2 : 58{,}040^2 :: {\cdot}005471 : {\cdot}02048$ lb.

The pressure influencing the flow of air in the two splits of the No. 1 seam therefore is

5·204142
·02048
·02048
5·245102 lbs. per square foot.

The velocity of the air in the east split of No. 1 seam due to this pressure is

$$\sqrt{\frac{5{\cdot}2451 \times 36}{({\cdot}0000000217) \times (6 \times 4 \times 800 \times 3)}} = 388{\cdot}67 \text{ feet per minute}$$

and the quantity of air is 388·67 × 36 = 13,992 cubic feet per minute.

The relative quantities in the No. 1 seam splits are:

$$\text{For the east split } \sqrt{\frac{36^3}{800}} = 7{\cdot}6367$$

$$\text{,, ,, west ,, } \sqrt{\frac{36^3}{1{,}000}} = 6{\cdot}8305$$

14·4672

Therefore the quantity passing in the west split of No. 1 seam is

As 7·6367 : 6·8305 :: 13,992 : 12,515 cubic feet per minute,

and the entire ventilation of this seam amounts to—

In the east split 13,992
„ west „ 12,515
26,507 cubic feet per minute.

The whole volume of air in the mine then is

In No. 1 seam 26,507 cubic feet per minute.
„ 2 „ 28,040 „ „ „ „
„ 3 „ 30,000 „ „ „ „
Total 84,547 „ „ „ „

The pressure for each of the shafts A and B between the surface and No. 1 seam is

$$p = \frac{(\cdot00000001) \times (62\cdot832 \times 300 \times 3) \times (84{,}547)^2}{314\cdot16^3} = \cdot13037 \text{ lb.}$$

or as $88{,}205^2 : 84{,}547^2$ :: ·14189 : ·13037 lb.

The total pressure of the mine then is

5·245102
·13037
·13037
5·505842 lbs. per square foot for the total

quantity of 84,547 cubic feet per minute.

But in order to compare the efficiency of the ventilation produced at the two shafts with that existing when the three were in operation we must raise the ventilating pressure to a similar extent. It was shown that with a ventilating pressure of 6·96395 lbs. per square foot measured at the top of upcast shaft C and at top of upcast shaft B with three shafts operative, 94,693 cubic feet per minute were produced. If with only two 20-foot diameter shafts working we raise the pressure from 5·50584 to 6·96395 lbs. we have

As $\sqrt{5\cdot50584} : \sqrt{6\cdot96395}$ :: 84,547 : 95,085 cubic feet per minute.

The increased quantity of air in each seam is proportional to the total increase and would be

Cubic feet per minute.
For No. 1 seam. As 84,547 : 95,085 :: 26,507 : 29,811
„ „ 2 „ „ 84,547 : 95,085 :: 28,040 : 31,535
„ „ 3 „ „ 84,547 : 95,085 :: 30,000 : 33,739
95,085

The proportion for each split in the different seams is—

In No. 1 seam.
For the east split. As 14·4672 : 7·6367 :: 29,811 : 15,736
„ „ west „ „ 14·4672 : 6·8305 :: 29,811 : 14,075
29,811

In No. 2 seam.
For the east split. As 15·364 : 8·164 :: 31,535 : 16,757
„ „ west „ „ 15·364 : 7·2 :: 31,535 : 14,778
31,535

	In No. 3 seam.	Cubic feet per minute.
For the east split.	As 16·455 : 8·8182 :: 33,739 :	18,080
„ „ west „	„ 16·455 : 7·6368 :: 33,739 :	15,659
		33,739

We may compare this result with the quantities circulating when the three shafts were in operation with an equal ventilating pressure of 6·96395 lbs. Thus—

Division of the Mine.		With downcast A 20 feet in diameter and upcasts B and C each 15 feet in diameter.	With downcast A and upcast B each 20 feet in diameter.
		Cubic feet of air per minute.	Cubic feet of air per minute.
No. 1 seam	East split	15,598	15,736
	West „	14,303	14,075
No. 2 seam	East split	16,567	16,757
	West „	14,925	14,778
No. 3 seam	East split	17,845	18,080
	West „	15,455	15,659
Total		94,693	95,085

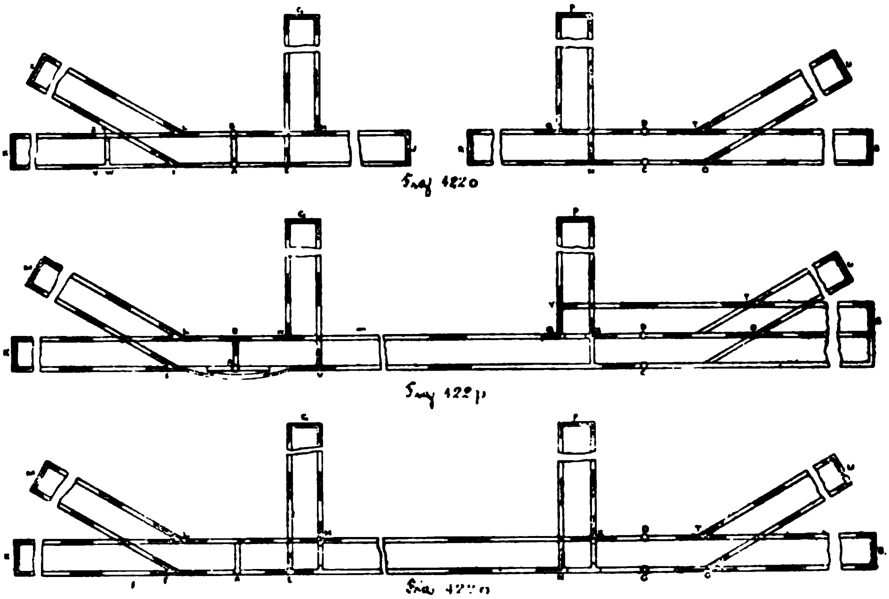
There would thus be a slight increase in the total ventilation of the mine under the given conditions by having a single upcast of 20 feet in diameter as compared with two such shafts each 15 feet in diameter. With the same total ventilating pressure, however, the quantities in the west splits of the two upper seams would be slightly reduced if no regulators were used. In the other splits an increased quantity would circulate, the change in the arrangements of shafts causing a redistribution of the air.

It has already been shown that a single downcast shaft 20 feet in diameter under the stated conditions gives a greater ventilation from the same pressure than two such shafts each of which is 15 feet in diameter, and the benefit in that case was shown to be even greater than in this where one upcast shaft 20 feet in diameter is designed to supplant two such shafts, each being 15 feet in diameter.

Suppose a large mining property to be worked by means of four shafts, as shown in Fig. 422O. Only one seam of coal is being worked, all the shafts being sunk to it. A is a downcast and coal winding shaft 700 yards in depth and 18 feet in diameter in the clear. B, the upcast for A, is 12 feet in diameter inside the walling and 690 yards in depth. The ventilation is produced by means of a furnace at B, which is not fed by a scale of fresh air from shaft A. C is a downcast and drawing shaft 680 yards in depth and 16 feet in diameter, finished. D is a fan upcast for C, 670 yards in depth and 18 feet inside diameter. The fan at the top of D ventilates the portion of workings connected to the downcast C, and the furnace at the bottom of B ventilates a separate portion of workings connected to downcast A. After a time these different portions of workings become connected as shown in Fig. 422P, and it is then decided to abandon shaft B, and either to place a fan at the top of shaft A, making C and D downcast as shown in Fig. 422P, or allow D to remain as the fan shaft and make A and C downcast as shown in Fig. 422Q.

Let us now consider the effect on the water-gauge if these alterations are carried out for the same total volume of air circulating in the mine, there being no regulators or means of checking the flow in any of the airways.

First, the workings connected with shafts A and B, Fig. 422O. It will be noticed that the air divides at the bottom of shaft A, one portion going in the direction A E 50 yards, when it is again split, and the other to A F 60 yards to its point of splitting. The split E G H is 400 yards in length, and at H this current re-unites in the returns with that of the split E J H, which is 300 yards in length to the point H. From H to the upcast shaft B the distance is 80 yards, and this airway receives the return air from the two splits E G H and E J H. The split F K L is 200 yards in length, and at L re-joins the current of air from split F M L, which is 300 yards in length. The distance from L to the upcast shaft B is 60 yards. The sectional area of the airways is greater in the intakes and returns A E, A F, H B and L B than in the splits, but we will take the average dimensions of all airways as being 7 feet by 7.



FIGS. 422O, 422P, 422Q.—ILLUSTRATING THE EFFECT ON THE WATER-GAUGE OF MAKING CERTAIN CHANGES IN THE MINE.

Of the air entering shaft A, supposing 20,000 cubic feet per minute flow through the airway A E, then the relative quantities in the two splits beyond are

$$\text{For E G H} = \sqrt{\frac{49^3}{400}} = 17{\cdot}15$$

$$\text{,, E J H} = \sqrt{\frac{49^3}{300}} = \underline{19{\cdot}803}$$

$$\underline{\underline{36{\cdot}953}}$$

and the actual quantities are

For E G H, As 36·953 : 17·15 :: 20,000 : 9,282
„ E J H, „ 36·953 : 19·803 :: 20,000 : 10,718

20,000 cubic feet per minute.

The pressure necessary to circulate the quantity in either split may now be found thus

$$\text{For E G H, } p = \frac{(\cdot 0000000217) \times (400 \times 3 \times 7 \times 4) \times (9{,}282)^2}{49^3} = \cdot 53394 \text{ lb.}$$

$$\text{or for E J H, } p = \frac{(\cdot 0000000217) \times (300 \times 3 \times 7 \times 4) \times (10{,}718)^2}{49^3} = \cdot 53394 \text{ lb.}$$

The pressure necessary to circulate 20,000 cubic feet of air per minute through the intake AE and the return BH, or a total length of 50 + 80 = 130 yards is

$$p = \frac{(\cdot 0000000217) \times (130 \times 3 \times 7 \times 4) \times (20{,}000)^2}{49^3} = \cdot 80567 \text{ lb.}$$

Apart from shaft friction the pressure for overcoming the mine resistances then is

·53394
·80567

1·33961 lbs. per square foot.

We have now to ascertain what quantities of air such pressure will produce in the splits F K L and F M L. Since the areas of the two airways are the same, the relative velocities are:—

$$\text{For F K L} = \sqrt{\frac{49^3}{200}} = 24\cdot 254$$

$$\text{„ F M L} = \sqrt{\frac{49^3}{300}} = 19\cdot 803$$

44·057

If the velocity in the longer split is 1, then that in the shorter is $\frac{24\cdot 254}{19\cdot 803}$ or 1·2247, and the velocity in the intake A F and in the return L B must be 1 + 1·2247 = 2·2247. The quantity of air passing then must be 2·2247 × 49 = 109. The relative pressure in overcoming the resistances in each split is the same, viz.:

$$\text{For F K L} = \frac{(\cdot 0000000217) \times (200 \times 3 \times 7 \times 4) \times (1\cdot 2247)^2}{49} = \cdot 00001116 \text{ lb.}$$

$$\text{„ F M L} = \frac{(\cdot 0000000217) \times (300 \times 3 \times 7 \times 4) \times (1)^2}{49} = \cdot 00001116 \text{ lb.}$$

and that for the intake A F and return L B measuring together 60 + 60 = 120 yards.

$$p = \frac{\cdot 0000000217 \times (120 \times 3 \times 7 \times 4) \times (2\cdot 2247)^2}{49} = \cdot 00002209 \text{ lb.}$$

The total relative pressure then is

For the splits	·00001116
For the main intake and return	·00002209
Total ...	·00003325 lb.

But the actual loss of pressure influencing the air from shaft A to shaft B through splits F K L and F M L has been shown to be 1·33961 lb. Since p varies in accordance with q^2 we get actually in the airways A F and L B.

As $\sqrt{{\cdot}00003325} : \sqrt{1{\cdot}33961} :: 109 : 21{,}88$: cubic feet per minute.

The proportion of this volume in the two splits is

For F K L, A $7 : 24{\cdot}254 :: 21{,}881 : 12{,}046$
„ F M L, „ $: 19{\cdot}803 :: 21{,}881 : 9{,}835$

21,881 cubic feet per minute.

The accuracy of these figures may be tested by working out the value of p for either of the splits with the proportion of air now given for it and for the airways A F and L B thus—

For the split F M L, $p = \dfrac{({\cdot}0000000217) \times (300 \times 3 \times 7 \times 4) \times (9{,}835)^2}{49^3} = {\cdot}4496$ lb.

„ „ airways A F & L B, $p = \dfrac{({\cdot}0000000217) \times (120 \times 3 \times 7 \times 4) \times (21{,}881)^2}{49^3} = {\cdot}8901$ lb.

1·3397 lb.

as compared with the 1·33961 lb. before calculated to be operating, the difference being due to a dropping of decimals in making the calculations.

The quantity of air in the shafts A and B then is

20,000
21,881

41,881 cubic feet per minute,

and we may now calculate the probable pressure for overcoming the friction in each, thus—

For shaft A, $p = \dfrac{({\cdot}00000001) \times (56{\cdot}548 \times 700 \times 3) \times (41{,}881)^2}{254{\cdot}469^3} = {\cdot}1264$ lb.

„ „ B, $p = \dfrac{({\cdot}00000001) \times (37{\cdot}699 \times 690 \times 3) \times (41{,}881)^2}{113{\cdot}097^3} = {\cdot}9462$ „

To this add the pressure necessary to overcome the friction of the mine . . . . } 1·3396 „

2·4122 „

making a total of 2·4122 lbs. pressure per square foot equal to a water-gauge of

$$\frac{2{\cdot}4122}{5.2} = {\cdot}46388 \text{ inch.}$$

Turning attention now to the workings connected with shafts C and D, Fig. 422 O. As shown by the arrows, the air divides at the bottom of shaft C, one split going in the direction C N a distance of 50 yards, when it is again divided, and the other to C O, a distance of 60 yards. At O it is again divided. The split N P Q is 400 yards in length and at Q re-unites with split N R Q, which is 300 yards in length to the junction of airways at Q. From Q to the upcast shaft D the distance is 80 yards. The split O S T is 200 yards in length, and at T re-joins the air from split O U T, which is 300 yards in length. The distance from T to the upcast shaft D is 60 yards. The sectional area of the airways is greater in the intakes and returns C N, C O, Q D and T D than in the splits, but we will assume that the average dimensions of all airways are 7 feet by 7.

It will thus be seen that all the circumstances of the workings connected with shafts C and D are precisely the same as those connected with shafts A and B, the shafts themselves only differing.

If we assume that there are 20,000 cubic feet of air per minute in the split C N, we shall have quantities in the different splits corresponding to those connected with shafts A and B.

Thus we have a pressure of ·53394 lb. for the splits N P Q and N R Q with volumes of 9,282 and 10,718 cubic feet per minute respectively.

Apart from shaft friction, the pressure necessary to overcome the mine resistances is 1·3396 lbs. per square foot. The split O S T will be swept with a ventilating current of 12,046 cubic feet per minute, while O U T will receive 9,835 cubic feet.

The quantity in the shafts C and D is 41,881 cubic feet per minute, and the pressure necessary to overcome the friction in these is

For shaft C, $p = \frac{(\cdot 00000001) \times (50\cdot 265 \times 680 \times 3) \times (41{,}881)^2}{201\cdot 062^3} = \cdot 22128$ lb.

„ „ D, $p = \frac{(\cdot 00000001) \times (56\cdot 548 \times 670 \times 3) \times (41{,}881)^2}{254\cdot 469^3} = \cdot 12099$ lb.

To this add the pressure necessary to overcome the friction of the mine . . . . 1·33961 lb.

1·68188 lb.

making a total of 1·68188 lbs. pressure per square foot as compared with 2·4122 lbs. necessary for the same quantity of air in the workings and shafts A and B, the decreased pressure being due to the increased sectional area of shaft D compared with that of shaft B.

As p varies in accordance with q^2 we may increase the pressure of the workings and of the shafts C and D to correspond with the total pressure for the workings and shafts A and B, and the quantity then circulating becomes

As $\sqrt{1\cdot 68188} : \sqrt{2\cdot 4122} :: 41{,}881 : 50{,}156$ cubic feet per minute.

and the proportion for the splits becomes

For N P Q As 41,881 : 50,156 :: 9,282 : 11,116
„ N R Q „ 41,881 : 50,156 :: 10,718 : 12,836
„ O S T „ 41,881 : 50,156 :: 12,046 : 14,426
„ O U T „ 41,881 : 50,156 :: 9,835 : 11,778

50,156 cubic feet per minute.

The increased pressure is

For the splits N P Q and N R Q. As 1·68188 : 2·4122 :: ·53394 : ·7658 lb.
„ „ mine resistances. „ 1·68188 : 2·4122 :: 1·3396 : 1·9214 lb.
„ „ splits O S T and O U T. „ 1·68188 : 2·4122 :: ·4496 : ·64483 lb.
„ main intake C N and return Q D. 1·9214 − ·7658 = 1·1556 lb.
„ „ „ C O „ „ T D. 1·9214 − ·64483 1·2766 lb.

The total volume of air circulating through all workings connected with shafts A and B and shafts C and D is

41,881
50,156

92,037 cubic feet per minute, with a water-gauge of ·46388 inch at each colliery inclusive of shaft friction.

We have now to consider the changed circumstances of the mine resulting from the workings at J and R, Fig. 422O, having advanced and met. Such connection would at once disturb previous ventilating arrangements and cause a re-distribution of the air. Fig. 422P shows a plan of the workings with this communication completed, with the shaft B abandoned, and the course of the air currents as now designed is indicated by the arrows. In this arrangement the fan is erected at A, and C and D are downcast shafts.

The air entering shaft C passes in one undivided current along a single intake airway to the junction of roads at V, where it is split into two currents, one of which passes round V G W to meet at W the return air from workings ventilated by shaft D, the other flowing to F, where it is again divided, one subdivision going round F K L and the other round F M L to become again united at L and pass in a single current to B, where it meets the return air from V G W and from workings ventilated by shaft D. The whole volume of air then passes through an airway B A of large sectional area into and up the shaft A.

At the bottom of shaft D the air is divided into two currents, one passing round by X P Y and at the point Y is joined by the return air from other workings. The increased volume representing the whole of the air entering shaft D now passes by the point Q to W, where it joins the return air from V G W and on to and up the upcast shaft A. The other current of air from shaft D passes to the point O, where it is split into two divisions, one going round O S T and the other round O U T to become re-united at T and continue as one current to the point Y, where it joins the return air from X P Y and so on to and up the upcast shaft A.

The lengths of the airways are—

C V	350 yards.	D X P Y	400 yards.
V G W	400 „	Y Q W	330 „
V F	140 „	D O	100 „
F K L	200 „	O S T	200 „
F M L	300 „	O U T	300 „
L B	60 „	T Y	180 „
W B	50 „		
B A	30 „		

The sectional area of the airways is greater in the main intakes before splitting takes place and in the main returns after re-union of the air currents, but we will assume the average dimensions of all airways except W B and B A to be 7 feet by 7 and W B and B A to be each 8 feet high by 12 feet wide.

If the split D O has a current of, say, 21,881 cubic feet per minute, then the proportion for the split O S T is 12,046 and of O U T 9,835 cubic feet per minute, with a pressure of ·4496 lb.

The pressure necessary for the main intake DO 100 yards in length, and return airway T Y 180 yards in length, or in all 280 yards, is

$$p = \frac{(\cdot 0000000217) \times (280 \times 3 \times 7 \times 4) \times (21{,}881)^2}{49^3} = 2{\cdot}077 \text{ lbs.}$$

Apart from shaft friction the pressure expended at Y to produce the current round D X P Y is

$$\begin{array}{l} \cdot 4496 \\ 2{\cdot}077 \\ \hline 2{\cdot}5266 \text{ lbs. per square foot} \end{array}$$

and the velocity in that airway is

$$v = \sqrt{\frac{2{\cdot}5266 \times 49}{(\cdot 0000000217) \times (7 \times 4 \times 400 \times 3)}} = 412{\cdot}07 \text{ feet per minute}$$

and the quantity of air passing is $412{\cdot}07 \times 49 = 20{,}191$ cubic feet per minute.

The whole volume of air in shaft D then is

21,881
20,191
42,072 cubic feet per minute.

The pressure necessary to overcome the friction of this volume from Y to W in the return is

$$p = \frac{(\cdot 0000000217) \times (330 \times 3 \times 7 \times 4) \times (42{,}072)^2}{49^3} = 9\cdot 0505 \text{ lbs.}$$

and the loss of pressure at W determining the current round V G W is

2·5266
9·0505
11·5771 lbs. per square foot.

Our difficulty now is that we do not know the volume of the current round V G W coming from shaft C, and without this knowledge it is difficult to ascertain the difference of water-gauge as measured at the two points W and B. Let us, however, assume that a quantity of air enters shaft C similar to that which passes down shaft D, and if this quantity is proved wrong, we can afterwards adjust it by substituting the proper quantity. The pressure necessary to overcome the friction of airway C V then is

$$p = \frac{(\cdot 0000000217) \times (350 \times 3 \times 7 \times 4) \times (42{,}072)^2}{49^3} = 9\cdot 599 \text{ lbs.}$$

Apart from shaft friction, which, however, for strict accuracy ought to be taken into consideration, the loss of pressure at point W being 11·5771 lbs. we have 11·5771 − 9·599 = 1·9781 lb. used in overcoming the friction in airway V G W. The velocity in it then is $\sqrt{\dfrac{1\cdot 9781 \times 49}{(\cdot 0000000217) \times (400 \times 3 \times 7 \times 4)}} = 364\cdot 6$ feet per minute, and the quantity circulating is 364·6 × 49 = 17,865 cubic feet per minute.

The volume of air from the points W to B then becomes

42,072
17,865
59,937 cubic feet per minute

and the pressure to overcome the friction in that airway is

$$p = \frac{\cdot 0000000217 \times (50 \times 3 \times 40) \times (59{,}937)^2}{96^3} = \cdot 52867 \text{ lb.}$$

making the loss of pressure in the mine to point B

11·5771
·52867
12·10577 lbs. per square foot.

The volume of air in the intake V F is

42,072 − 17,865 = 24,207 cubic feet per minute.

The relative quantities for F K L and F M L are

$$\text{F K L} = \sqrt{\frac{49^3}{200}} = 24\cdot 254$$

$$\text{F M L} = \sqrt{\frac{49^3}{300}} = 19\cdot 803$$

44·057

and the actual quantities are

F K L. As 44·057 : 24·254 :: 24,207 : 13,326
F M L. „ 44·057 : 19·803 :: 24,207 : 10,881

24,207 cubic feet per minute.

The pressure necessary for each split is the same, and may be found for either F K L or F M L thus—

For F K L. As $12{,}046^2 : 13{,}326^2 :: \cdot4496 : \cdot55026$ lb.
or „ K M L. „ $9{,}835^2 : 10{,}881^2 :: \cdot4496 : \cdot55026$ lb.

The pressure spent in overcoming friction in airways V F and L B, which are together 200 yards in length

$$=\frac{(\cdot0000000217)\times(200\times3\times7\times4)\times(24{,}207)^2}{49^3}=1\cdot8157 \text{ lb.}$$

or, as $12{,}046^2 : 24{,}207^2 :: \cdot4496 : 1\cdot8157$ lb.

The addition of the pressures now ascertained is

For C V = 9·599
„ V F and L B = 1·8157
„ F K L and F M L = ·55026

11·96496 lbs. per square foot.

That is on the assumption that 42,072 cubic feet of air per minute descend shaft C, the pressure to overcome friction in the airways to the point B is 11·96496 lbs. per square foot.

But on the assumption that a quantity of 42,072 cubic feet per minute enter shaft D it has been calculated that the friction in other airways to the point B is equal to 12·10577 lbs. per square foot, showing that both quantities cannot be right. We may, however, calculate the proper amount of increase in the volume entering shaft C or of decrease in the volume entering shaft D to make the calculations of the loss of pressures at B harmonize. This will be very nearly so if we adopt the former alternative and increase the quantity of air entering at C in proportion to the two calculated pressures at B. Thus—

As $\sqrt{11\cdot96496} : \sqrt{12\cdot10577} :: 42{,}072 : 42{,}320$ cubic feet per minute.

But this will not give a strictly correct result, because when the quantity of air entering shaft C is increased there is a slight disturbance in the pressure at B owing to a proportionate increase in the quantity of air going round V G W. This disturbance, however, is so very slight that practically we may disregard it.

If 42,072 cubic feet of air per minute enter shaft D then 42,320 cubic feet per minute enter shaft C, and the adjusted quantities must be

In airway V G W As 42,072 : 42,320 :: 17,865 : 17,970
„ F K L „ 42,072 : 42,320 :: 13,326 : 13,405
„ F M L „ 42,072 : 42,320 :: 10,881 : 10,945

Total 42,320 cubic feet per min.

and the correspondingly adjusted pressures are

For C V As 11·96496 : 12·10577 :: 9·599 : 9·712
„ F K L and F M L „ 11·96496 : 12·10577 :: ·55026 : ·5567
„ V F and L B „ 11·96496 : 12·10577 :: 1·8157 : 1·8371

Total 12·1058 lbs. per sq. ft.

We have now passing along the return airway B A

42,072
42,320
84,392 cubic feet per minute, the pressure

for overcoming which is

$$\frac{({\cdot}0000000217) \times (30 \times 3 \times 40) \times (84{,}392)^2}{96^3} = {\cdot}62885$$

That for shaft A is as $41{,}881^2 : 84{,}392^2 :: {\cdot}1264 : {\cdot}51323$
,, ,, ,, D ,, $41{,}881^2 : 42{,}072^2 :: {\cdot}12099 : {\cdot}1221$
,, ,, ,, C ,, $41{,}881^2 : 42{,}320^2 :: {\cdot}22128 : {\cdot}22594$

1·49012 lbs.

which, added to 12·10577, previously ascertained for the mine, makes a total of 13·59589 lbs. per square foot for the whole of the workings and shafts with a total ventilation of 84,392 cubic feet per minute.

Referring now to the calculations made for the ventilation of the colliery before the workings at J and R (Fig. 422O) were connected, it was then shown that there were 92,037 cubic feet per minute circulating with a pressure of 2·4122 lbs. per square foot or a water-gauge of ·46388 inch. In order to compare the water-gauge after the alterations in the mine are carried out with that before, let us increase the total volume of air being discharged at shaft A from 84,392 cubic feet per minute to 92,037, the amount previously circulated by the two ventilating powers. The pressure then becomes—

As $84{,}392^2 : 92{,}037^2 :: 13{\cdot}59589 : 16{\cdot}17$ lbs.

or a water-gauge of $\frac{16{\cdot}17}{5{\cdot}2} = 3{\cdot}11$ inches.

The proportion of air entering the downcast shafts now is—

Entering D As 84,392 : 92,037 :: 42,072 : 45,884
,, C ,, 84,392 : 92,037 :: 42,320 : 46,153
Total 92,037

The quantities of air in the different splits of the mine are—

		Cubic feet per minute.
In O S T	As 42,072 : 45,884 :: 12,046 :	13,138
,, O U T	,, 42,072 : 45,884 :: 9,835 :	10,726
,, D O and T Y		23,864
,, D X P Y	,, 42,072 : 45,884 :: 20,191 :	22,020
	Total volume of air in shaft D	45,884
In F K L	As 42,072 : 46,153 :: 13,326 :	14,619
,, F M L	,, 42,072 : 46,153 :: 10,881 :	11,936
,, V F and L B		26,555
,, V G W	,, 42,072 : 46,153 :: 17,865 :	19,598
	Total volume of air in shaft C	46,153
	In airway B A and shaft A	92,037

The increased pressures are—

							lbs. per square foot.
For D O and T Y	As	13·59589	: 16·17	::	2·077	:	2·4703
„ splits O S T and O U T	„	13·59589	: 16·17	::	·4496	:	·5347
„ D X P Y							3·0050
From Y to W	„	13·59589	: 16·17	::	9·0505	:	10·7644
For C V and V G W (V G W being 13·7694 − 11·5512 = 2·2182)							13·7694
„ W B	As	13·59589	: 16·17	::	·52867	:	·6287
„ B A	„	13·59589	: 16·17	::	·62885	:	·7479
„ shaft A	„	13·59589	: 16·17	::	·51323	:	·6104
* „ „ D	„	13·59589	: 16·17	::	·1221	:	·1452
* „ „ C	„	13·59589	: 16·17	::	·22594	:	·2687
					13·59589	:	16·1703

For C V	As	13·59589	: 16·17	::	9·712	:	11·5512
„ F K L and F M L	„	13·59589	: 16·17	::	·5567	:	·6621
„ V F and L B	„	13·59589	: 16·17	::	1·8371	:	2·1851
Expended on the mine C to B					12·1058	:	14·3984

Expended on the mine C to B	As	13·59589	: 16·17	::	12·1058	:	14·3984
To which add the amount calculated for B A							·7479
„ „ „ „ shaft A							·6104
„ „ „ „ „ D							·1452
„ „ „ „ „ C							·2687
Total							16·1706

The accuracy of these pressures may be tested by the formula $p = \frac{ksv^2}{a}$ when we have—

lbs. per square foot.

For D O and T Y $$p = \frac{(\cdot0000000217) \times (280 \times 3 \times 7 \times 4) \times (23{,}864)^2}{49^3} = 2\cdot4705$$

„ O S T $$p = \frac{(\cdot0000000217) \times (200 \times 3 \times 7 \times 4) \times (13{,}138)^2}{49^3} = \cdot5348$$

„ O U T $$p = \frac{(\cdot0000000217) \times (300 \times 3 \times 7 \times 4) \times (10{,}726)^2}{49^3} = \cdot5347$$

„ D X P Y $$p = \frac{(\cdot0000000217) \times (400 \times 3 \times 7 \times 4) \times (22{,}020)^2}{49^3} = 3\cdot0050$$

„ F K L $$p = \frac{(\cdot0000000217) \times (200 \times 3 \times 7 \times 4) \times (14{,}619)^2}{49^3} = \cdot6622$$

„ F M L $$p = \frac{(\cdot0000000217) \times (300 \times 3 \times 7 \times 4) \times (11{,}936)^2}{49^3} = \cdot6622$$

„ V F and L B $$p = \frac{(\cdot0000000217) \times (200 \times 3 \times 7 \times 4) \times (26{,}555)^2}{49^3} = 2\cdot1851$$

„ V G W $$p = \frac{(\cdot0000000217) \times (400 \times 3 \times 7 \times 4) \times (19{,}598)^2}{49^3} = 2\cdot3803$$

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* A slight error arises from adding the pressure of these shafts here instead of their difference being allowed for earlier in the calculations, but the error is not of sufficient importance to appreciably affect the result obtained.

lbs. per sq. foot.

For C V $$p = \frac{({\cdot}0000000217) \times (350 \times 3 \times 7 \times 4) \times (46{,}153)^2}{49^3} = 11{\cdot}5510$$

„ W B $$p = \frac{({\cdot}0000000217) \times (50 \times 3 \times 40) \times (65{,}482)^2}{96^3} = {\cdot}6310$$

„ B A $$p = \frac{({\cdot}0000000217) \times (30 \times 3 \times 40) \times (92{,}037)^2}{96^3} = {\cdot}7479$$

„ Y to W $$p = \frac{({\cdot}0000000217) \times (330 \times 3 \times 7 \times 4) \times (45{,}884)^2}{49^3} = 10{\cdot}7644$$

Shaft A $$p = \frac{({\cdot}00000001) \times (700 \times 3 \times 56{\cdot}548) \times (92{,}037)^2}{254{\cdot}469^3} = {\cdot}6104$$

„ D $$p = \frac{({\cdot}00000001) \times (670 \times 3 \times 56{\cdot}548) \times (45{,}884)^2}{254{\cdot}469^3} = {\cdot}1452$$

„ C $$p = \frac{(00000001) \times (680 \times 3 \times 50{\cdot}265) \times (46{,}153)^2}{201{\cdot}062^3} = {\cdot}2687$$

In order to maintain the same total volume of air in the mine in the first and second arrangement under given conditions, the water-gauge must be increased from ·46388 inch at two separate collieries so far as ventilation is concerned, to 3·11 inches at the single establishment. In the first instance the effective horse-power in the air—

At the upcast B is $$\frac{41{,}881 \times {\cdot}46388 \times 5{\cdot}2}{33{,}000} = 3{\cdot}0613$$

and at shaft D it is $$\frac{50{,}156 \times {\cdot}46388 \times 5{\cdot}2}{33{,}000} = 3{\cdot}6663$$

Making together 6·7276

while in the second instance it is $$\frac{92{,}037 \times 3{\cdot}11 \times 5{\cdot}2}{33{,}000} = 45{\cdot}104$$

The distribution of the air in the mine is as follows:—

Division of the Mine.	First arrangement with shafts A and C downcast and B and D upcast.		Second arrangement with shafts C and D downcast and A upcast, B being abandoned.	
	Volume of air. Cub. ft. per min.	Pressure in overcoming friction. lbs. per sq. ft.	Volume of air. Cub. ft. per min.	Pressure in overcoming friction. lbs. per sq. ft.
In split E G H, corresponding to V G W	9,282	·53394	19,598	2·2182
„ „ E J H, removed in 2nd arrangement	10,718			
„ „ F K L	12,046	·4496	14,619	·6621
„ „ F M L	9,835		11,936	
„ „ N P Q, corresponding to D X P Y	11,116	·7658	22,020	3·0050
„ „ N R Q, removed in 2nd arrangement	12,836			
„ „ O S T	14,426	·64483	13,138	·5347
„ „ O U T	11,778		10,726	
Total	92,037		92,037	

A comparison of these figures shows the advantage of reduced pressure to be derived from splitting the air.

We now pass to consider the third arrangement, in which D is to remain as the upcast shaft, while A and C are downcast, B being abandoned as shown in Fig. 422Q.

In this case, the air entering shaft C passes in its entirety to O, at which point splitting into two currents takes place. One part of the divided volume passes round O S T and the other round O U T, the two divisions becoming re-united at the point T and together following the return airway to shaft D as in the first arrangement.

After descending shaft A, here the whole volume of air becomes divided into two currents, one passing to the point E, where it is again split, while the other follows the main intake A F, to the point F, where it is also subdivided. One split from the point E follows the course E N P X and at the point X, in the main return airway joins the air from other splits and along with it passes to and up shaft D. The other split from the point E ventilates the district E G H, and at the point H in the return airway unites with other air currents, the increased volume following the return airway to the point X where it is further augmented by the current from E N P X, and continues with it to and up the shaft D. A split from the point F passes round F K L and another passes round F M L, becoming re-united at the point L as in the first arrangement, and these afterwards pass in one volume to the point H, where the volume is again increased by the current from E G H, the further course to shaft D having been already indicated.

The lengths of the airways are—

C O	60 yards	A E	50 yards
O S T	200 „	E N P X	700 „
O U T	300 „	E G H	400 „
T D	60 „	H X	300 „
		X D	50 „
		A F	60 „
		F K L	200 „
		F M L	300 „
		L H	140 „

The sectional area of the airways is greater in the main intakes before splitting takes place and in the main returns after re-union of the air-currents, but we will assume the average dimensions of all airways except X D, A E, C O and D T to be 7 feet by 7 and X D, A E, C O and D T to be 8 feet high and 12 feet wide.

Let us assume that 12,046 cubic feet per minute pass round F K L and 9,835 cubic feet round F M L with a pressure of ·4496 lb. used in these splits as in the first instance. Then a volume of 21,881 cubic feet passes along the airways A F and L H, a total distance of 200 yards. The pressure necessary for this quantity for a distance of 120 yards has been shown to be ·8901 lb. per square foot, and, therefore, for 200 yards of similar sized airway it will be

As 120 : 200 :: ·8901 : 1·4835 lb. per square foot.

Apart from shaft friction the loss of pressure at H influencing the current round E G H then is

·4496
1·4835
1·9331 lb. per square foot.

We do not know the quantity passing along A E at present, and so are unable to calculate the pressure necessary for it. The pressure will not, however, be great, and we shall get a very close approximation to the volume going round E G H by taking the proper pressure of 1·9331 lb. for it, including that necessary to overcome the friction of a larger volume in the airway A E and applying it to ascertain the velocity as though that were uniform from the shaft A to the point H.

Approximately the velocity round E G H is

$$\sqrt{\frac{1{\cdot}9331 \times 49}{({\cdot}0000000217 \times \{(400 \times 3 \times 7 \times 4) + (50 \times 3 \times 40)\}}} = 332 \text{ feet per min.}$$

and the quantity of air is $332 \times 49 = 16{,}268$ cubic feet per minute. If this be added to the 21,881 cubic feet which joins it at H, we have 38,149 cubic feet per minute in the return airway H X, the pressure for overcoming which is

$$\frac{({\cdot}0000000217) \times (300 \times 3 \times 7 \times 4) \times (38{,}149)^2}{49^3} = 6{\cdot}7646 \text{ lbs.}$$

or, As $9{,}835^2 : 38{,}149^2 :: {\cdot}4496 : 6{\cdot}7646$ lbs.

Apart from shaft friction the loss of pressure at X then which regulates the inflowing current E N P X is

1·9331
6·7646

8·6977 lbs. per square foot

approximately including its course along A E.

According to this pressure, which, however, includes that necessary for a larger volume in the airway A E, the velocity in E N P X is

$$\sqrt{\frac{8{\cdot}6977 \times 49}{({\cdot}0000000217) \times \{(700 \times 3 \times 7 \times 4) + (50 \times 3 \times 40)\}}} = 550{\cdot}53 \text{ ft. per min.}$$

and the quantity of air is $550{\cdot}53 \times 49 = 26{,}976$ cubic feet per minute approximately.

As a check to these calculations, the pressure necessary for the circulation of $26{,}976 + 16{,}268 = 43{,}244$ cubic feet per minute along A E may now be found thus—

$$\frac{({\cdot}0000000217) \times (50 \times 3 \times 40) \times (43{,}244)^2}{96^3} = {\cdot}2752 \text{ lb.}$$

or as $59{,}937^2 : 43{,}244^2 :: {\cdot}52867 : {\cdot}2752$ lb., leaving the loss of pressure from E to H, which induces the current E G H, as $1{\cdot}9331 - {\cdot}2752 = 1{\cdot}6579$ lb.

The velocity of the volume along E G H due to such pressure is

$$\sqrt{\frac{1{\cdot}6579 \times 49}{({\cdot}0000000217) \times (400 \times 3 \times 7 \times 4)}} = 333{\cdot}79 \text{ feet per minute.}$$

and the volume itself is $333{\cdot}79 \times 49 = 16{,}356$ cubic feet per minute, which nearly approaches the quantity of 16,268 previously calculated. If we take the quantity in even figures, viz., 16,300, we shall be near the mean of the two calculations and close to the true quantity of air circulating in the split E G H. This altered quantity, however, will, in turn, slightly affect the pressure expended at X. The quantity passing along H X as now calculated is $21{,}881 + 16{,}300 = 38{,}181$ cubic feet per minute instead of the 38,149 previously calculated. The altered value of p for H X then becomes

As $9{,}835^2 : 38{,}181^2 :: {\cdot}4496 : 6{\cdot}775$ lbs.

This makes the pressure expended at X

1·9331
6·775

8·7081 lbs. per square foot instead of the 8·6977, the approximate amount previously found.

If from 8·7081 we deduct ·2752 lb. the pressure expended in overcoming the friction of the air in A E, we get 8·4329 lbs. per square foot as the effective pressure which causes the current along E N P X. Its velocity then is—

$$\sqrt{\frac{8{\cdot}4329 \times 49}{({\cdot}0000000217) \times (700 \times 3 \times 7 \times 4)}} = 569{\cdot}07 \text{ feet per minute,}$$

and the quantity of air passing is 569·07 × 49 = 27,884 cubic feet per minute. The approximate quantity now calculated for the airway A E is 16,300 + 27,884 = 44,184 cubic feet per minute as compared with previous approximate quantity of 43,244. For practical purposes this may be considered the correct quantity. The pressure necessary to overcome friction in airway A E for such a quantity is—

$$\text{As } 59{,}937^2 : 44{,}184^2 :: {\cdot}52867 : {\cdot}2873 \text{ lb.}$$

leaving the loss of pressure from E to H, which controls the current E G H, as 1·9331 − ·2873 = 1·6458 lb. per square foot.

The air in the split E G H will now be—

$$\text{As } \sqrt{1{\cdot}6579} : \sqrt{1{\cdot}6458} :: 16{,}356 : 16{,}296 \text{ cubic feet per minute.}$$

The quantity passing along H X would be altered to 21,881 + 16,296 = 38,177 cubic feet per minute, for which the pressure is

$$\text{As } 9{,}835^2 : 38{,}177^2 :: {\cdot}4496 : 6{\cdot}7746 \text{ lbs.}$$

Apart from shaft friction this makes the pressure expended at X

1·9331
6·7746

8·7077 lbs. per square foot.

If from 8·7077 lbs. we deduct ·2873 the pressure expended in overcoming the friction of the volume in A E we get 8·4204 lbs. per square foot as the effective pressure for the current along E N P X. Its quantity then is

$$\text{As } \sqrt{8{\cdot}4329} : \sqrt{8{\cdot}4204} :: 27{,}884 : 27{,}864.$$

We have now for E G H 16,296
E N P X 27,864

44,160 cubic feet per minute,

which is very nearly the same amount as last calculated, and may be accepted as being practically accurate.

The total of the air in the main return X D is—

38,177
27,864

66,041 cubic feet per minute,

and the pressure necessary to overcome its friction is—

$$\text{As } 59{,}937^2 : 66{,}041^2 :: {\cdot}52867 : {\cdot}64183 \text{ lb.}$$

The total pressure for the workings between shafts A and D is

	lbs. per sq. foot.	lbs. per sq. foot.
In the splits F K L and F M L		·4496
In airways A F and L H		1·4835
„ „ H X		6·7746
„ „ A E	·2873	
„ „ E N P X or E G H and H X	8·4204	
„ „ X D	·64183	·64183
	9·34953	9·34953

This loss of pressure determines the flow of air down the shaft C round the splits O S T and O U T.

As shown before, the relative velocities in these airways are

$$\text{For O S T, } \sqrt{\frac{49^3}{200}} = 24{\cdot}254$$

$$\text{,, O U T, } \sqrt{\frac{49^3}{300}} = 19{\cdot}803$$

$$\underline{\underline{44{\cdot}057}}$$

If the velocity in the longer split is 1, then that in the shorter is $\frac{24{\cdot}254}{19{\cdot}808} = 1{\cdot}2247$, and the velocity in the main intake C O, and in the return T D, would, if of the same sectional area as O S T and O U T, be $1 + 1{\cdot}2247 = 2{\cdot}2247$. As, however, the sectional areas are as 49 : 96, the velocity in C O and T D will be reduced as 96 : 49 :: 2·2247 : 1·1355. The quantity of air passing then must be $1{\cdot}1355 \times 96 = 109{\cdot}01$. As shown before, the relative pressure for the splits is the same, viz.,

$$\text{For O S T } \frac{({\cdot}0000000217) \times (200 \times 3 \times 7 \times 4) \times (1{\cdot}2247)^2}{49} = {\cdot}00001116 \text{ lb.}$$

$$\text{,, O U T } \frac{({\cdot}0000000217) \times (300 \times 3 \times 7 \times 4) \times (1)^2}{49} = {\cdot}00001116 \text{ lb.,}$$

and that for the intake C O and return T D measuring together 60 + 60 = 120 yards.

$$p = \frac{({\cdot}0000000217) \times (120 \times 3 \times 40) \times (1{\cdot}1355)^2}{96} = {\cdot}000004197 \text{ lb.}$$

The total relative pressure then is

For O S T and O U T ·00001116
,, C O and T D ·000004197
Total ·000015357 lb. per square foot.

But the actual pressure influencing the air flow from shaft C to shaft D is 9·34953 lbs. per square foot. Since p varies in accordance with q^2 we get actually in the airways C O and D T

$$\text{As } \sqrt{{\cdot}000015357} : \sqrt{9{\cdot}34953} :: 109{\cdot}01 : 85{,}057$$

The proportion of this volume in the two splits is

For O S T As 44·057 : 24·254 :: 85,057 : 46,825
,, O U T As 44·057 : 19·803 :: 85,057 : 38,232
85,057 cub. ft. per min.

The accuracy of the figures may be tested by working out the value of p for one of the splits and also for the airways C O and D T thus—

For the split O U T As $9{,}835^2 : 38{,}232^2 :: {\cdot}4496 : 6{\cdot}794$
,, C O and D T

$$p = \frac{({\cdot}0000000217) \times (120 \times 3 \times 40) \times (85{,}057)^2}{96^3} = 2{\cdot}5552$$

9·3492 lbs. per sq. ft.

It may be observed that considerable benefit is derived from having the main intake C O and return T D of enlarged sectional area. If only of the same size as the split airways inside, viz. 7 feet by 7, then with the same pressure of

9·34953 lbs. at D we should have only 57,805 cubic feet per minute entering shaft C. The relative velocities of the air would then be

In O S T	1·2247
„ O U T	1·
„ C O and D T	2·2247

As shown before, the relative pressures would be

For the splits	·00001116
That for C O and D T is	·00002209
	·00003325 lb. per square foot.

The quantity in C O and D T then would be

As $\sqrt{\cdot00003325} : \sqrt{9\cdot34953} :: 109 : 57{,}805$ cubic feet per minute.

The proportion of this volume in the two splits would be

For O S T As $44\cdot057 : 24\cdot254 :: 57{,}805 : 31{,}822$
„ O U T „ $44\cdot057 : 19\cdot803 :: 57{,}805 : 25{,}983$

57,805 cubic feet per min.

The value of p would be

For the split O U T As $9{,}835^2 : 25{,}983^2 :: \cdot4496 : 3\cdot1379$
„ C O and D T „ $21{,}881^2 : 57{,}805^2 :: \cdot8901 : 6\cdot212$

9·3499 lbs. per sq. ft.

In this case by far the greater part of the total pressure is spent in overcoming the friction in the main intake and main return, whereas when these are 12 feet by 8 the greater part is spent in overcoming the friction in the splits.

We have now to calculate the pressure necessary to overcome friction in the shafts themselves.

First, in shaft A, which is 700 yards in depth, with a circumference of 56·548 feet and an area of 254·469 square feet, we have a volume of $21{,}881 + 44{,}160 = 66{,}041$ cubic feet per minute, and consequently the value of p is

As $41{,}881^2 : 66{,}041^2 :: \cdot1264 : \cdot3143$ lb. per square foot.

In shaft C, which is 680 yards in depth, with a circumference of 50·265 feet and an area of 201·062 square feet, we have a volume of 85,057 cubic feet per minute, and the value of p is

As $41{,}881^2 : 85{,}057^2 :: \cdot22128 : \cdot91236$ lb. per square foot.

In shaft D, which is 670 yards in depth, with a circumference of 56·548 feet and an area of 254·469 square feet, we have a volume of $66{,}041 + 85{,}057 = 151{,}098$ cubic feet per minute, and the value of p is

As $41{,}881^2 : 151{,}098^2 :: \cdot12099 : 1\cdot5748$ lb. per square foot.

The pressure necessary to overcome friction in the workings has been shown to be

	lbs. per square foot.
	9·34953
To which must now be added that for shaft A	·3143
„ „ „ C	·91236
„ „ „ D	1·5748
Making a total of	12·15099

But with this pressure there is in the aggregate a ventilation of 151,098 cubic feet of air per minute circulating at the colliery. We may, however, now see what the pressure would be for a quantity of 92,037 cubic feet so as to compare with the first and second arrangements shown in Figs. 422O and 422P.

Thus p = As $151{,}098^2 : 92{,}037^2 :: 12·15099 : 4·50835$ lbs., or a water-gauge of $\frac{4·50835}{5·2} = ·86699$ inch.

The air entering shaft A would then be

	As 151,098 : 92,037 :: 66,041 :	40,227
and shaft C	,, 151,098 : 92,037 :: 85,057 :	51,810
		92,037 cubic feet per minute,

this volume being discharged from shaft D.

The proportion for the various splits of the mine would be—

		Cubic feet per minute.
In F K L	As 66,041 : 40,227 :: 12,046 :	7,337
,, F M L	,, 66,041 : 40,227 :: 9,835 :	5,991
,, A F and L H		13,328
,, A E	,, 66,041 : 40,227 :: 44,160 :	26,899
	Total in shaft A	40,227
,, E G H	,, 66,041 : 40,227 :: 16,296 :	9,926
,, A F and L H as above		13,328
,, H X		23,254
,, E N P X	,, 66,041 : 40,227 :: 27,864 :	16,973
	Total in shaft A	40,227
,, O S T	,, 85,057 : 51,810 :: 46,825 :	28,522
,, O U T	,, 85,057 : 51,810 :: 38,232 :	23,288
	Total in shaft C and airways C O and D T	51,810

The pressures necessary for such quantities are—

		lbs. per square foot
In F K L and F M L	As 12·15099 : 4·50835 :: ·4496 :	·16681
,, A F and L H	,, 12·15099 : 4·50835 :: 1·4835 :	·55042
	Pressure expended at H	·71723
,, A E	,, 12·15099 : 4·50835 :: ·2873 :	·1066
,, E G H	,, 12·15099 : 4·50835 :: 1·6458 :	·61063
	Pressure expended at H	·71723
,, H X	,, 12·15099 : 4·50835 :: 6·7746 :	2·51356
	Expended at X	3·23079
,, X D	,, 12·15099 : 4·50835 :: ·64183 :	·23814
	Pressure expended at bottom of shaft D	3·46893

		lbs. per square foot.
In A E as above		·1066
„ E N P X	As 12·15099 : 4·50835 :: 8·4204	: 3·12419
	Expended at X	3·23079
„ X D as above		·23814
	Expended at bottom of shaft D	3·46893
„ shaft A	As 12·15099 : 4·50835 :: ·3143	: ·11661
„ „ C	„ 12·15099 : 4·50835 :: ·91236	: ·33851
„ „ D	„ 12·15099 : 4·50835 :: 1·5748	: ·5843
	Total	4·50835
In O S T and O U T	As 12·15099 : 4·50835 :: 6·794	: 2·5208
„ C O and D T	„ 12·15099 : 4·50835 :: 2·5552	: ·9481
	Expended at bottom of shaft D	3·4689

The accuracy of these pressures may be tested by the formula $p = \frac{ksv^2}{a}$ when we have

lbs. per square ft.

For F K L $p = \frac{(\cdot0000000217) \times (200 \times 3 \times 7 \times 4) \times (7{,}337)^2}{49^3} = \cdot16681$

„ F M L $p = \frac{(\cdot0000000217) \times (300 \times 3 \times 7 \times 4) \times (5{,}991)^2}{49^3} = \cdot16682$

„ A F & L H $p = \frac{(\cdot0000000217) \times (200 \times 3 \times 7 \times 4) \times (13{,}328)^2}{49^3} = \cdot55044$

„ A E $p = \frac{(\cdot0000000217) \times (50 \times 3 \times 40) \times (26{,}899)^2}{96^3} = \cdot1065$

„ E G H $p = \frac{(\cdot0000000217) \times (400 \times 3 \times 7 \times 4) \times (9{,}926)^2}{49^3} = \cdot6106$

„ H X $p = \frac{(\cdot0000000217) \times (300 \times 3 \times 7 \times 4) \times (23{,}254)^2}{49^3} = 2\cdot5135$

„ E N P X $p = \frac{(\cdot0000000217) \times (700 \times 3 \times 7 \times 4) \times (16{,}973)^2}{49^3} = 3\cdot1244$

„ X D $p = \frac{(\cdot0000000217) \times (50 \times 3 \times 40) \times (40{,}227)^2}{96^3} = \cdot23814$

„ O S T $p = \frac{(\cdot0000000217) \times (200 \times 3 \times 7 \times 4) \times (28{,}522)^2}{49^3} = 2\cdot5208$

„ O U T $p = \frac{(\cdot0000000217) \times (300 \times 3 \times 7 \times 4) \times (23{,}288)^2}{49^3} = 2\cdot5208$

„ C O & D T $p = \frac{(\cdot0000000217) \times (120 \times 3 \times 40) \times (51{,}810)^2}{96^3} = \cdot9481$

„ Shaft A $p = \frac{(\cdot00000001) \times (700 \times 3 \times 56\cdot548) \times (40{,}227)^2}{254\cdot469^3} = \cdot11661$

„ „ C $p = \frac{(\cdot00000001) \times (680 \times 3 \times 50\cdot265) \times (51{,}810)^2}{201\cdot062^3} = \cdot3386$

„ „ D $p = \frac{(\cdot00000001) \times (670 \times 3 \times 56\cdot548) \times (92{,}037)^2}{254\cdot469^3} = \cdot5843$

It will be noted that the same total volume of air is maintained in the mine in the first, second, and third arrangements under given conditions, the water-gauge being respectively ·46388, 3·11, and ·86699 inch.

The effective horse-power in the air in this instance is

$$\frac{92{,}037 \times \cdot 86699 \times 5\cdot 2}{33{,}000} = 12\cdot 573$$

as compared with 45·104 in the second, and 6·7276 horse-power in the first arrangement.

The distribution of the air in the mine in the three arrangements is as follows—

Division of the Mine.	First arrangement with shafts A and C downcast and B and D upcast.		Second arrangement with shafts C and D downcast and A upcast, B being abandoned.		Third arrangement with shafts A and C downcast and D upcast, B being abandoned.	
	Volume of Air. Cubic feet per minute.	Pressure in overcoming friction. Lbs. per square foot.	Volume of Air. Cubic feet per minute.	Pressure in overcoming friction. Lbs. per square foot.	Volume of Air. Cubic feet per minute.	Pressure in overcoming friction. Lbs. per square foot.
In split E G H, corresponding to V G W...	9,282	·53394	19,598	2·2182	9,926	·61062
In split E J H, removed in 2nd and 3rd arrangements	10,718					
In split F K L............	12,046	·4496	14,619	·6621	7,337	·16681
„ F M L............	9,835		11,936		5,991	
„ N P Q, corresponding to D X P Y and E N P X............	11,116	·7658	22,020	3·0050	16,973	3·12419
In split N R Q, removed in 2nd and 3rd arrangements	12,836					
In split O S T............	14,426	·64483	13,138	·5347	28,522	2·5208
„ O U T............	11,778		10,726		23,288	
	92,037		92,037		92,037	

THE EFFECT OF OBSTRUCTIONS AND REGULATORS IN THE AIRWAYS.

Obstructions consisting of Falls of Roof and Sides.—The foregoing calculations are based on the supposition of the air being allowed to flow into the different divisions of a mine without check in any of the workings. In practice this rarely occurs, because adjustments of quantities require to be made to suit varying circumstances. Generally there are irregularities in the sectional area of airways in a mine, even if at first they are driven of one specific size. The effect of an obstruction such as a fall of the roof or sides is to restrict or altogether block the passage for air in the gallery where it occurs. Trams standing at sidings or in motion along the roadways are obstructions to the passage of air. The effective area of the downcast or upcast shaft may be much reduced by pipes, &c., laid in them and by the running of cages, although it is not usual to take into account such obstructions when calculating their frictional resistances.

In the case of a colliery having one cage in a small downcast shaft and another in the upcast, both worked by the same winding engine, there is a marked difference in the volume of air in the mine at each change in the direction of the winding. On the descent of the cage in the downcast shaft, the motion of both cages assists the ventilation; but on the reversal of the winding, the inflowing air is retarded by the ascent of the cage in the downcast, while at the same time the flow is checked in the upcast by the cage in its descent there.

A heavy fall of the roof in an airway may be considered in respect of its influence on the water-gauge of the whole colliery and on that of the split in which the fall occurs. If we imagine a case in which the air is carried round the workings in one continuous current, which plan of ventilation at one time prevailed, the effect of a fall forming a partial obstruction must be to increase the water-gauge. Precisely the same result would follow a fall in a main airway through which all the air of a mine was flowing. If a fan produces the ventilation and the same steam supply be continued for the use of the engine driving it, the speed of the fan is increased in consequence of the falling off in the quantity of air passing through the fan. A higher water-gauge would result from this increased speed, for the pressure depends upon the speed of the fan. A normal water-gauge may be maintained by reducing the steam supply, and, if the fan engine is controlled by a governor, on a reduction of the load of air to be dealt with by the fan a reduced steam supply follows.

Usually the fan engineman at a colliery has orders to maintain a certain speed or to keep a constant water-gauge. Under ordinary circumstances of the mine, regularity of speed ensures corresponding regularity of water-gauge, which may be taken as a standard for the engineman's guidance. Any deviation from this would be easily and quickly noticed by observations made in the fan engine-house. If there is an obstruction in the airway and the engine be kept at the same speed the water-gauge would be increased, but the quantity of air would be lessened. On the other hand, if doors were left open in the mine the water-gauge would be decreased, but the volume of air would become greater. The work of the fan may be assisted by dip workings, or retarded by workings to the rise.

If a furnace be used to produce the ventilation, the temperature in the upcast shaft after such a fall will be increased owing to the distribution of the heat of the fire throughout a smaller quantity of air. For the same coal supplied to the furnace there would be a higher water-gauge and a reduction in the velocity of the air. The ventilating pressure depends upon the temperature of the column of air in the upcast shaft. The precise effect on the water-gauge will be greater or less in accordance with the size of the fall and the amount of opening left for the passage of air.

A modern extensive mine, however, has many main splits of ventilation in one seam of coal or at different levels in two or more seams, and a heavy fall in one split may have no appreciable effect on the total water-gauge, whether the ventilation be produced by fan or by furnace. The greater the number of splits the less noticeable is the effect on the water-gauge if a fall occurs in one of them. If the fall is sufficient to entirely stop the ventilation of one split, a re-distribution of the air follows, by reason of which all other splits get a slightly increased quantity. The total increase in other splits, however, never amounts to the loss of air in the airway where the fall has occurred, and the total ventilation is therefore less. If the speed of the fan is regulated to maintain a constant water-gauge, the shaft friction is reduced, leaving a larger proportion of the total ventilating pressure to force air round the airways still unobstructed, each thus getting a slight increase in quantity. If the ventilation be commanded by a furnace and the weight of coal burnt be regulated to maintain a constant water-gauge the effect is the same.

It must be borne in mind that the water-gauge at any colliery is subject to frequent minor fluctuations above and below a certain standard height set up for constant guidance, and it is highly probable that where there are a number of splits a fall of roof in one does not cause perceptible change in the water-gauge or is not sufficient to cause alarm or likely to be attributed to any cause but that of constant minor fluctuations such as result from variation of the fan speed or irregularity of stoking the furnace.

We may consider the effect of a fall of roof on the water-gauge of a particular split by reference to Fig. 422O. If a heavy fall occurs at the point V or at Z it is immediately followed by a greatly reduced quantity of air in the split F K L, while at the same time there will be an increase in the water-gauge of that split. This would be apparent by testing the water-gauge before and after the occurrence of the fall at the point W, the instrument being applied at a stopping arranged for the purpose. There may be no marked change in the total water-gauge of the mine such as to arouse the suspicions of the officials, but the reduced velocity of the air-current in the split F K L would be so noticeable as to lead to quick discovery by those having the constant supervision of the airways. If water-gauges were kept continuously attached to all main separation doors in the connecting roads between the intakes and returns of a mine and the readings regularly registered, the fall in the split F K L would be indicated by the reading of that water-gauge placed at the point W. No useful purpose would be served by the placing of water-gauges in the workings, and it is customary only to fix them permanently in the manager's office and in the fan engine-house connected with the fan drift at the surface or in an office underground, the water-gauge being then connected by a pipe with the return airway. With a furnace ventilation, the water-gauge is usually applied on the separation doors between pits at the bottom seam or other levels. If the fall takes place at any point in the intake between F and W or in the return airway between the points Y and L there is a reduction in the water-gauge at the point W, and the larger the fall the greater will the reduction be, until, if the airway is entirely blocked, there will be no difference whatever in the two limbs of the water-gauge.

The exact effect of the fall can only be ascertained by a careful measurement of the water-gauge and of the quantity of air circulating before and after the fall. Thus, supposing a fall at the point V or at Z increases the water-gauge for the split F K L from 1 to 1·5 inch and reduces the quantity of air one-half, and that the water-gauge at the point W reads ·7 before the fall, then the friction in F W and Y L is equal to 1 − ·7 = ·3 inch of water-gauge before the fall takes place. Afterwards, when the quantity is reduced one-half, the friction in these airways will be reduced to ¼ or ·075 inch, whilst round W K Y the frictional resistances in the airways amount to ¼ of ·7 or ·175 inch. Together these resistances make ·25 inch due to the passage of the reduced quantity of air throughout the split. If the water-gauge has been raised to 1·5 inch, it is plain that 1·25 inch of water-gauge is the measure of the resistance due to the obstruction itself.

A slight modification of this calculation would be necessary for a fall so extensive as to reduce the area of the airway over a considerable proportion of its length. Such a fall would seriously reduce the length of the original airway and thereby decrease the frictional resistance of the airway considered apart from the obstructed portion.

*Regulators.**—We have endeavoured to show in preceding examples the quantities of air which would flow into the different splits of a mine with a given water-gauge, under the supposition that the air is free to flow into each without check other than that offered by the length and perimeter of the airway itself. We have also given examples showing the disturbing influence of falls in such airways. In few if any mines, however, is the air allowed to distribute itself unrestrainedly into the different districts. It may be that the district most remote from the shafts is that which requires the largest volume of air, either because gas is emitted there or because of its more extensive face room and number of workmen, horses, &c. It is possible, of course, that such long split would naturally receive the largest volume of air without any check being placed upon the short splits if its sectional area be so much larger as to reduce its frictional resistances sufficiently. Usually, however, the main roads in the same seam are nearly of the same size, being then best adapted for the passing traffic and for permanent maintenance, and other means are adopted to ensure a greater volume in a long split. As before stated, this is done by placing regulators in the shorter airways. An ordinary wooden door propped partly open will act as a regulator by allowing air to pass through the restricted passage. Similarly a piece of brattice cloth hung from the roof in a horse road will act as a regulator if it is too short to reach the floor or if an aperture be cut in it. The proper form for a main permanent regulator in an underground door is that shown in Fig. 388, which has been fully described. The sliding shutter in it allows of accurate adjustment, and may be afterwards kept locked to prevent unauthorised interference with it. A regulator may be formed on a main road by building brickwork or masonry, or by placing sheet iron or boards at the sides, leaving merely a central opening between for the passage of trams. Such devices, however, leave no convenient means for re-adjustment. Fig. 422R shows a form of regulator frequently used in the return air-courses only required for a travelling road. The framing is set in masonry, the sliding portion being similar to that described for Fig. 388. The amount of opening given to the regulator is fixed experimentally, careful measurements of the volumes of air being made in all splits after each single alteration, until the desired proportion is obtained.

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* A very useful and practical work on this subject, entitled "The Natural Philosophy of a Ventilating Regulator," has been written by H. W. Halbaum, and is published at 27, Wallgate, Wigan.

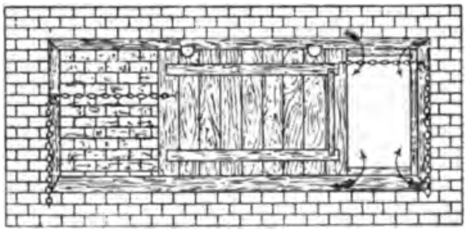

Fig. 422R.—Form of Regulator ordinarily used in the Return Air-courses

The laws which govern the flow of fluids through such openings as those formed by mine regulators are generally accepted as being applicable to the circulation of air through obstructions in the underground airways. Experiments have been made which support this theory. Wherever the gallery is restricted by a regulator placed in it or by one formed naturally the flow of air through the aperture is governed by the velocity of efflux of a fluid under pressure through an orifice in a thin plate. With an opening whose length is three times its diameter the flow is greater by about 30 per cent., but such openings do not accord with ordinary mine regulators. Precisely the same formula for the determination of the equivalent orifice of the whole mine given elsewhere may be applied to mine regulators, the co-efficient for the *vena contracta* being taken at ·65 and the velocity through the opening being dependent on a certain head. That a regulator absorbs pressure may easily be proved by placing a water-gauge upon it. There will certainly be a depression unless the restriction in the airway is so slight as to be unnoticeable. If the sliding shutter be more nearly closed so as to reduce the passage for air, the depression of the water-gauge is increased. If we take the case of a mine ventilated by one undivided current and place a regulator in the airway the water-gauge is thereby increased. If placed in the main airway of a mine through which all the air is flowing, the result will be the same. The fixing of a regulator on one of two splits reduces the quantity in its own airway while increasing that in the other, but not to the same extent. The effect of placing a regulator on any split of a mine or in any part of the workings is to reduce the total quantity of air going into the mine. Suppose a mine has ten free splits of ventilation and a regulator be placed in one. There immediately follows a re-distribution of the air, by reason of which all other splits get a slightly increased quantity, while in the regulated airway itself the quantity is reduced. The total increase in other splits, however, will not amount to the loss of air in the airway containing the regulator. Each alteration in the position of the sliding shutter of the regulator produces a fresh distribution of the air, and if regulators be introduced into the other airways fresh disturbances follow. Primarily, the object of regulators is to distribute quantities, although it is quite conceivable in the case of a fiery mine that the application of a regulator may be beneficial to the removal of fire-damp under certain conditions by its re-distribution of pressures. If, for instance, the gas were given off at a fault or slight blower in the face at such a pressure as to foul some working places without penetrating into the main intake airway, it might be possible to increase the pressure in that intake and reduce it in the return airway by means of a newly-fixed or more nearly closed old regulator there and thus induce a more ready flow of the gas into the return airway immediately behind the regulator. Although the quantity of air in that district would be reduced by the regulator, the re-distribution of pressure would be more favourable to the discharge of fire-damp into the return, which is its proper outlet. The quantity of gas in such case must necessarily be small, and it is by no means suggested that it is proper to attempt the clearing of gas by reducing the volume of air. As the difference between the pressures in the face and in the return airway of a district is increased by placing the regulator at the entrance to the return, and gas given off in the goaf is consequently encouraged to flow into the return, such placing is beneficial in a fiery mine. In a non-fiery mine a regulator in the return at a point more or less distant from its commencement will prove quite as efficacious in the distribution of quantity. If a district is not already regulated it is unlikely that such means would be resorted to for clearing out the gas, for in all probability the return airway would become too much charged with fire-damp for it to be safe to enter it. No advance should be made in any airway containing an explosive mixture, because of the risk of explosion, even with a securely protected safety lamp, or of the risk to life by suffocation if made without a light. A serious blower not only quickly fouls the return airway, but issues with such force as to overcome and push back the intake air to a point more or less remote. It can only be cleared with difficulty by concentrating a large volume of air into the district, and even then time must be given for the blower to lessen or exhaust itself.

Supposing Fig. 422s to represent a plan of pillar and stall workings divided into two ventilating currents. A A′ being the main intake, the split taking place as shown by the arrows, one current of air returning round workings to points F and G and the other round workings to points D and E, both splits being of equal length and similar in all respects. If, say, 10,000 cubic feet of air per minute passed round each district with 2·5 inches of water-gauge for the whole of the split from a point outside A to its junction with another air-course outside the points B and C, the water-gauge from the junction outside the point A to point S being 1 inch, from the point S to F or S to D each being ·8 inch, and that between the points F and G or D and E each ·2 inch, while from the point G to the junction outside B, or from E to the junction outside C the water-gauge is ·5 inch. Now, let regulators be fixed, one at the point G and the other at E, and the quantity round each district be thereby reduced one-half. This would reduce the water-gauge for the airway to ·625 inch, and if we maintain the same total water-gauge there will be expended 2·5 − ·625 = 1·875 inch on the regulators. We may now compare the pressures at the various points. Before the regulators were put in the pressure at the point S was 2·5 − 1· × 5·2 = 7·8 lbs. per square foot, at the point F or D 2·5 − 1·8 × 5·2 = 3·64 lbs. per square foot, and at the point G or E 2·5 − 2 × 5·2 = 2·6 lbs. per square foot. Afterwards the pressure at the point S becomes 2·5 − ·25 × 5·2 = 11·7 lbs. per square foot, at the point F or D 2·5 − ·45 × 5·2 = 10·66 lbs., and at the point G or E 2·5 − ·5 × 5·2 = 10·4 lbs. Immediately on the outbye side of the regulator at the point G or E, the pressure becomes 2·5 − (·5 + 1·875) × 5·2 = ·65 lb. per square foot. The difference of pressure then between that point and S is 11·7 − ·65 = 11·05 lbs., as compared with 7·8 − 2·6 = 5·2 lbs., the difference between the same points before the regulators were put in. The difference of pressure between the points F and S or D and S is only 11·7 − 10·66 = 1·04 lb., as compared with 7·8 − 3·64 = 4·16 lbs., the difference between the same points before the regulators were put in. If the coal is unwrought at the points Q and R, and the pillars H, J, K, L, M, N, O, P have been worked, the difference of pressure between S and F or S and D before regulators were put in would be more favourable to the escape of gas from the goaf or from a break at the face into the return between F and G or between D and E than afterwards. But if the regulators be placed at F and D the reverse holds good, and the difference of pressure becomes more favourable to the escape of gas into the return airway F G or D E. On the outbye sides of the regulators the pressure would be 2·5 − (·45 + 1·875) × 5·2 = ·91 lb., and the difference of pressure between those points and S would be 11·7 − ·91 = 10·79 lbs. If the mine be fiery and these districts have to be regulated, there is thus an advantage in placing the regulators at F and D in preference to G and E, because fire-damp issues will more easily escape into the return airway so.

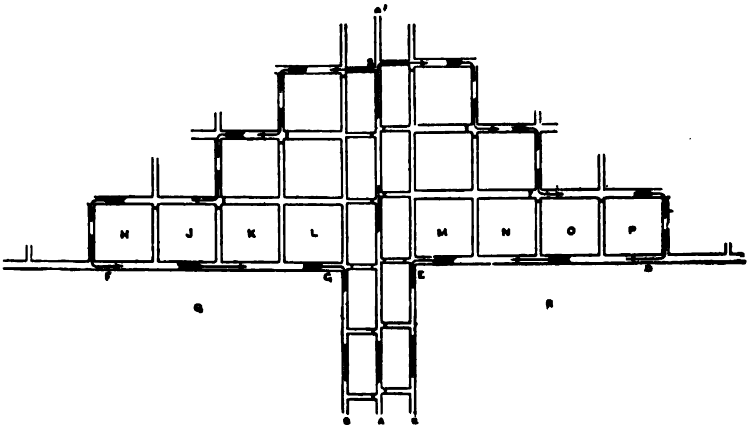

Fig. 422S.—To Illustrate the Effect of Regulators Placed in the Airways of a Mine.

Again, take the case of a colliery working the whole and broken mine together, as in Fig. 422T. Here the air passes from the downcast shaft A along the intake airway C to D, continuing through workings in the whole mine E, F, G, and from thence through broken workings to the point H, continuing through the points J, K to and up the upcast shaft B. A regulator, whether placed at E, at J, or at K, would have the same effect in reducing the quantity in the district. But by placing it at H, there is a better distribution of pressures tending to assist the flow of gas from the goaf into the airway behind the regulator, and this then is its proper position in a fiery mine.

It will be readily seen that with the two regulated districts shown in Fig. 422S whether each receives an equal quantity of air or not; either a fall or an obstruction placed in the airway will cause an alteration in the quantity of both. Bratticing the fore-winning place S A will cause slight variations in the two volumes, and this will be more marked in the case of an improperly placed brattice. Fig. 422U shows enlarged views of the fore-winning place. In A the whole of the intake air is made to pass behind the brattice before splitting takes place. When fresh holings to the right and left are made, this brattice is removed and the air is split direct without this restriction, and as the fore-winning place is continued the brattice is re-erected inside and lengthened with the advance of the driving from time to time. There will consequently be slight disturbances in the quantities of air in the two districts from time to time. In B, C and D the partition is better arranged and the air is split before reaching the brattice. Only one division of the air passes into the reduced sectional area of the passage to keep the face clean. B shows this division of the air as being taken in along the smaller area of the driving, and C shows it taken in along the larger sectional area, while D shows an arrangement if holings are opposite each other. There will be slight variations in volumes even in these arrangements, because of the distance holings are apart and the constantly increasing length of brattice between any two of such holings. Moreover, the back-winning places and the whole " sheth " of bords on each side of the fore-winning places may have to be bratticed to keep the faces clean, and there must be slight variations of the quantities in the two districts due to alterations

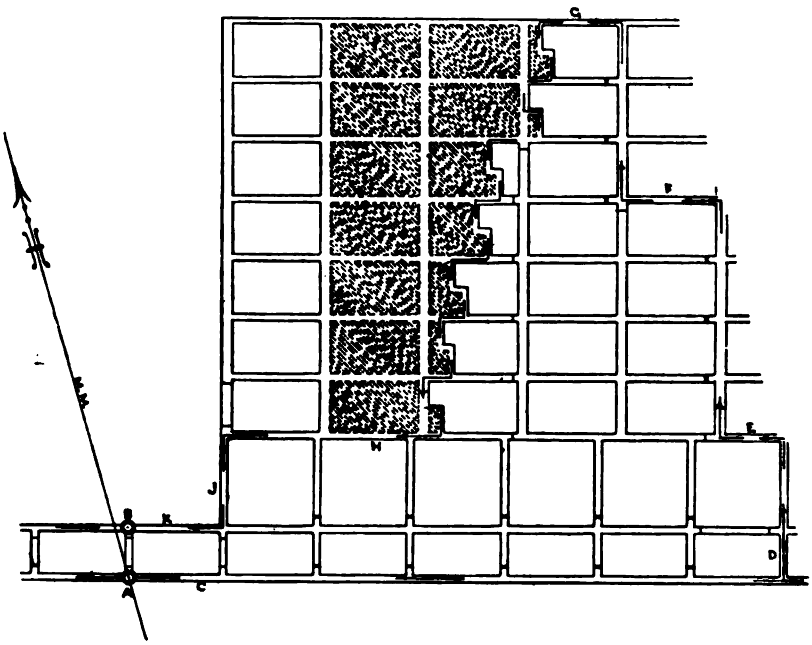

Fig. 422T.—To Illustrate the Effect on the Ventilation of a Colliery of placing a Regulator in an Airway, and to show its proper Position.

in the relative amounts of brattice in them which are subject to constant changes. Bratticing the fore-winning place in accordance with A, Fig. 422U, however, is likely to make greater fluctuations in quantities of air than that arising from bratticing the bords, because of oftentimes passing the whole volume of air which feeds the two districts behind the brattice, while only occasionally splitting the air into two full-sized airways in the entire absence of brattice.

Opinions differ as to whether the intake air-current should enter by the smaller or larger side of the partition. Theoretically, as the current expands in its course round the workings, the larger volume should return through the larger section of passage. Practically, it will not matter much, so far as quantity is concerned, which side of the brattice is made the intake; the resistances on both sides must be overcome in turn. Experience seems to show that under certain conditions some benefit to purification of the face is derived by assigning the return air to the smaller section of passage, because the pressures are then distributed with the best possible effect. In workings swept by a brisk current of air we prefer making the smaller passage the intake. If we take the case of a drift in which a great length of brattice has to be carried, either in the form of a partition, air-tubes, or pipes, as in Fig. 422V, generally a much more feeble current of air is in circulation than in workings which have only short lengths of brattice. As regards quantity, it would appear not to matter whether the air is coursed as at A or as at B in Fig. 422V. If a small fan has to be worked, however, it will be a better arrangement for it to exhaust the air as at A than to force it into the drift as at B, and this arrangement may tend to purification of the air at the face, because the greatest possible pressure is brought on the face of the drift, and the difference between this pressure and that at the entrance to the tubes encourages a flow of vaporous particles into the return.

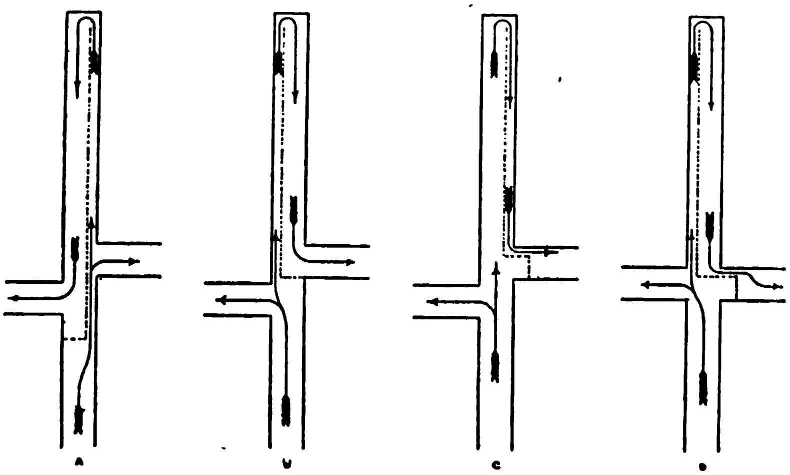

Fig. 422U.—To Illustrate the Effect of Obstructions formed by Brattice in the Airways of a Mine.

The effect of placing a regulator in any mine cannot be calculated without taking into consideration the whole circumstances of the mine, the depth and size of shafts, and full particulars of the airways leading from the shafts before and after splitting takes place. Splits subject to a common ventilating pressure must have that pressure increased by placing a regulator in any one of them. The increased resistance of the regulator reduces the quantity of air in the regulated split and also the total quantity of air circulating in the mine. The increased pressure in the unregulated splits increases their ventilation; this increase will be greater or less, in accordance with the length and dimensions of airways between the junction of splits and the shafts, and also by the number and particulars of all other splits in the mine. The total ventilating pressure of the mine must be increased by placing a regulator in any split.

Supposing we have a downcast and an upcast shaft, each of which is 500 yards deep and 10 feet in diameter, connected by a single gallery 1,000 yards long, its dimensions being 8 feet by 6. For a 10-lb. pressure in the airway we should have as the velocity

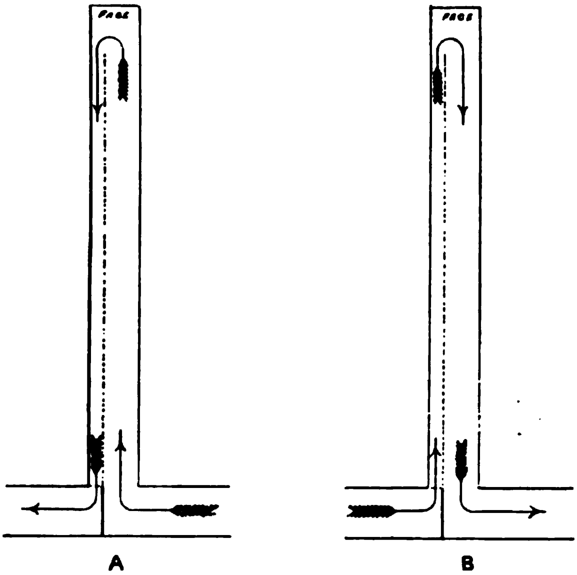

Fig. 422v.—The Ventilation of a Stone Drift or Driving in the Coal by Air-boxes, Pipes, or Brattice.

$$v = \sqrt{\frac{10 \times 48}{\cdot 0000000217 \times 84{,}000}} = 513\cdot 16 \text{ feet per minute,}$$

and $q = 513\cdot 16 \times 48 = 24{,}631$ cubic feet per minute.

The pressure in each shaft would be

$$p = \frac{(\cdot 00000001) \times (500 \times 3 \times 31\cdot 416) \times (24{,}631)^2}{78\cdot 54^3} = \cdot 59 \text{ lb. per square foot.}$$

The total pressure would be:

For the gallery	10·
,, downcast shaft	·59
,, upcast ,,	·59
	11·18 lbs. per square foot.

Let us now calculate the effect of placing a regulator in the airway with different areas of opening, first to command a ventilating current of 20,000 cubic feet, secondly of 10,000 cubic feet, and thirdly of 5,000 cubic feet per minute, on the supposition that the same total ventilating pressure is maintained in each case.

If the ventilation be reduced to 20,000 cubic feet per minute, the pressure necessary to overcome the friction in each shaft becomes

As 24,631^2 : 20,000^2 :: ·59 : ·389 lb.

For the two shafts it would be ·389 × 2 = ·778, leaving 11·18 − ·778 = 10·402 lbs. as the pressure for the airway. The pressure, however, necessary to overcome the road resistance must be

As 24,631^2 : 20,000^2 :: 10 lbs. : 6·593 lbs.,

and therefore that expended on the regulator must be 10·402 − 6·593 = 3·809 lbs., or $\frac{3{\cdot}809}{5{\cdot}2} = {\cdot}7325$ inch of water-gauge. According to formula the regulator area must be $\frac{{\cdot}388 \times 20}{\sqrt{{\cdot}7325}} = 9{\cdot}067$ square feet.

If the quantity be reduced to 10,000 cubic feet, the pressure in each shaft is

As 20,000^2 : 10,000^2 :: ·389
or „ 4 : 1 :: ·389 : ·0973 lb., or in both shafts ·0973 × 2 = ·1946 lb., leaving 11·18 − ·1946 = 10·9854 lbs. as the pressure for the airway. Only a portion of this is absorbed by the road resistances, thus:—

As 20,000^2 : 10,000^2 :: 6·593
or „ 4 : 1 :: 6·593 : 1·6482 lbs.

Consequently the pressure expended on the regulator must be 10·9854 − 1·6482 = 9·3372 lbs., or $\frac{9{\cdot}3372}{5{\cdot}2} = 1{\cdot}7956$ inch of water-gauge. The regulator area then must be $\frac{{\cdot}388 \times 10}{\sqrt{1{\cdot}7956}} = 2{\cdot}895$ square feet.

If the quantity be reduced to 5,000 cubic feet per minute, the pressure in each shaft is

As 10,000^2 : 5,000^2 :: ·0973
or „ 4 : 1 :: ·0973 : ·02432 lb., or in both shafts ·02432 × 2 = ·04864 lb., leaving 11·18 − ·04864 = 11·13136 lbs. as the pressure for the airway. The portion of this absorbed by road resistances is

As 10,000^2 : 5,000^2 :: 1·6482
or „ 4 : 1 :: 1·6482 : ·412 lb.

The pressure spent on the regulator then is 11·13136 − ·412 lbs. = 10·71936 lbs. or $\frac{10{\cdot}71936}{5{\cdot}2} = 2{\cdot}0614$ inches of water-gauge. The area of opening in the regulator then must be $\frac{{\cdot}388 \times 5}{\sqrt{2{\cdot}0614}} = 1{\cdot}3512$ square feet.

But it is unlikely that the total ventilating pressure *would* be maintained after fixing the regulator. Possibly the power producing the ventilation might be kept constant. On this assumption we may calculate the difference in the total ventilating pressures with the reduced quantities circulating. The power u varies in accordance with the cube of the quantity, or q^3. The horse-power is $\frac{p \times q}{33{,}000}$. Before the regulator was fixed then the horse-power was $\frac{11{\cdot}18 \times 24{,}631}{33{,}000} = 8{\cdot}345$. With the regulator fixed to reduce the current of ventilation to 20,000 cubic feet and the same total pressure the horse-power would be $\frac{11{\cdot}18 \times 20{,}000}{33{,}000}$ or as 24,631 :

20,000 :: 8·345 : 6·775. If the horse-power is to be maintained at 8·345 for the 20,000 cubic feet we have—

$$8{\cdot}345 = \frac{p \times 20{,}000}{33{,}000} \text{ whence } p = \frac{8{\cdot}345 \times 33{,}000}{20{,}000} = 13{\cdot}77 \text{ lbs. per square foot.}$$

With the regulator opening reduced until 10,000 cubic feet sweep the airway and with the same horse-power we have $p = \frac{8{\cdot}345 \times 33{,}000}{10{,}000} = 27{\cdot}54$ lbs. With the regulator opening still further reduced to pass 5,000 cubic feet, for the same horse-power we have $p = \frac{8{\cdot}345 \times 33{,}000}{5{,}000} = 55{\cdot}07$ lbs., or a water-gauge of 10·59 inches. Such a high water-gauge is rarely found in any mine.

If instead of a single gallery communicating between an upcast and a downcast shaft, let us take an instance of two splits of air commencing at the downcast and ending at the upcast. Each shaft is 500 yards deep and 10 feet in diameter as in the last instance, and each airway is 8 feet by 6, but one is 1,000 yards long and the other 500.

If we take a common pressure of 10 lbs. for the two splits, then in the long airway—

$$v = \sqrt{\frac{10 \times 48}{{\cdot}0000000217 \times 84{,}000}} = 513{\cdot}16 \text{ feet per minute,}$$

$$\text{and } q = 513{\cdot}16 \times 48 = 24{,}631 \text{ cubic feet per minute.}$$

In the 500-yard airway, by the formula $R = \sqrt{\frac{a^3}{s}}$ the quantity passing is as $\sqrt{\frac{1}{2}} : \sqrt{\frac{1}{1}}$ or as $\sqrt{{\cdot}5} : 1 :: 24{,}631 : 34{,}834$ cubic feet. The volume of air in each shaft is 24,631 + 34,834 = 59,465 cubic feet per minute.

The pressure in each shaft for this quantity would be—

$$p = \frac{({\cdot}00000001) \times (500 \times 3 \times 31{\cdot}416) \times (59{,}465)^2}{78{\cdot}54^3} = 3{\cdot}439 \text{ lbs.}$$

$$\text{or as } 24{,}631^2 : 59{,}465^2 :: {\cdot}59 : 3{\cdot}439 \text{ lbs. per square foot.}$$

The total pressure would be—

For the two splits	10·
,, ,, downcast shaft	3·439
,, ,, upcast ,,	3·439
	16·878 lbs. per square foot.

If now a regulator be placed in the short airway it must cause a re-distribution of pressures. Supposing it is so arranged as to pass an equal quantity with the long split, the resistance of the regulator then is equal to the difference of the resistances in the two airways. For a pressure of 10 lbs. in these, the quantities passing must be 24,631 cubic feet per minute in each airway. The frictional resistances in the short airway must be half that for the long, because they pass equal quantities and are similar in area and perimeter, and the frictional resistances must therefore be proportional to their lengths. That is, as 1,000 : 500, or as 2 : 1 :: 10 : 5 lbs. for the 500-yard airway. This leaves 10 − 5 = 5 lbs. as the resistance of the regulator.

But it is impossible that the shaft resistances can remain the same because of the reduced quantity now circulating in them. There would only be a total volume of 24,631 + 24,631 = 49,262 as compared with 59,465 cubic feet per minute in the shafts before the regulator was put in. The pressure for each shaft must be in proportion to the square of the quantity, and would therefore be

$$\text{As } 59{,}465^2 : 49{,}262^2 :: 3{\cdot}439 : 2{\cdot}36 \text{ lbs.}$$

If the pressure for the splits is to remain the same as before, the total pressure must be reduced, because $10 + 2{\cdot}36 + 2{\cdot}36 = 14{\cdot}72$ lbs. as compared with 16·878 before the regulator was put in. The horse-power of ventilation was then

$$\frac{16{\cdot}878 \times 59{,}465}{33{,}000} = 30{\cdot}413.$$

If this same power is maintained to produce 49,262 cubic feet of air, then $p = \frac{30{\cdot}413 \times 33{,}000}{49{,}262} = 20{\cdot}373$ lbs., of which $2{\cdot}36 + 2{\cdot}36 = 4{\cdot}72$ lbs. are absorbed by the shafts. This leaves $20{\cdot}373 - 4{\cdot}72 = 15{\cdot}653$ lbs. as the pressure for the splits. The quantity of air in the long airway then is

$$\text{As } \sqrt{10} : \sqrt{5{\cdot}653} :: 24{,}631 : 30{,}817 \text{ cubic feet per minute.}$$

or we may say the velocity in the air for the long airway

$$\text{is } \sqrt{\frac{15{\cdot}653 \times 48}{{\cdot}0000000217 \times 84{,}000}} = 642{\cdot}02 \text{ feet per minute.}$$

and the quantity passing is $642{\cdot}02 \times 48 = 30{,}817$ cubic feet per minute. It is impossible, therefore, if the total quantity is 49,262 cubic feet per minute after the regulator is fixed, and the same power of ventilation maintained, for the two splits to have equal quantities of air in them. The quantity for the shorter must be $49{,}262 - 30{,}817 = 18{,}445$ cubic feet per minute. The pressure necessary to overcome the frictional resistance in the 500-yard airway due to this quantity is

$$p = \frac{({\cdot}0000000217) \times (500 \times 3 \times 28) \times (18{,}445)^2}{48^3} = 2{\cdot}8038 \text{ lbs.}$$

The pressure expended on the regulator then is $15{\cdot}653 - 2{\cdot}8038 = 12{\cdot}8492$ lbs., or $\frac{12{\cdot}8492}{5{\cdot}2} = 2{\cdot}471$ inches of water-gauge. The regulator area then must be

$$\frac{{\cdot}388 \times 18{\cdot}445}{\sqrt{2{\cdot}471}} = 4{\cdot}5528 \text{ square feet.}$$

If we wish to maintain the same total ventilation of 59,465 cubic feet of air per minute in the shafts, and to split it into two even volumes by placing a regulator in the short split, we shall now show that it can only be done at an increase in the ventilating power. The pressure for the shafts would be 3·439 lbs. for the downcast and 3·439 lbs. for the upcast, together making 6·878 lbs. per square foot. The volume of air for each split is $\frac{59{,}465}{2} = 29{,}732$ cubic feet per minute. The pressure necessary to overcome frictional resistances in the 1,000-yard airway is as $24{,}631^2 : 29{,}732^2 :: 10 : 14{\cdot}57$ lbs. That for the shafts being 6·878, we have now a total ventilating pressure of $6{\cdot}878 + 14{\cdot}57 = 21{\cdot}448$ lbs., as compared with 16·878 lbs. pressure for the same quantity of air before the regulator was put in. The pressure spent in overcoming the frictional resistances of the 500-yard airway would be one-half that needed for the 1,000-yard airway, viz., $\frac{14{\cdot}57}{2} = 7{\cdot}285$ lbs., the other 7·285 common to the splits being expended on the regulator. This is equivalent to a water-gauge of $\frac{7{\cdot}285}{5{\cdot}2} = 1{\cdot}4$ inch. The area of opening in the regulator then must be $\frac{{\cdot}388 \times 29{\cdot}732}{\sqrt{1{\cdot}4}} = 9{\cdot}746$ square feet. The horse-power of ventilation has been increased from 30·413 to $\frac{21{\cdot}448 \times 59{,}465}{33{,}000} = 38{\cdot}65$, or directly in proportion to the increased pressure if the quantity remains the same, viz., as $16{\cdot}878 : 21{\cdot}448 :: 30{\cdot}413 : 38{\cdot}65$.

If a regulator be once set to command a certain proportion of the air in an airway, that proportion will be maintained, even if the total pressure be altered. For instance, take the last example, where, with a regulator opening of 9·746 square feet in the 500-yard airway, 29,732 cubic feet per minute are passing in each split, the total pressure being 21·448 lbs. Let us assume that the total ventilating pressure be reduced one-half, viz., to 10·724 lbs. Since p varies as q^2 we should have in the shafts—

$$\text{As } \sqrt{21{\cdot}448} : \sqrt{10{\cdot}724} :: 59{,}465$$

$$\text{or, as } \sqrt{2} : \sqrt{1} : 59{,}465 : 42{,}048 \text{ cubic feet per minute.}$$

The pressure for each shaft then is one-half of what it previously was, or

$$\text{As } 59{,}465^2 : 42{,}048^2 :: 3{\cdot}439 : 1{\cdot}72 \text{ lb.}$$

For the two shafts it would be 3·44 lbs., leaving $10{\cdot}724 - 3{\cdot}44 = 7{\cdot}284$ lbs. as the pressure for the splits, or one-half of what it previously was.

The quantity in the long airway will be

$$\text{As } \sqrt{10} : \sqrt{7{\cdot}284} :: 24{,}631 : 21{,}022 \text{ cubic feet per minute.}$$

The pressure spent in overcoming the frictional resistances of the 500-yard airway would be one-half of that required for the 1,000-yard split, for an equal quantity of air, that is $\frac{7{\cdot}284}{2} = 3{\cdot}642$ lbs., the other 3·642 lbs. being spent on the regulator. This is equivalent to a water-gauge of $\frac{3{\cdot}642}{5{\cdot}2} = {\cdot}7004$ inch. The quantity of air in thousands of cubic feet per minute, which would pass through a regulator having an opening of 9·746 square feet, with ·7004 inch of water-gauge, is

$$\frac{9{\cdot}746 \times \sqrt{{\cdot}7004}}{{\cdot}388} = 21{\cdot}022,$$

which is the same quantity of air as that passing into the longer airway with the altered pressure, showing that the same proportion is maintained.

Let us now take another example, in which the size of each shaft and its depth is the same as in the last example, and also in which there are two splits of air, which, however, do not commence at the downcast, but 100 yards away from it, nor end at the upcast, but converge 100 yards distant from it. These outer airways are, say, each 8 feet square, and the split airways are 8 feet by 6, and 500 and 1,000 yards long respectively, as in the last example.

With a common pressure of 10 lbs. for the splits, we should have 24,631 cubic feet per minute in the long airway, and 34,834 in the short one as before, making a total of 59,465. The pressure in each shaft for this quantity would be 3·439 lbs. as before. That for each 100-yard outer airway is

$$p = \frac{({\cdot}0000000217) \times (100 \times 3 \times 32) \times (59{,}465)^2}{64^3} = 2{\cdot}81 \text{ lbs.}$$

The total pressure would be

For the two splits	10·	lbs.
„ „ outer return airway	2·81	„
„ „ „ intake „	2·81	„
„ „ downcast shaft	3·439	„
„ „ upcast „	3·439	„
	22·498	lbs. per square foot.

If a regulator be placed in the short airway, there must of necessity be a redistribution of pressures. If so arranged that equal quantities ventilate the splits, and the pressure of 10 lbs. be maintained in them, then there must be 24,631 cubic feet per minute in each. But there must in that case be less pressure required for the outer airways and the shafts, because the quantity in them is reduced from 59,465 to 49,262 cubic feet per minute. The total pressure must be reduced after the regulator is put in, if the pressure of the splits remains constant.

If the total ventilation be maintained at 59,465 cubic feet per minute, and is split into two equal quantities by placing a regulator in the short split, we should have the following pressures—

For the outer return airway	2·81
,, ,, ,, intake ,,	2·81
,, ,, downcast shaft	3·439
,, ,, upcast ,,	3·439
,, ,, two splits	14·57
Total	27·068 lbs. per square foot.

The total pressure before the regulator was fixed was 22·498, and has since become 27·068 lbs., for the same quantity of air. The pressure necessary to overcome the frictional resistances of the 500-yard airway would be one-half that needed for the 1,000-yard airway, or 7·285 lbs., and there would be 7·285 expended on the regulator. The area of opening must be 9·746 square feet, as calculated in the previous example. Before the regulator was put in, the horse-power of ventilation was $\frac{22·498 \times 59,465}{33,000} = 40·54$, and has since become $\frac{27·068 \times 59,465}{33,000}$ $= 48·78$.

If the horse-power be reduced to what it was before the regulator was put in, we should have as the quantity circulating—

$$\text{As } \sqrt[3]{48·78} : \sqrt[3]{40·54} :: 59,465 : 55,910 \text{ cubic feet per minute.}$$

On the assumption that the horse-power of ventilation is the same before and after the regulator is fixed, its effect is to reduce the total quantity of air from 59,465 to 55,910 cubic feet per minute. On the assumption that the quantity is maintained, the horse-power of ventilation must be increased from 40·54 to 48·78, such increase being due to the resistance offered by the regulator.

Let us take as another example, the case of two shafts of the same depth and diameter as before, viz., 500 yards and 10 feet each. The main intake is again 100 yards long and 8 feet square, the main return being of similar length and size. Inside these two outer airways the air is split into five distinct currents. These five airways are each 6 feet square, but of unequal length, No. 1 being 200 yards, No. 2, 400 yards, No. 3, 600 yards, No. 4, 800 yards, and No. 5, 1000 yards. It is desired to so regulate the air as to give each an equal volume.

If we take the same total quantity of air in circulation as in the last example, viz., 59,465 cubic feet per minute before any regulators are fixed the relative volumes for the splits are—

For No. 1 split, $\sqrt{\frac{36^3}{2}} = 152·735$

,, ,, 2 ,, $\sqrt{\frac{36^3}{4}} = 108·$

,, ,, 3 ,, $\sqrt{\frac{36^3}{6}} = 88·182$

,, ,, 4 ,, $\sqrt{\frac{36^3}{8}} = 76·368$

,, ,, 5 ,, $\sqrt{\frac{36^3}{10}} = 68·305$

Total 493·59

For a total ventilation of 59,465 cubic feet per minute, the proportion going into each airway must be—

For No. 1 split, As 493·59 : 152·735 :: 59,465 : 18,401
„ „ 2 „ „ 493·59 : 108· :: 59,465 : 13,011
„ „ 3 „ „ 493·59 : 88·182 :: 59,465 : 10,624
„ „ 4 „ „ 493·59 : 76·368 :: 59,465 : 9,200
„ „ 5 „ „ 493·59 : 68·305 :: 59,465 : 8,229
Total 59,465

This proportion of total volume will be found to hold equally good for airways of similar lengths to these and other area, so long as all five airways are equal in area and perimeter.

We may now find the value of *p* for the splits, by using the figures of one.

For No. 5 split, $p = \frac{(\cdot0000000217) \times (1{,}000 \times 3 \times 6 \times 4) \times (8{,}229)^2}{36^3} = 2\cdot2673$ lbs.

The total pressure for the mine then would be—

For the split airways	2·2673
„ „ outer return airway	2·81
„ „ „ intake „	2·81
„ „ downcast shaft	3·439
„ „ upcast „	3·439
	14·7653 lbs. per square foot.

By increasing the number of splits of the given dimensions from two to five, other things remaining as before, the ventilating pressure would be reduced from 22·498 to 14·7653 lbs. per square foot.

If it is desired to pass equal volumes of air into each split, regulators must be placed in all except the longest, and the area of opening in each will differ, because the resistance offered by each regulator must be such as to equal the difference between the frictional resistances of the longest airway and that airway in which the regulator is placed. Let us assume that we maintain the quantity of 8,229 cubic feet per minute in No. 5 airway, and fix a regulator in No. 4 to reduce its quantity to equal that of No. 5. The frictional resistances for No. 5 are equal to a pressure of 2·2673 lbs. and therefore those for No. 4 will be

As 10 : 8 :: 2·2673 : 1·8138 lb.

The resistance of the regulator in No. 4 airway then must be equal to 2·2673 − 1·8138 = ·4535 lb. or $\frac{\cdot4535}{5\cdot2} = \cdot08721$ inch of water-gauge. The area of opening in the regulator must be—

$$\frac{\cdot388 \times 8\cdot229}{\sqrt{\cdot08721}} = 10\cdot811 \text{ square feet.}$$

Dealing next with No. 3 airway, its frictional resistances are—

As 10 : 6 :: 2·2673 : 1·3604 lb.

The resistance of the regulator in No. 3 airway then must be equal to 2·2673 − 1·3604 = ·9069 lb., or $\frac{\cdot9069}{5\cdot2} = \cdot1744$ inch of water-gauge. The area of the regulator opening must be—

$$\frac{\cdot388 \times 8\cdot229}{\sqrt{\cdot1744}} = 7\cdot645 \text{ square feet.}$$

The frictional resistances of No. 2 airway are—

As 10 : 4 :: 2·2673 : ·9069 lb.

Therefore, the pressure produced by the resistance of the regulator in No. 2 airway is 2·2673 − ·9069 = 1·3604 lbs., or $\frac{1\cdot3604}{5\cdot2} = \cdot2616$ inch of water-gauge. The area of this regulator opening must be—

$$\frac{\cdot388 \times 8\cdot229}{\sqrt{\cdot2616}} = 6\cdot242 \text{ square feet.}$$

The frictional resistances of No. 1 airway are—

As 10 : 2 :: 2·2673 : ·4535 lb.

The pressure produced by the regulator in No. 1 airway is 2·2673 − ·4535 = 1·8138 lb., or $\frac{1\cdot8138}{5\cdot2} = \cdot3488$ inch of water-gauge. The area of opening then is—

$$\frac{\cdot388 \times 8\cdot229}{\sqrt{3488}} = 5\cdot406 \text{ square feet.}$$

But if the pressure for the split airways is the same after the regulators are fixed as before, viz., 2·2673 lbs., it is impossible for the pressure in the shafts and outer airways also to remain unaltered, because the total volume of air is now decreased from 59,465 to five equal splits of 8,229 cubic feet each, or a total of 41,145 cubic feet per minute. The pressure in each shaft for such quantity is

As $59{,}465^2 : 41{,}145^2$ :: 3·439 : 1·6464 lb.

and in each outer airway „ $59{,}465^2 : 41{,}145^2$ :: 2·81 : 1·3453 lb.

The total pressure of the mine now would be—

For the split airways	2·2673
„ outer return airway	1·3453
„ „ intake „	1·3453
„ downcast shaft	1·6464
„ upcast „	1·6464
	8·2507 lbs. per square foot.

If we raise the total pressure to what it was previous to the placing of regulators, we shall have in circulation—

As $\sqrt{8\cdot2507} : \sqrt{14\cdot7653}$ :: 41,145 : 55,042 cubic feet per minute.

If the regulators remain undisturbed, equal quantities will continue to pass into each split, so that if the total quantity be 55,042 cubic feet, each split receives $\frac{55{,}042}{5} = 11{,}008$ cubic feet per minute. That the regulator openings which have been fixed to give certain proportions of air in the splits, continue to give the same proportion if the total quantity be varied, may be proved by assuming the total quantity of 41,145 to be decreased to one-fourth of this or 10,286 cubic feet per minute. Each split should then receive $\frac{10{,}286}{5} = 2{,}057$ cubic feet per minute or one-fourth of its original volume. The value of p for the longest split then would be—

As $4^2 : 1^2$ :: 2·2673 : ·1417 lb.

The frictional resistances in No. 4 airway will be—

As 10 : 8 :: ·1417 : ·1134 lb.,

and the resistance of the regulator placed there must be equal to ·1417 − ·1134 = ·0283 lb. or $\frac{\cdot0283}{5\cdot2} = \cdot00544$ inch of water-gauge. The area of opening in the regulator then must be $\frac{\cdot388 \times 2\cdot057}{\sqrt{\cdot00544}} = 10\cdot81$ square feet, or the same as calculated when each split received 8,229 cubic feet per minute. If the openings for the other regulators be calculated they will be found of the same size as already fixed for the higher quantity of air. Having once adjusted regulators in the mine to distribute the air in definite proportions each district will continue to receive the same definite proportion of the total ventilation so long as the circumstances of the mine remain unaltered, even if the total volume be varied from time to time. The introduction of a fresh regulator must of course cause an entirely fresh redistribution of the air.

Turning now to the example previously worked out and illustrated by Fig. 422P. Here the following volumes of air were shown to be in circulation—

In downcast shaft D	45,884	cubic feet per minute.
„ „ „ C	46,153	„ „ „ „
„ upcast „ A and airway B A	92,037	„ „ „ „

The total ventilation is subdivided in the mine as follows—

In O S T	13,138	cubic feet per minute.
„ O U T	10,726	„ „ „ „
„ D O and T Y	23,864	„ „ „ „
„ D X P Y	22,020	„ „ „ „
Total volume of air in shaft D	45,884	„ „ „ „

In F K L	14,619	cubic feet per minute.
„ F M L	11,936	„ „ „ „
„ V F and L B	26,555	„ „ „ „
„ V G W	19,598	„ „ „ „
Total volume of air in shaft C	46,153	„ „ „ „

The pressures necessary for the production of these volumes are—

For splits O S T and O U T	·5347	lbs. per square foot.
„ „ D O and T Y	2·4703	„ „ „ „
„ „ D X P Y	3·0050	„ „ „ „
From Y to W	10·7644	„ „ „ „
For C V and V G W (V G W being 2·2182)	13·7694	„ „ „ „
„ W B	·6287	„ „ „ „
„ B A	·7479	„ „ „ „
„ shaft A	·6104	„ „ „ „
„ „ D	·1452	„ „ „ „
„ „ C	·2687	„ „ „ „
	16·1703	„ „ „ „

For C V	11·5512	lbs. per square foot.
„ F K L and F M L	·6621	„ „ „ „
„ V F and L B	2·1851	„ „ „ „
From C to B	14·3984	„ „ „ „
For B A	·7479	„ „ „ „
„ shaft A	·6104	„ „ „ „
„ „ D	·1452	„ „ „ „
„ „ C	·2687	„ „ „ „
	16·1706	„ „ „ „

Suppose now it were desirable for all the districts of the mine to receive equal volumes of air.

As the district O U T naturally takes the smallest volume of air, let us take the quantity of 10,726 cubic feet per minute going into it as a standard, and regulate the other districts to have an equal air supply. The shaft D, being the source of supply for three districts, would admit 10,726 × 3 = 32,178 cubic feet of air per minute, and the pressure in it would be As $45{,}884^2 : 32{,}178^2 :: \cdot 1452 : \cdot 07141$ lb. Dealing first with the airway O S T, which must have a regulator placed in it in order to reduce its volume to 10,726 cubic feet per minute, the frictional resistance of the airway must be As $13{,}138^2 : 10{,}726^2 :: \cdot 5347 : \cdot 3564$ lb. The resistance of the regulator in O S T, then, must be equal to a pressure of $\cdot 5347 - \cdot 3564 = \cdot 1783$ lb., or $\frac{\cdot 1783}{5\cdot 2} = \cdot 03429$ inch of water-gauge. The area of opening in the regulator must be $\frac{\cdot 388 \times 10\cdot 726}{\sqrt{\cdot 03429}} = 22\cdot 475$ square feet. The quantity of air in D O and T Y being now reduced from 23,864 cubic feet per minute to 21,452, the pressure would be As $23{,}864^2 : 21{,}452^2 :: 2\cdot 4703 : 1\cdot 9962$ lb. The loss of pressure from shaft D to point Y, influencing the current of air in D X P Y, is

In shaft D	·07141
In the splits O S T and O U T	·5347
In D O and T Y	1·9962
	2·60231 lbs. per square foot.

We do not require to know the volume of air this pressure would induce in D X P Y. What we have to ascertain is the portion of this pressure of 2·60231 lbs. requisite for the circulation of 10,726 cubic feet per minute in that airway. The frictional resistances of the airway D X P Y due to this volume are As $22{,}020^2 : 10{,}726^2 :: 3\cdot 005 : \cdot 713$ lb. The resistance of the regulator in this airway, then, must be equal to a pressure of $2\cdot 60231 - \cdot 713 = 1\cdot 88931$ lb., or $\frac{1\cdot 88931}{5\cdot 2} = \cdot 36333$, inch of water-gauge. Its area of opening must be $\frac{\cdot 388 \times 10\cdot 726}{\sqrt{\cdot 36333}} = 6\cdot 9043$ square feet. The loss of pressure from Y to W is As $45{,}884^2 : 32{,}178^2 :: 10\cdot 7644 : 5\cdot 294$ lbs. The total loss of pressure in shaft D and in the airways to W is

2·60231
5·294
7·89631 lbs. per square foot.

This must also be the loss of pressure of the air in shaft C and in the airways C V and V G W.

That for shaft C is	As $46{,}153^2 : 32{,}178^2 :: \cdot 2687 :$	·13061
„ airway C V	„ $46{,}153^2 : 32{,}178^2 :: 11\cdot 5512 :$	·615
		5·74561 lbs. per sq. ft.

The pressure to be expended in V G W then is $7\cdot 89631 - 5\cdot 74561 = 2\cdot 1507$ lbs. Its frictional resistances for a quantity of 10,726 cubic feet per minute are—

As $19{,}598^2 : 10{,}726^2 :: 2\cdot 2182 :: \cdot 7129$ lb.

That expended on the regulator then is 2·1507 − ·7129 = 1·4378 lb. or $\frac{1\cdot4378}{5\cdot2} =$ ·2765 inch of water-gauge. The area of the regulator opening for V G W is $\frac{\cdot388 \times 10\cdot726}{\sqrt{\cdot2765}} = 7\cdot9147$ square feet. The quantity of air passing from W to B has decreased from 45,884 + 19,598 = 65,482 cubic feet to 10,726 × 4 = 42,904 cubic feet per minute, and consequently the pressure necessary for it must be

$$\text{As } 65{,}482^2 : 42{,}904^2 :: \cdot6287 : \cdot2698 \text{ lb.}$$

The total loss of pressure in shaft D and in the airways to B then is 7·89631 + ·2698 = 8·16611 lbs. The split F M L, to pass a quantity of 10,726 cubic feet per minute, will require a pressure of ·53467 lb., thus—

$$\text{As } 11{,}936^2 : 10{,}726^2 :: \cdot6621 : \cdot53467 \text{ lb.}$$

The quantity in split F K L has to be reduced from 14,619 to 10,726 cubic feet per minute. The pressure necessary to overcome the frictional resistances of this airway is

$$\text{As } 14{,}619^2 : 10{,}726^2 :: \cdot6621 : \cdot3564 \text{ lb.}$$

That for the regulator therefore is ·53467 − ·3564 = ·17827 lb. or $\frac{\cdot17827}{5\cdot2} =$ ·03428 inch of water-gauge, and its area of opening is $\frac{\cdot388 \times 10\cdot726}{\sqrt{\cdot03428}} = 22\cdot477$ square feet. It will be observed that the calculations for the split F K L are similar to those for the split O S T, and the area of opening in the regulators in these two splits is the same. The quantity of air in V F and L B having decreased from 26,555 to 21,452 cubic feet per minute, the loss of pressure in these airways is—

$$\text{As } 26{,}555^2 : 21{,}452^2 :: 2\cdot1851 : 1\cdot426 \text{ lb. per square foot.}$$

The loss of pressure from shaft C to the point B due to frictional resistances is as follows—

In shaft C	·13061
„ airway C V	5·615
„ „ V F and L B	1·426
„ splits F K L and F M L	·53467
	7·70628 lbs. per square foot.

It has been shown, however, that the loss of pressure from shaft D through its airways to point B is 8·16611 lbs. and that from shaft C to the same point must be equal, because it is the pressure at this point which determines the relative quantities from the two downcast shafts. To control the quantities of air, then, as desired there must be a regulator placed between L and B or between V and F. This regulator, which must absorb the difference between 8·16611 and 7·70628, amounting to ·45983 lb., is indispensable to an equal distribution of the air: without it greater volumes must be induced to flow through the workings F K L and F M L than through airways O S T and O U T. So far as restriction of volume and absorption of pressure are concerned, the same object would be attained by a regulator fixed between V and F as if the position chosen for it be between L and B. In the former case, however, it would be in the intake airway, and the most suitable position is between L and B. This regulator, to absorb ·45983 lb., or $\frac{\cdot45983}{5\cdot2} = \cdot08843$ inch of water-gauge, must have an area of opening of $\frac{\cdot388 \times 21\cdot452}{\sqrt{\cdot08843}} = 27\cdot99$, or say 28 square feet. The loss of pressure in the airway B A and shaft A is as follows—

		lbs. per square foot.
In the shaft A	As $92{,}037^2 : 64{,}356^2 :: \cdot6104$:	$\cdot29845$
„ airway B A	„ $92{,}037^2 : 64{,}356^2 :: \cdot7479$:	$\cdot36568$
		$\cdot66413$
The loss between shaft C or shaft D and the point B is		$8\cdot16611$
Making a total pressure of		$8\cdot83024$

inclusive of that necessary to overcome the frictional resistances of the shafts themselves.

We have now shown that the effect of placing regulators of the calculated areas in airways where necessary in order to ensure an equal distribution of the air in the airways of the mine has resulted in a reduction of the total quantity of air from 92,037 to 64,356 cubic feet per minute and a decrease in the pressure required amounting to the difference between 16·1703 lbs. and 8·83024 lbs. per square foot. The areas of regulator openings are—

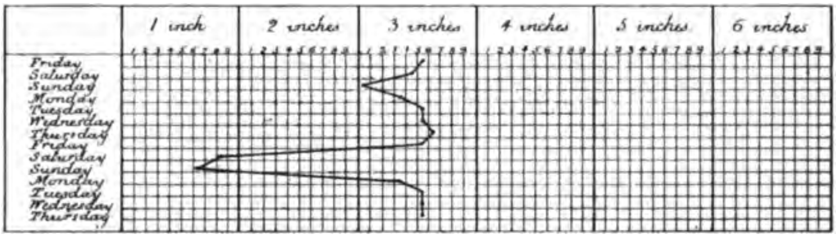

Fig. 422w.—Diagram of Water-gauge Readings at — Colliery, taken at 8 a.m. each day for the fortnight ending 16 January, 1896.

For O S T	22·475	square feet
„ D X P Y	6·9043	„ „
„ V G W	7·9147	„ „
„ F K L	22·477	„ „
„ L B	27·99	„ „

We may raise the total volume of air in circulation by increasing the ventilating pressure without affecting the relative volumes going into the airways, if at the same time we allow the regulators to remain without adjustment of opening. If, for instance, we wish to raise the total volume of air to 92,037 cubic feet per minute, the original volume in the mine, then each airway would receive a sixth part of this, or 15,339 cubic feet per minute. The total pressure must, however, in that case be increased

As $64{,}356^2 : 92{,}037^2 :: 8\cdot83024 : 18\cdot06$ lbs. per square foot.

The effect of the regulators in the mine, then, if the same volume of air is maintained after their insertion, is to increase the ventilating pressure from 16·1703 to 18·06 lbs. If, instead of an equal distribution of the whole volume of air, it is desirable for the districts to have different quantities, the exact amount of regulator opening in each case may be calculated as in the preceding examples.

Unless great care and watchfulness be exercised by the fan engineman or the furnaceman the water-gauge will vary during even a short period of time, owing to a reduced fan speed or a slackening of the furnace. In a fiery mine it is especially important to maintain a constant fan speed and regular water-gauge measurement.

Fig. 422w shows a diagram of water-gauge readings taken for a fortnight. The water-gauge was placed in the ordinary position for furnace ventilation at the separation doors in the mine between the upcast and downcast shafts. The daily readings show that the normal water-gauge is 2·55 inches, but owing to a slackening of the furnaces it has been allowed to fall to 2 inches on a Sunday, which is a non-working day, and even as low as ·6 over a week end when the furnaces were out to facilitate repairs in the upcast shaft. These readings do not, of course, include the loss of pressure absorbed in the shafts themselves by the friction of the air currents.

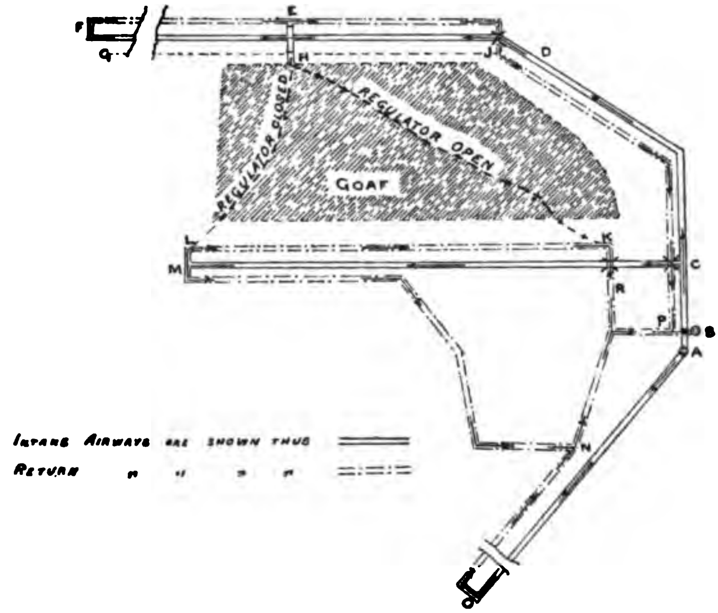

Fig. 422x.—To Illustrate the Effect of Regulators Placed in the Airways of a Mine.

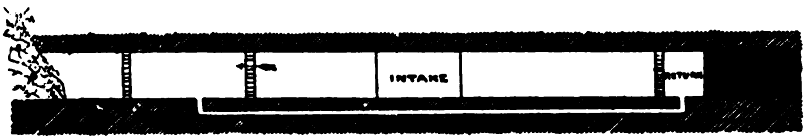

Fig. 422y.—Section.

The effect of placing regulators in certain positions in the mine was pointed out by examples given in a paper by Messrs. John J. Atkinson and John Daglish in the North of England Mining Institute *Transactions*, to which allusion may now well be made.

Example 1 (pp. 126-7, vol. xii.) is a case which occurred on the Hutton seam, a plan of which is shown in Fig. 422x. Here A is the downcast and B the upcast shaft, each being about 260 yards in depth. The arrows show the direction of the air-currents in the mine. The intake airways are drawn in solid lines, the returns are shown by means of dotted lines. The district of workings F is very extensive and reaches a mile beyond where shown on the plan. When the regulator R was open, air from the intake current near E leaked through the stopping at H into the goaf and through it into the return air between L and K, as shown by the dotted line H K.

On partially closing the regulator R, the course of the air through the goaf reversed, passing from the inner part of the return airway near L through the goaf to the stopping at H, beyond which it joined the intake air near E, supplying the district F.

The partial closing of the regulator caused a re-distribution of the air in the mine when the relative pressures favoured the change which took place. Supposing 10 lbs. pressure to be necessary for the splits from point C to point P, and that the loss of pressure from C to H is 3 lbs. and from C to L only 2 lbs. The resistance of the current returning from the district M from the point L on through the regulator to P would be only 8 lbs., and if such a quantity of air passed through the goaf from L to H as to account for 1 lb. of pressure, there would also be a pressure of 8 lbs. to be overcome from L to H and round the district F to the point P. The proper position for the regulator to be placed in is near L before the current reaches the edge of the goaf (see remarks on Fig. 422S). The return airway for the district F would be in a very much better position if placed between the intake and the goaf as shown by the dotted line G J, so that any leakages from the goaf may be direct into the return airway and not into the intake. Even without a regulator at R, however, a fall of stone in the return from the district M may produce all the effects which resulted from the regulator being placed at R. All openings between the intake of one split of air and the return of another split, or between an intake and goaf, should as far as possible be avoided.

To prevent the possibility of this goaf air again pressing towards the intake, two stout stoppings, A, B, Fig. 422Y, were built in the stenton H and a large pipe C C brought under the intake roadway from the return E, ending between the stoppings. A small scale of air was allowed on the stopping B next the intake, the air from which passed through the pipe into the return. By this means the pressure of the current was always from the intake on the stopping B, and effectually prevented any gas from the goaf reaching the intake. The gas was then carried with the scale of air through the pipe into the return airway E.

Example 2 (pp. 128—131).—In this case A and B represent on the plan, Fig. 422Z, the position of the downcast and upcast shafts respectively. Regulators are represented by the letter R with a letter placed before to distinguish them from each other. Regulating doors are represented by the letters R D with a letter placed before to distinguish them from each other. Ordinary doors are represented by the letter D with a preceding letter (other than R) to distinguish them from each other. Air-crossings are represented thus, **X**. The arrows indicate the course of the air-currents, intake airways being drawn with solid lines, while return airways are represented by dotted lines. A more closely dotted line is used to represent the parts where workmen were employed.

Of the air descending shaft A the principal current passed to C, where it divided into two currents, one passing on to D, at which point a small split was taken from it, but the principal part of the current passed on from D to and beyond the point F, where it ventilated a considerable extent of working and then returned to the point G, and thence through H, X, and Y to and up the upcast shaft B. The small split starting from the current just described, at the point D passed through an opening in the regulating door A R D, then through another at B R D and to the point J, where it split; one portion going to the left to H, where it joined the current returning from G, the united volume passing outward to the upcast shaft

B; the other portion passing to the right from J went to K and then, combined with a return current from L, passed through the regulator C R and thence outward to the points X and Y and on to the upcast shaft B.

A few workmen were employed near the point J in working the broken mine near the edge of the large goaf, their coals being brought out by way of B R D, A R D, D, and C to the shaft A.

The principal split of air, starting from the point C, passed to the point M where it was divided into two currents, the principal of which passed to the right to the point N. A small scale of air was taken from the point M, through the regulating door D R D, and another at the point N, through the regulating door F R D, but the chief part of the current passed from N to O, where a small split was taken from it, through the regulator E R D, to ventilate the small goaf, after which this small split passed over the air-crossing T, and so returned through W, X, and Y to the upcast shaft.

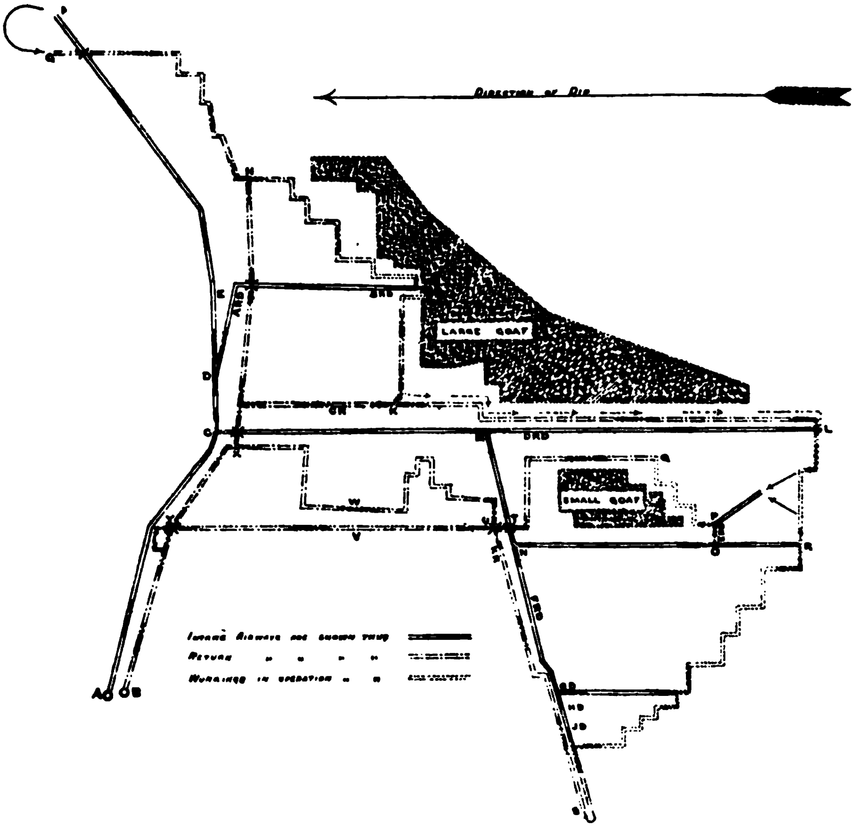

Fig. 422z.—To Illustrate the Effect of Regulators placed in the Airways of a Mine.

The principal current passed forward from O to R, where it was finally split right and left. The split passing to the right from R to the point S was joined by the scales of the doors G D, H D, and J D on its way, and proceeded outward through the regulator K R, and by way of the return V to the point Y, and so on to the upcast shaft. The other split, passing to the left from the point R, flowed to the point L excepting what leaked through the deal stoppings and went to the small goaf: from L the last working place, it passed outward in a return airway by the side of the large goaf to K, where it was joined by the split returning from the point J, and along with it passed outward through the regulator C R and thence through X and Y to the upcast shaft B.

These were the ordinary arrangements for the ventilation of the mine, but if the regulating door A R D was pinned open for a short time, while its companion or doubling regulating door B R D was kept closed, it was observed that a portion of the split of air which passed from the point J towards the point K no longer returned outward by way of the regulator C R, nor did any current whatever continue to pass outward from the point L to the point K; but on the contrary, a part of the air from J, on reaching the point K, passed inward from the point K to L, thus converting the return airway between these points into an intake, the air of which returned by one or both of the other general return airways.

The accidental leaving open a regulating door in such a position as that at A R D, even if its fellow regulating door remain properly closed at the time, must be attended with danger if any explosive gas exists at the edge of the large goaf, because then this reversed current after becoming explosive is carried past the workmen's lights at L.

The strata dipped from the point L to the point C, the former being the most elevated part of the workings.

The cause of the reversal of the direction of the air current between the points L and K on opening the regulating door A R D is the re-distribution of pressures throughout the mine brought about by such opening. More or less it would affect the pressure in each airway, and there would in consequence be a readjustment of the various volumes. The pressure necessary to overcome the frictional and regulator resistances of two or more splits is the same. Under the ordinary condition of the mine, then, the loss of pressure from the splitting point C through J to K is exactly the same as that in the other split from the point C through M and L to K. When, however, the door A R D was opened, the additional pressure previously absorbed by the regulator was sufficient to cause a reversal of the air from K to L and the new point of junction of the currents would be the point of equal pressure.

The authors of the paper go on to say, " The understanding of the preceding remarks may perhaps be facilitated by considering—1st. That two splits of air separating from each other have of necessity the same common tension at the point where they separate, or at the splitting point. 2nd. The same splits, on reuniting, have also, at the point of reunion, a common and equal tension. 3rd. Then, from the 1st and 2nd, it follows that the entire resistance, or lost tension, occurring between the splitting point and the point of reunion, is exactly the same in each split; and that the quantity of air that will pass by way of each split will just be such as to produce this result. 4th. The tension of air is increased by the amount of pressure due to the gravitation of the air in such descents as it traverses, in proportion to the vertical fall of such descents, and again, in proportion to the density of the air over each respective portion of such descents, and the tension of air is, in like manner lessened by all the ascents traversed by currents of air."

Again, " This case shows the great danger that may arise from conducting the return current of air from a split having a short run into the return from one having a long run, particularly at a point after the latter has passed a goaf, without having permanent fixed regulators, in positions where they are not liable to be disarranged.

"Had circumstances allowed of ordinary regulators being placed near the points H and K in the three return airways J H, J K, and L K, all the purposes answered by the regulating doors A R D and B R D (both of which were fixed in working roads), as well as those answered by the regulator C R, would have been attained by their adoption. Had such regulators been adopted, the two regulating doors A R D and B R D, and also the regulator C R, might have been dispensed with.

"And this substitution of three permanent regulators in the waste, for the two regulating doors in the working roads and one permanent regulator in the waste, would have given a more perfect control over the currents of air depending upon them."

Example 3 (pp. 136-7). This, although not bearing upon the use of regulators, may be well given here. It is a case of two large collieries marked A and B, in Fig. 423, the distance between them being about three-quarters of a mile.

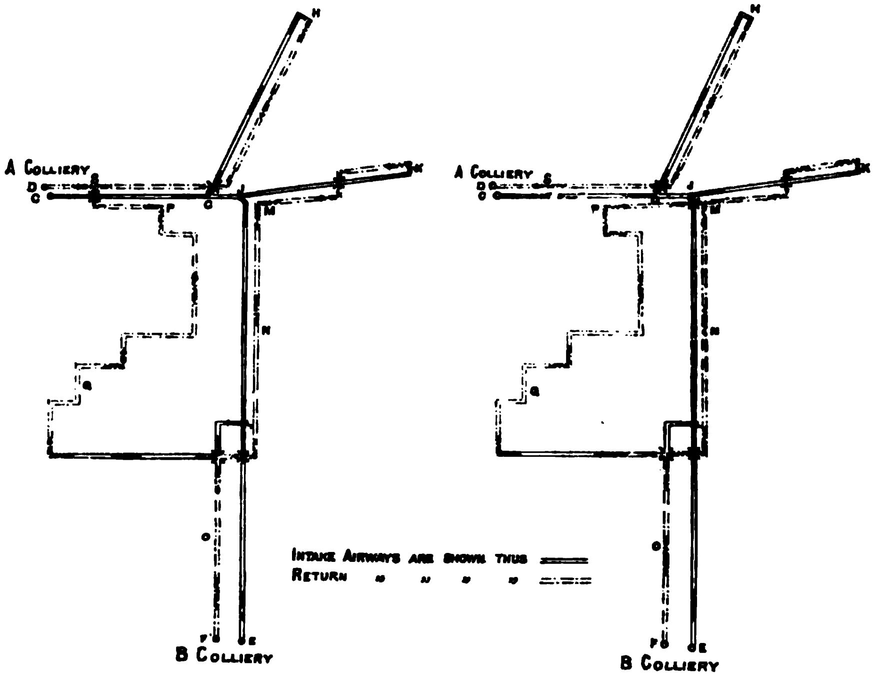

Fig. 423. Fig. 423A

To Illustrate the Effect one Colliery may have on the Ventilation of another if the Workings between the two are Connected.

Each colliery consisted of two shafts, C being the downcast and D the upcast for colliery A, while E was the downcast and F the upcast for colliery B.

The depth of the colliery A to the main coal, which is the seam hereafter referred to, was 216 yards, and that of the colliery B about 280 yards. The shafts were sunk 80 yards lower, and at that level reached the Hutton seam.

Each colliery was, for the most part, ventilated independently of the other, but some air was taken from the colliery B to ventilate two districts of working, K and Q, connected with the colliery A. This air was a division at the shaft E, and formed an intake to the point L, where it was split into two currents. One of these splits or sub-divisions after leaving L passed around the district Q, and joined the return of the colliery A at the point S, and, along with it, went to and up the upcast shaft D of that colliery.

The other split continued a course from L to J, and thence through the district K, after ventilating which it returned by way of N and O to and up the upcast shaft F of the colliery B.

The arrows indicate the direction of the air-currents, air-crossings being shown with a cross. Between the points J and G are a pair of doors separating the air of the two collieries.

The working of the district K proceeded from colliery A, although ventilated by air from the colliery B.

The ordinary conditions of the mine were as stated; but at times, when the ventilation of the colliery A was obstructed or feeble, the current of air ceased to flow from the point L through the district Q, and a current of return air connected with the colliery A passed from the point S through the district Q, and joined the intake air from the colliery B at the point L. The augmented volume then passed through the district K, after which it returned to and ascended the upcast shaft F of the colliery B.

When this reversal of the air occurred, the return air passing from the point S through the district Q to L was most liable to have been charged with firedamp, owing to slackness of ventilation at the colliery A.

Such reversal of the air could only occur when the pressure at the point S, together with the additional pressure operating at the point L, due to the gravitation of the air-column extending from the higher point S, through the district Q to the lower point L, was greater in amount than that of the fresh air coming from the colliery B on reaching the point L, where it was joined by the reversed current.

Any obstruction placed in the shaft D would tend to cause this reversal of air, to guard against the liability of which a new airway, extending from the point P to the point M (Fig. 423A), was driven and an air-stopping was built in the old airway between P and S. After the completion of these works the air passed from the point L, through the district Q, and then on by way of the new airway from P to M, where it joined the return from the district K, and with it passed through N and O to and up the upcast shaft F.

This prevented any intermixture of the air-currents of the two collieries and rendered the ventilation of each separate and distinct from that of the other.

Owing to the air being heated or charged with firedamp in the workings, or perhaps from the effect of both, the return air of most mines is less dense than that of the intake. Usually the leakage to be guarded against is from intake to return, because the pressure in the return airway is less than the pressure in adjacent parts of the intake airway. But it is quite possible for there to be leakage from the return to the intake, although it rarely occurs. Instances of continuous leakage from the return airway of one split into the intake airway of another split have already been given. In a mine having only flat workings continuous leakage from a return to its own intake cannot exist. An intermittent or passing leakage may possibly occur as the result of an obstruction suddenly placed or forming in the return airway at a point outside that of the leakage, or from a sudden pushing forward of the air by such causes as the firing of a shot in the airway inside the point of leakage, or a fall of roof, or the opening or shutting of a door. Continuous leakage from a return airway to its own intake may possibly occur in a mine having dip workings, where the return air is of less density than that of the intake, or in a mine having rise workings, where the return is of greater density than that of the intake. It is caused by natural ventilating pressure. A regulator placed at the outside end of the return airway of such district assists the leakage from return to intake. In shallow mines it is quite possible for the return of a rise district of working to be cooler and of greater density than the intake, and a regulator placed as before stated would assist a leakage from the return to the intake. It will add to the density of the air in the return if CO_2 be given off in the workings. Where the leakage occurs in dip workings the return air must be of less density than that of the intake, the difference producing a natural ventilating pressure which is more than sufficient to overcome the friction of the airways in the district inside the point of leakage. A leakage is meant to be a small quantity of air as compared with the quantity circulating in a district.

NATURAL VENTILATING PRESSURE.

This may arise in the shafts or in the airways of rise and dip workings from a difference in the density of the air in the downcast and upcast shafts, or in the intake and return airways of the mine.

Dealing first with shafts, usually the air in the upcast is of less density than that in the downcast. This produces a certain natural water-gauge. If there were no shaft friction, a higher water-gauge would be registered at the shaft bottom in all such cases than in the fan drift. Ordinarily, however, we should expect to find the fan drift water-gauge higher than that at the pit bottom, because the pit bottom water-gauge does not include the pressure necessary to overcome the friction of the shafts, and this pressure is generally greater than the natural water-gauge. It has at times been observed that the water-gauge at the shaft bottom is greater than the water-gauge on the fan drift, and where this is so it shows that there is an excess of natural ventilating pressure beyond what is needed to overcome the friction of the air in the shafts. It is also quite possible for the water-gauge at the shaft bottom and in the fan drift to be the same, the difference in density then being exactly sufficient to overcome the friction of the air in the shafts. Again, this difference in density may be only sufficient to partly overcome such friction. If the air be of equal density in the two shafts there can be no natural ventilating pressure.

The importance of this shaft natural ventilating pressure is obvious if we take an instance in which it exists, and where it has become desirable to increase the total ventilation of the mine by supplanting one fan by another. If the quantity is to be raised from 100,000 cubic feet to 200,000 per minute through the same airways, the water-gauge must be increased As $1^2 : 2^2$, or 4 times. If the airways of the mine are steep, there is a natural water-gauge due to this which may be quite as important as the natural shaft water-gauge; but we will assume that these airways are practically level, and that, therefore, no natural ventilating pressure can arise from the difference of density in the intakes and returns acting in favour of or against the fan. Let us assume the water-gauge on the fan drift to be 1·5 inches and that at the pit bottom to be 2 inches, and that there is a natural ventilating pressure, calculated from the densities of the downcast and upcast air of 1 inch, half of which is expended upon the friction in the shafts, and the remainder, ·5 inch, is indicated at the pit bottom as an addition to the fan water-gauge. The total water-gauge of the mine, instead of being 1·5 inch as registered in the fan drift, would really be 2·5 inches, and the total water-gauge for the increased quantity desired must be 2·5 × 4 = 10 inches. With the same sources of heat as before, there would be reduction in the natural water-gauge in the shafts inversely in proportion to the increase of quantity, or As 2 : 1 :: 1 : ·5. It would be necessary for the maker of the new fan to guarantee that at a certain stated speed, considered a fair working one, it should give a water-gauge of 10 − ·5 = 9·5 inches, and designed to pass 200,000 cubic feet per minute at that speed. The increased quantity would increase the friction in the shafts to ·5 × 4 = 2 inches of water-gauge. The natural water-gauge produced in the shafts being thus only ·5, the pit bottom water-gauge would no longer exceed the fan drift water-gauge, but would be about 9·5 + ·5 − 2· = 8 inches, when the fan water-gauge is 9·5 inches.

We may now show the error which would arise from taking the fan water-gauge as the total. This would lead to the new fan being specified to give 1·5 × 4 = 6 inches of water-gauge, instead of 9·5. The natural water-gauge of the shafts would then be about ·6 inch, and the total, therefore, would be 6·6 inches. Instead of getting the desired quantity of 200,000 cubic feet per minute, there would really only be in circulation As $\sqrt{2\cdot5} : \sqrt{6\cdot6} :: 100,000 : 162,481$ cubic feet per minute, a quantity which would be very disappointing to those expecting to find 200,000 cubic feet per minute in the mine.

English collieries with thick seams seldom, if ever, require such a high-gauge fan as that just calculated. A Capell fan, at work at Woodthorpe Colliery, Sheffield, has, however, worked at over 9 inches of water-gauge and usually shows 7 inches in regular work. On the Continent of Europe, the difficulty of thin seams is met by high-gauge fans, several of which work at from 10 to 12 inches of water-gauge.

Turning attention now to the natural water-gauge of airways, which cannot occur in a mine having only level airways, let us assume the case of a coal seam having a considerable inclination being worked to shafts in which there is a natural ventilating pressure. If there are three splits of equal sectional area and equal rubbing surface, one being a rise split, the second a level, and the third a dip split, there will be a certain effect on the ventilation due to the natural ventilating pressure of the airways. Let us assume that there is the same difference of density between the return and intake air in the dip and rise districts and that the vertical height of the return air column is the same in each. The pressure for the splits where they lead from the downcast shaft will be the fan's water-gauge plus the natural ventilating pressure of the shafts, but minus the friction in the shafts. As an equal ventilating pressure applies to all the splits, which, let us assume, is 10 lbs., they should all get equal quantities. But the effect of natural ventilation makes each split have a different total ventilating pressure. In the level district there cannot be any natural ventilating pressure produced. The rise and dip splits would have an equal natural ventilating pressure, which let us assume to be 1 lb. per square foot. In the dip district the ventilation is assisted and in the rise district retarded to this extent, that is the three splits have respectively a ventilating pressure of 11, 10, and 9 lbs. per square foot, which is used to overcome the friction in the airways and workings. The dimensions of the splits being equal, the relative quantities passing through them will be in the ratio of the square roots of 11, 10, and 9. If there were a total quantity of air in the mine of 60,000 cubic feet per minute, the proportion going into each split would be as follows:—

$$\begin{aligned}\sqrt{11} &= 3\cdot31662\\ \sqrt{10} &= 3\cdot16228\\ \sqrt{9} &= \underline{3\cdot}\\ &\ 9\cdot4789\end{aligned}$$

		Cubic feet per minute.
In the dip split	As 9·4789 : 3·31662 :: 60,000 :	20,994
„ level „	„ 9·4789 : 3·16228 :: 60,000 :	20,017
„ rise „	„ 9·4789 : 3· :: 60,000 :	18,989
	Total	60,000

Usually dip workings are easier to ventilate than those to the rise because the column of intake air is heavier than that of the return and more easily displaces it. In the rise workings the heavier column of intake air is drawn uphill and the lighter return air downhill. It is quite possible, however, though it rarely happens, that rise workings are more easily ventilated than those to the dip because the return air is denser than the intake. This may arise from their being cooler, and from CO_2 being given off in the mine.

Mr. John J. Atkinson made many experiments to prove the relative proportions of air which would be obtained in short level and long rise or long dip splits having fixed and unvarying sectional areas and rubbing surface, on increasing or diminishing the gross quantity of air in the mine. In these experiments it was found that with a short level split and a long rise one, with the density of the return air less than that of the intakes, on very materially lessening the total quantity of air by contracting the general intake or return at a point passed by the whole volume of air, before reaching the splitting point or after passing the point of reunion, the smaller volume of air flowed altogether through the short level split, and instead of a portion going into the long split, the current of air there became reversed, the air coming outwards in the usual intake airway and joined the volume of air coming in from the shaft at what had previously been the splitting point, and along with it coursed the short split. The supply of air to the long split was through the ordinary return of that split.

Other experiments proved that with a short level and a long dip split, with the density of the return air less than that of the intakes, on reducing the total quantity of air by contracting the main intake or main return at a point passed by the whole volume of air, the current in the short split became reversed, then coming out to join the reduced volume of fresh air at the previous splitting point and along with it passing inwards through the intake of the long dip current and out again through the return airway to the inner end of the shorter route, where it divided, a portion of it coming out of the short split; this portion traversed a continuous circuit, as an eddy, over the total length of both the longer and shorter splits.

The following experiment is given in the paper by Messrs. John J. Atkinson and John Daglish before referred to, and will be found in the "Transactions of the North of England Institute of Mining Engineers," Vol. XII., pp. 133—135. The shaft was about 500 yards in depth to the Hutton seam of coal, in which the experiment was made. The length of the longer or rise split from E, Fig. 423B, through G H to F, the point of reunion of the air-currents was about 3,500 yards, and that of the shorter split E to F only about 20 yards. The vertical height. from the splitting point E to the highest part of the longer route, was 260 feet. By means of a regulator D placed in the shorter split when the main regulator C in the main intake was fully open, two anemometers, one placed in the long and the other in the short split, were made to revolve at the same rate. After this adjustment was made in the regulator D, the total quantity of air was reduced more and more by partial closings in the main regulator C. At each alteration observations proved that the anemometer in the short split revolved at rates which gradually bore a higher and higher ratio to the corresponding rates at which the anemometer in the longer split revolved, showing that each alteration produced an increased share or proportion of the total quantity of air in the shorter split, until at length, when the main regulator was nearly closed, the current of air in the longer split reversed and came outwards from G to E, where it joined the very small quantity of fresh air then passing the main regulator, and with it passed through the short split from E to F, a portion of this reversed current going inwards from F, round H and G, returning to E, thus forming a continuous eddy.

The experiment in dip workings, Fig. 423C, was made at a different colliery. In this case the shafts were 350 yards in depth to the Hutton seam where the experiment was made.

The vertical depth of the long split from the splitting point E to the point G was 93 feet. The length from E through G and H to the point of reunion at F was about 2,500 yards; and the length of the short level split from the point of division at E through the regulator D to the point of junction at F was only about 25 yards.

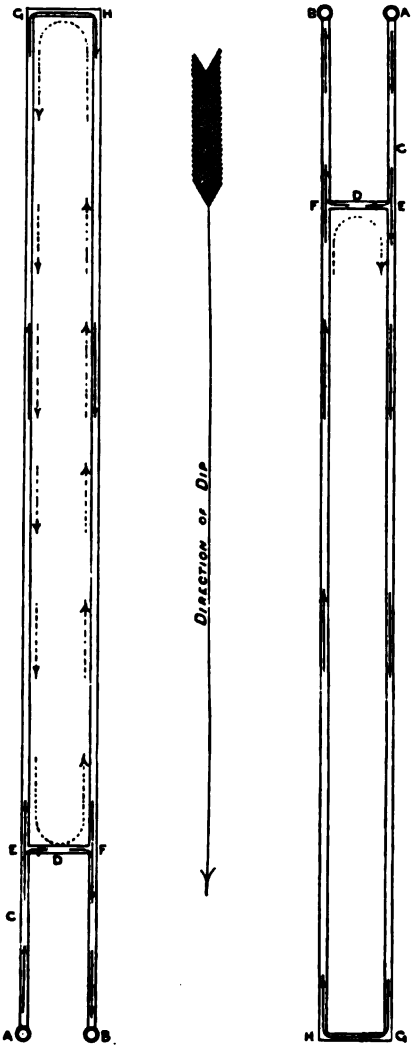

Fig. 423B. Fig. 423C.

Illustrating the Relative Proportions of Air obtained in Short Level and Long Rise, or Long Dip Splits on increasing or diminishing the Gross Quantity.

An anemometer was placed in the long split and another in the short split, and the regulator D was adjusted from time to time until, with the main regulator C in the main intake fully open, the two anemometers revolved at the same speed. In the position to give this result the regulator D was finally fixed.

After this, the main regulator C was partly closed and the amount of opening reduced from time to time, observations being made after each change of the rate of revolution of the anemometers. These proved that the anemometer in the long split revolved at rates which gradually bore a higher and higher ratio to the corresponding rates at which the anemometer in the shorter split revolved, showing that each alteration produced an increased share or proportion of the total quantity of air in the longer split, until at length, when it was sufficiently closed, the current passing into the short split entirely stopped, and a split left the return from the long dip split at the point F, and passed in a contrary direction through the course of the short split to the original splitting point at E, when it met and mixed with the slight current of fresh air then passing the main regulator C and with it passed once more into the long dip split in an eddy, traversing a complete circuit.

This experiment was varied by removing the main regulator C from the intake airway and then placing another in the return airway between the point F and shaft B. This change in the position of the main regulator did not produce any change in the results.

Where the long split current was a dip one the current in the short split reversed, but in the former experiment where the longer split was a rise one this longer split current reversed.

In further explanation of the preceding remarks on this subject the following examples are given:—

Example 1.—Supposing two airways lead from the downcast to the upcast shaft, having equal perimeters and areas, but one is, say, 800 yards long, and the other 400 yards long, the total air circulating being 12,428 cubic feet per minute, the quantity going into each airway will be inversely as the square root of the rubbing surface. As, however, the airways have equal perimeters, it is equally true that their quantities will be inversely as the square root of their lengths. Therefore to find the relative quantity that the 800-yard airway gets as compared with the 400-yard airway's quantity, proceed thus, as $\sqrt{800} : \sqrt{400}$, or as $\sqrt{2} : \sqrt{1}$, or as 1·4142 : 1;—that is, for every cubic foot going into the 800-yard airway, 1·4142 cubic feet will pass into the 400-yard airway. Knowing the total quantity circulating to be 12,428 cubic feet per minute, and the relative quantities to be 1 and 1·4142, the actual quantities will be found thus:—

For the 800 yard airway as 2·4142 : 1 : : 12,428 : 5,148 c. ft.
„ „ 400 „ „ as 2·4142 : 1·4142 : : 12,428 : 7,280 c. ft.

If, instead of these two airways, two of the same area and perimeter, but each 600 yards long between the two shafts could be substituted with the same pressure as before, there would not be quite so large a quantity of air as a total, for the quantity in a 600-yard road would be, as $\sqrt{600} : \sqrt{800}$: : 5,148 or as $\sqrt{3} : 2$: : 5,148 : 5,944 cubic feet for each 600-yard airway = 5,944 × 2 = 11,888 cubic feet, for the two.

From this it is seen that the total quantity of air produced with a given ventilating pressure and rubbing surface would be less in two airways of equal length, than in two of unequal lengths, but as the power producing the ventilation is the quantity passing multiplied by the pressure, in the case of the two airways of equal length the power would be lessened in proportion to the reduced quantity. This can be more strikingly shown by imagining a case of more extreme modification. Take two airways, each of which is 400 feet long and passes 9,000 cubic feet per minute. With a constant pressure if one of them be shortened to 100 feet, its rubbing surface being $\frac{1}{4}$th of what it was, the volume of air then would be 9,000 × $\sqrt{4}$ = 18,000 cubic feet. If the length of the second be increased to 1,600 feet, its rubbing surface being 4 times what it previously was, then the volume of air would be 9,000 × $\sqrt{\frac{1}{4}}$ = 4,500 cubic feet. The total volume of air in the two airways therefore would be 18,000 + 4,500 = 22,500 cubic feet, or an increase of 25 per cent., although the rubbing surface is increased in the proportion of 17 to 8 or more than doubled, and the pressure has remained constant, though the power has been increased in accordance with the increased quantity.

That this is so, assume the airways to be 5 feet by 6 feet, giving a 30-foot area and 22 feet as the perimeter, and testing them by Atkinson's formula $p = \frac{ksv^2}{a}$ we have

for each of the equal airways $\frac{\cdot 26881 \times (400 \times 22) \times \cdot 09}{30} = 7\cdot 0966$ feet of air-column,

for the shortened airway $\frac{\cdot 26881 \times (100 \times 22) \times \cdot 36}{30} = 7\cdot 0966$ feet of air-column,

for the lengthened airway $\frac{\cdot 26881 \times (1{,}600 \times 22) \times \cdot 0225}{30} = 7\cdot 0966$ feet of air-column.

From this it is clear that though the *resistance* increases in proportion to the length of the airway, the quantity or volume of air varies inversely as the square root of the rubbing surface, other things being equal. For this reason, to compare the relative quantities that would flow through a square and a circular airway whose areas and lengths are equal, adopt the same proportion; thus if 10,000 cubic feet passed through a 6-foot square airway, and it were altered to a circular one of the same area and the pressure remained the same, the perimeter of the square airway would be 24, and the perimeter of a circle whose area = 36 is 21·27. Therefore, As $\sqrt{21\cdot 27} : \sqrt{24} :: 10{,}000 : 10{,}622$; that is, 10,622 cubic feet would pass through the circular airway. Again, if the square airway were altered to a circular one of the same area, and instead of the pressure remaining constant, the power had done so, in that case the quantity which would pass through the circular airway would be as $\sqrt[3]{21\cdot 27} : \sqrt[3]{24} :: 10{,}000 : 10{,}410$, as will be better understood when later on the power necessary to circulate the air is dealt with.

Example 2.—Supposing 20,000 cubic feet of air are produced in an airway 10 feet × 6 feet = 60 feet area, how much will be produced in an airway 5 feet × 6 feet = 30 feet area, the pressure being the same in each ?

The perimeter of the 60 feet area airway is 10 + 10 + 6 + 6 = 32 feet, and assuming 1,000 feet length of airway (it is immaterial what length is taken, so that it is the same in both airways) the rubbing surface in the 60 feet area airway is 32 × 1,000 = 32,000 square feet, and the velocity of the air is $\frac{20{,}000}{60} = 333\cdot\dot{3}$ which expressed in thousands of feet per minute = $\dot{3}$. By the formula $p = \frac{ksv^2}{a}$

$$\frac{\cdot 26881 \times 32{,}000 \times (\cdot\dot{3})^2}{60} = 15\cdot 9295 \text{ feet of air column.}$$

The 30-foot airway has a perimeter 6 + 6 + 5 + 5 = 22 and the rubbing surface on 1,000 feet length = 22,000.

To find the velocity in the 30-foot airway $v^2 = \frac{pa}{ks} \therefore v = \sqrt{\frac{pa}{ks}}$ and substituting the values of these letters $v = \sqrt{\frac{15·9295 \times 30}{·26881 \times 22,000}} = \sqrt{\frac{477·885}{5,913·82}} = \sqrt{·0808082} = ·28427$ in thousands of feet ∴ ·28427 × 1,000 = 284·27 the velocity in feet per minute in the 30 feet area airway, and 284·27 × 30 = 8,528 cubic feet per minute in the 30 feet area airway.

This result may be obtained in a simpler way. The one airway is just half the area of the other, and if the perimeters had borne the same ratio it is plain that ½ the quantity in the 60-foot airway would flow through the 30-foot airway, that is, 10,000 cubic feet would have passed through the 30-foot airway if its perimeter had been $\frac{32}{2} = 16$, but as its perimeter is 22, and as (other things being equal) the quantity varies inversely as the square root of the rubbing surface, the quantity could be found thus—As $\sqrt{22} : \sqrt{16} :: 10,000 : 8,528$ cubic feet as before.

Now, assume that the perimeters and rubbing surfaces are the same in the 60 feet area airway and the 30 feet area airway, then with the same pressure proceed to find what quantity would pass into the 30-foot airway, thus:—Whilst the relative areas of the two airways are in the same proportion as their rubbing surfaces, the quantities will also be proportionate to their areas, thus as 60 : 30 :: 20,000 : 10,000. But in that case the large airway's rubbing surface would be 2, and that of the smaller 1; if it is assumed to be the same, then (other things being equal) the quantity is inversely as the square root of the rubbing surface, and as $\sqrt{2} : \sqrt{1} :: 10,000 : 7,071$, which is the quantity in cubic feet per minute the smaller airway would get if 20,000 passed into the larger, and their perimeters and rubbing surfaces were the same.

The advantage of having large airways may be thus strikingly shown, for supposing one airway of 60 feet area, 10 feet by 6 feet, passing 20,000 cubic feet per minute as just given, and it were substituted for two airways each 5 feet by 6 feet, together representing the same area as the large one, the pressure remaining the same, the quantity of air through the two airways would be as $\sqrt{44} : \sqrt{32} :: 20,000 : 17,056$ cubic feet per minute; that is, the two airways would have 17,056 cubic feet per minute as compared with 20,000 cubic feet in the single airway of equal area.

Example 3.—If an airway 7 feet by 8 passes 63,480 cubic feet of air per minute to a point 320 yards from the downcast shaft, after which it is split into 4 airways as follow:—

1. 6 feet by 5 feet being 400 yards long
2. 6 feet by 6 feet ,, 300 ,, ,,
3. 6 feet by 4 feet ,, 280 ,, ,,
4. 5 feet by 4 feet ,, 240 ,, ,,

and it is required to know what current each of the four airways will pass, and what the water-gauge will be.

In considering this question it is plain the four airways get the 63,480 cubic feet amongst them; first proceed to find the relative volumes going into them, so as to afterwards ascertain their actual quantities. The relative and actual quantities are not affected by the length of the main aircourse leading to the point of split. That is, that given the 63,480 cubic feet at the point of split, the relative quantities passing into those splits will be the same whether the air has travelled some distance before reaching the point of split, or if on the other hand the splits lead from the bottom of the downcast shaft, so long as the total quantity is 63,480 cubic feet as stated. All the splits will be subject to the same pressure.

The relative quantities going into airways subject to the same pressure may be found by the formula $\sqrt{\frac{a^3}{s}}$, or $\sqrt{\frac{a}{\sqrt[3]{s}}} = R$, where $R =$ the relative volume

a = the sectional area of the airway, and s the rubbing surface of the airway. If the airways are of the same length then o the value of their perimeters may be substituted for s. Applying this formula to the splits in question,

For No. 1 airway $\sqrt{\frac{30^3}{26,400}}$ = 1·0113 as the relative volume
„ „ 2 „ $\sqrt{\frac{36^3}{21,600}}$ = 1·4697 do. do.
„ „ 3 „ $\sqrt{\frac{24^3}{16,800}}$ = ·907115 do. do.
„ „ 4 „ $\sqrt{\frac{20^3}{12,960}}$ = ·785674 do. do.
Total 4·173789 of the do.

And the actual volumes will be found by proportion thus :—
for No. 1 airway As 4·173789 : 1·0113 :: 63,480 : 15,381·15 cubic ft.
„ „ 2 „ As 4·173789 : 1·4697 :: 63,480 : 22,352·83 „ „
„ „ 3 „ As 4·173789 : ·907115 :: 63,480 : 13,796·53 „ „
„ „ 4 „ As 4·173789 : ·785674 :: 63,480 : 11,949·5 „ „
Total 63,480·01 „ „

The accuracy of this result may be checked by Atkinson's formula $p = \frac{ksv^2}{a}$ when the value of p should work out the same for each of the four airways.

For No. 1 airway $p = \frac{{\cdot}0217 \times 26,400 \times {\cdot}5127^2}{30}$ = 5·0196 lbs. per square foot.
„ 2 „ $p = \frac{{\cdot}0217 \times 21,600 \times {\cdot}6209^2}{36}$ = 5·0196 „ „
„ 3 „ $p = \frac{{\cdot}0217 \times 16,800 \times {\cdot}57485^2}{24}$ = 5·0196 „ „
„ 4 „ $p = \frac{{\cdot}0217 \times 12,960 \times {\cdot}59747^2}{20}$ = 5·0196 „ „

showing that the same pressure satisfies all the splits.

But this pressure of 5·0196 lbs. is only part of the total, there is still to consider that required to pass the whole volume through the large airway. To find the value of p in this airway extending from the pit to the point of splitting $\frac{{\cdot}0217 \times 28,800 \times 1{\cdot}1336^2}{56}$ = 14·34 lbs. per square foot. Then 5·0194 + 14·34 = 19·3594 lbs. per square foot as the total pressure, and the water-gauge, according to the formula, would be $\frac{19{\cdot}3594}{5{\cdot}2}$ = 3·72 inches.

Another method of finding the quantities that would pass along the four splits is, by using Atkinson's formula, $v = \sqrt{\frac{pa}{ks}}$. Assume any value say 1 for p, and having found the relative values of v, the relative quantities in the airways will be these relative velocities multiplied by the areas of the different airways, from which the actual quantities may be obtained.

This method requires more figuring than the formula $R = \sqrt{\frac{a^3}{s}}$, but the student is recommended to try it and compare the result with what is here given.

Example 4.—A mine is ventilated by three splits of air A, B, C;

A being 500 yards long, 5 feet by 6 feet; B is 800 yards long and 5 feet by 4 feet, and C is 700 yards long, 7 feet by 3 feet, all starting and rejoining at the same point. If the quantity in A is 35,000 cubic feet per minute, how much will B and C, which are subject to the same pressure as A, each take?

	Rubbing Surface.	Area.
For A	33,000	30
B	43,200	20
C	42,000	21

By the formula $R = \sqrt{\frac{a^3}{s}}$ the relative quantities are

$$A = \sqrt{\frac{30^3}{33{,}000}} = \cdot 90453, \text{ the relative volume.}$$

$$B = \sqrt{\frac{20^3}{43{,}200}} = \cdot 43033 \quad \text{,,}$$

$$C = \sqrt{\frac{21^3}{42{,}000}} = \cdot 46957 \quad \text{,,}$$

Total 1·80443

A's actual quantity is 35,000 cubic ft. per min.
∴ B's is as ·90453 : ·43033 :: 35,000 : 16,651·2 do.
& C's is as ·90453 : ·46957 :: 35,000 : 18,169·7 do.

& the total quantity flowing is 69,820·9 do.

Example 5.—A mine is ventilated by 3 splits of air, A, B, C; A, taking 2,500 cubic feet per minute, B, 1,500 cubic feet per minute, and C, 2,000 cubic feet per minute, out of a total of 6,000 cubic feet, what will each split take if the total ventilation be increased to 75,000 cubic feet per minute?

The quantities would be in proportion thus—

A's quantity is As 6,000 : 75,000 : : 2,500 : 31,250 cubic ft. per min.
B's ,, ,, 6,000 : 75,000 : : 1,500 : 18,750 do.
C's ,, ,, 6,000 : 75,000 : : 2,000 : 25,000 do.

Total 75,000 do.

Example 6.—If the quantity passing round a mine in one current before splitting is 10,000 cubic feet per minute where the area of the aircourse is 20 feet (5 feet by 4 feet), and the rubbing surface is 24,000 square feet, what quantity will circulate when the current is split into 2, 3, 4, 5, 6, and 10 equal divisions, the pressure remaining the same?

The formula $R = \sqrt{\frac{a^3}{s}}$ to find relative quantities is equally applicable to this case. First of all take the case of one current of 10,000 cubic feet before splitting, to find the quantity that would pass into 2 equal divisions, the pressure remaining the same.

In the first case before splitting the area is 20 feet, and rubbing surface 24,000 square feet.

In the second case, with 2 splits, there would be an area of 40 feet, and a rubbing surface, the same as before, 24,000. Therefore the relative quantities are

As $\sqrt{\frac{20^3}{24{,}000}} : \sqrt{\frac{40^3}{24{,}000}} :: 1$ or as $\sqrt{20^3} : \sqrt{40^3} :: 1 : 2{\cdot}8284$; therefore this simple rule is obtained, that if the rubbing surface and pressure remain unaltered the relative volumes obtained from splitting the air will be in the proportion of $\sqrt{a^3}$. Proceed now to find the relative quantities for 3, 4, 5, 6 and 10 equal divisions thus

For 3 splits as $\sqrt{20^3} : \sqrt{60^3} :: 1 : 5{\cdot}19614$ as the relative volume.
„ 4 „ $\sqrt{20^3} : \sqrt{80^3} :: 1 : 8{\cdot}$ do.
„ 5 „ $\sqrt{20^3} : \sqrt{100^3} :: 1 : 11{\cdot}18033$ do.
„ 6 „ $\sqrt{20^3} : \sqrt{120^3} :: 1 : 14{\cdot}6969$ do.
„ 10 „ $\sqrt{20^3} : \sqrt{200^3} :: 1 : 31{\cdot}622774$ do.

If 10,000 cubic feet be the volume before splitting then

				cub. ft. along
For 2 splits as	1 : 2·8284	:: 10,000 :	28,284, or 14,142	each split.
„ 3 „	1 : 5·19614	:: 10,000 :	51,961·4, or 17,320·5	do.
„ 4 „	1 : 8·	:: 10,000 :	80,000, or 20,000	do.
„ 5 „	1 : 11·18033	:: 10,000 :	111,803·3, or 22,360·6	do.
„ 6 „	1 : 14·6969	:: 10,000 :	146,969, or 24,495	do.
„ 10 „	1 : 31·622774	:: 10,000 :	316,227·74, or 31,622·774	do.

The same result may be arrived at by working out Atkinson's formula.

Example 7.—Supposing in a mine 50,000 cubic feet of air at the shaft are split into 5 distinct currents of equal volume and subject to the same pressure, that is 10,000 cubic feet pass along each of 5 roads of different lengths, No. 1 being 200 yards, No. 2, 400 yards, No. 3, 600 yards, No. 4, 800 yards, and No. 5, 1,000 yards long, and we wish to know the proportional area of the airways.

Let us assume that No. 1 airway is 3 feet × 3 feet. Then as it is 200 yards long we have $R = \sqrt{\frac{9^3}{200 \times 3 \times 12}} = {\cdot}3182$ as the relative volume for No. 1 airway, and as equal volumes are to go into each airway it is evident that ·3182 must be the relative volume for each of the other airways. Now, taking No. 2 airway we are met with the difficulty that the value of o is unknown; but we will assume that it as well as all the other airways are square, then the value of o will be $4\sqrt{a}$ or $\sqrt{16\,a}$. Therefore $\sqrt{\frac{a^3}{\sqrt{16\,a} \times l}} = R$. $\therefore \frac{a^3}{\sqrt{16\,a} \times l} = R^2$; $a^3 = R^2 \times \sqrt{16\,a} \times l$ and $a^3 = \sqrt{16\,a\,R^4\,l^2}$; $a^6 = 16\,a\,R^4\,l^2$; $a^5 = 16\,R^4\,l^2$, and therefore $a = \sqrt[5]{16\,R^4\,l^2}$.

Substituting the value of R and of l in the No. 2 airway we have $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 1{,}200^2} = 11{\cdot}87$ and $\sqrt{11{\cdot}87} = 3{\cdot}446$ feet square as the size for No. 2 airway.

For No. 3, $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 1{,}800^2} = 13{\cdot}97$ and $\sqrt{13{\cdot}97} = 3{\cdot}737$ feet square as the size for No. 3 airway.

For No. 4, $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 2{,}400^2} = 15{\cdot}67$ and $\sqrt{15{\cdot}67} = 3{\cdot}958$ feet square as the size for No. 4 airway.

For No. 5, $a = \sqrt[5]{16 \times {\cdot}3182^4 \times 3{,}000^2} = 17{\cdot}13$ and $\sqrt{17{\cdot}13} = 4{\cdot}14$ feet square as the size for No. 5 airway.

The accuracy of the results obtained may be shown by working out the value of p for the 10,000 cubic feet stated to pass along each airway, thus—

For No. 1 airway $p = \frac{{\cdot}0000000217 \times 7{,}200 \times (10{,}000)^2}{9^3} = 21{\cdot}43$ lbs.

„ „ 2 „ $p = \frac{{\cdot}0000000217 \times 16{,}540{\cdot}8 \times (10{,}000)^2}{11{\cdot}87^3} = 21{\cdot}43$ lbs.

For No. 3 airway $p = \dfrac{{\cdot}0000000217 \times 26{,}906{\cdot}4 \times (10{,}000)^2}{13{\cdot}97^3} = 21{\cdot}43$ lbs.

" " 4 " $p = \dfrac{{\cdot}0000000217 \times 37{,}996{\cdot}8 \times (10{,}000)^2}{15{\cdot}67^3} = 21{\cdot}43$ lbs.

" " 5 " $p = \dfrac{{\cdot}0000000217 \times 49{,}680 \times (10{,}000)^2}{17{\cdot}13^3} = 21{\cdot}43$ lbs.

The undesirability of small airways is shown by the great pressure required. If we take the No. 1 airway as being 6 feet × 6 feet, instead of 3 feet × 3 feet, we may ascertain the areas of the others by proportion, thus—

No. 1 airway 36 feet = 6 feet × 6
" 2 " As 9 : 36 :: 11·87 : 47·48
or " 1 : 4 :: 11·87 : 47·48 = 6·89 feet × 6·89
" 3 " " 1 : 4 :: 13·97 : 55·88 = 7·475 feet × 7·475.
" 4 " " 1 : 4 :: 15·67 : 62·68 = 7·917 feet × 7·917.
" 5 " " 1 : 4 :: 17·13 : 68·52 = 8·278 feet × 8·278.

Taking No. 1 airway, the proportional value of p for the 9 feet and 36 feet area airways will be

$$\text{As } \frac{{\cdot}0000000217 \times 7{,}200 \times \left(\frac{10{,}000}{9}\right)^2}{9} : \frac{{\cdot}0000000217 \times 14{,}400 \times \left(\frac{10{,}000}{36}\right)^2}{36}$$

$$= \frac{1 \times 1 \times 4^2}{1} : \frac{1 \times 2 \times 1^2}{4}$$

$$= \frac{16}{1} : \frac{2}{4} \quad \text{or As } 32 : 1$$

That is, the pressure for the 36 feet area airway is $\dfrac{21{\cdot}43}{32} = {\cdot}67$ lb., and this pressure holds good for the other airways, as may be proved by calculation.

Example 8.—Supposing 9,000 cubic feet per minute circulate through a regulator 30 inches × 20, and it is desired to find how much will circulate if made 30 inches by 30, the total pressure for the whole mine remaining unaltered.

The quantity of air which would pass a regulator after the area of opening is altered must depend upon the ventilation arrangements as a whole, and cannot be calculated apart from their full consideration. Usually, the amount of opening given to a regulator is fixed experimentally. We will take two cases of altering the opening from 30 inches × 20 to 30 inches × 30 in a regulator, to show that the quantity passing must vary under different conditions of ventilation.

First take the case of a downcast and an upcast shaft, each 12 feet in diameter and 600 yards in depth, in which two splits of air ventilate the workings, the one being 1,000 yards long and the other 500, each being distinct and separate from the downcast to the upcast shaft and 6 feet high by 6 feet wide.

The regulator opening of 30 × 20 inches is equal to 4·1666 square feet. By the formula for the equivalent orifice, $a = \dfrac{{\cdot}388\,q}{\sqrt{h}}$. Therefore $h = {\cdot}1505\left(\dfrac{q}{a}\right)^2$

Substituting for this, we have $h = {\cdot}1505\left(\dfrac{9}{4{\cdot}1666}\right)^2 = {\cdot}7022$ nch of water-gauge or $\cdot7022 \times 5{\cdot}2 = 3{\cdot}6513$ lbs. per square foot, absorbed by the regulator. The pressure necessary to overcome friction in an airway 500 yards and 6 feet by 6 feet is $p = \dfrac{({\cdot}0000000217) \times (500 \times 3 \times 24) \times (9{,}000)^2}{36^3} = 1{\cdot}3562$ lb., and the total loss of pressure in the airway is $3{\cdot}6513 + 1{\cdot}3562 = 5{\cdot}0075$ lbs. This pressure being common to the 1,000-yard airway, the velocity of air in it is $\sqrt{\dfrac{5{\cdot}0075 \times 36}{({\cdot}0000000217) \times (1{,}000 \times 3 \times 24)}} = 339{\cdot}68$ feet per minute, and the quantity is $339{\cdot}68 \times 36 = 12{,}228$ cubic feet per minute. The total quantity of air in the mine is $9{,}000 + 12{,}228 = 21{,}228$ cubic feet per minute. The loss of pressure in each shaft from the friction of this quantity of air is

$$\frac{({\cdot}00000001) \times (37{\cdot}699 \times 600 \times 3) \times (21{,}228)^2}{113{\cdot}097^3} = {\cdot}2114 \text{ lb.}$$

The total loss of pressure, then, is

In the downcast shaft .	·2114
„ upcast „ . .	·2114
„ split airways . .	5·0075
	5·4303 lbs. per square foot.

A greater quantity of air will pass through the short airway on further opening the regulator. Let us assume that when the opening is 30 × 30 inches 10,000 cubic feet of air per minute pass in the 500-yard airway.

The pressure absorbed by a regulator 2½ × 2½ feet, or 6·25 square feet in area, passing 10,000 cubic feet of air per minute is $\cdot 1505 \times \left(\dfrac{10}{6{\cdot}25}\right)^2 = {\cdot}3853$ inch of water-gauge, or $\cdot 3853 \times 5{\cdot}2 = 2{\cdot}0034$ lbs. per square foot. The pressure necessary to overcome friction in the 500-yard airway is as $9{,}000^2 : 10{,}000^2 :: 1{\cdot}3562 : 1{\cdot}6743$ lb. The pressure common to the splits in that case would be $2{\cdot}0034 + 1{\cdot}6743 = 3{\cdot}6777$ lbs. per square foot. The quantity in the long airway would be as $\sqrt{5{\cdot}0075} : \sqrt{3{\cdot}6777} :: 12{,}228 : 10{,}479$ cubic feet. The whole volume of air in the mine then is $10{,}000 + 10{,}479 = 20{,}479$ cubic feet per minute. The loss of pressure in each shaft from the passage of this air is as $21{,}228^2 : 20{,}479^2 :: {\cdot}2114 : {\cdot}1967$ lb.

The total loss of pressure would be

In the downcast shaft .	·1967
„ upcast „ . . .	·1967
„ split airways . .	3·6777
	4·0711 lbs. per square foot.

On the assumption that the total pressure is maintained at its original amount of 5·4303 lbs., however, we shall have a total volume of air as follows—

As $\sqrt{4{\cdot}0711} : \sqrt{5{\cdot}4303} :: 20{,}479 : 23{,}652$ cubic feet per minute.

The effect, then, of increasing the regulator opening from 30 inches by 20 to 30 inches by 30, with a constant total ventilating pressure and under the conditions of the mine is to increase the total volume of air from 21,228 cubic feet per minute to 23,652. The volume passing into the 500-yard airway under the altered circumstances is—

As 20,479 : 23,652 :: 10,000 : 11,549 cubic ft. per min.
And in the long split, As 20,479 : 23,652 :: 10,479 : 12,103 „ „ „ „
23,652 „ „ „ „

The increase due to the greater opening in the regulator, then, under the given conditions is the difference between 11,549 and 9,000, or 2,549 cubic feet of air per minute.

Secondly, let us take a case where the shafts are exactly of the same size and of the same depth as in the first case, and where the airways are all 6 feet by 6 in size, each leading from the downcast to the upcast shaft without communicating with each other. We will assume there are 6 such splits, 5 being of equal length, viz., 1,000 yards and one of only 500 yards. In the last-named the regulator is fixed.

When the opening is 30 inches by 20 and 9,000 cubic feet of air per minute pass through it, the pressure absorbed by the regulator is 3·6513 lbs. per square foot and that to overcome friction in the airway is 1·3562 lb., making a total loss of pressure of 5·0075 lbs. as in the first case. This pressure being common to the splits, there would pass into each 1,000-yard airway the same quantity as calculated for the long airway in the first case, viz., 12,228 cubic feet per minute. The total volume of air in the shafts, then, would be (12,228 × 5) + 9,000 = 70,140 cubic feet per minute. The loss of pressure in each shaft from the friction of this quantity is as $21{,}228^2 : 70{,}140^2 ::$ ·2114 : 2·3077 lbs.

The total loss of pressure is—

In the downcast shaft .	2·3077 lbs. per square foot.
„ upcast „ . .	2·3077 „ „ „
„ split airways . .	5·0075 „ „ „
	9·6229 „ „ „

Assuming a quantity of 10,000 cubic feet of air per minute to pass through the regulator when the opening is enlarged to 30 inches by 30, the pressure common to the splits would become 3·6777 lbs. per square foot as in the first case, and the quantity in each 1,000-yard airway would be 10,479 cubic feet per minute. The whole volume of air in the mine, then, is (10,479 × 5) + 10,000 = 62,395 cubic feet per minute. The loss of pressure in each shaft due to the friction of this air is—

As $21{,}228^2 : 62{,}395^2 ::$ ·2114 : 1·8262 lb.

The total loss of pressure is—

In the downcast shaft .	1·8262
„ upcast „ . .	1·8262
„ split airways . .	3·6777
	7·3301 lbs. per square foot.

But if the total pressure is to remain as at first, viz., 9·6229 lbs., the total quantity of air in the mine would be as $\sqrt{7·3301} : \sqrt{9·6229} ::$ 62,395 : 71,490 cubic feet per minute. The effect of opening the regulator from 30 inches by 20 to 30 inches by 30, with a constant total ventilating pressure, has thus increased the whole volume of air in the mine from 70,140 to 71,490 cubic feet per minute. The quantity passing into the 500-yard airway under the altered circumstances is—

As 62,395 : 71,490 :: 10,000 : 11,458 cubic ft. per min.
And in each long split As 62,395 : 71,490 :: 10,479 : 12,007 „ „ „ „

The increase due to the greater opening in the regulator under the conditions of this mine, then, is the difference between 11,458 and 9,000, or 2,458 cubic feet of air per minute.

This increase differs from the 2,549 cubic feet shown to be the result of increasing the regulator to the same extent in the first case where different conditions of the mine prevailed. The quantity of air, therefore, which will pass through a regulator can only be calculated by taking into consideration the whole of the ventilating arrangements in that mine in which it is placed.

Example 9.—If 100,000 cubic feet of air pass per minute with a 2-inch water-gauge, and it is required to know the units of work producing the ventilation, 100,000 × 5·2 × 2 = 1,040,000 units of work.

Example 10.—If it is desired to know the units of work required to overcome the friction in a circular aircourse 6 feet in diameter and 500 yards long, with 4,807 cubic feet passing per minute, $p = \frac{ksv^2}{a} = \frac{·0000000217 \times 28{,}274·4 \times 170·013^2}{28·2744} =$ ·6272. Therefore, 4,807 × ·6272 = 3,015 units of work.

Example 11.—If 20,000 cubic feet of air pass, the water-gauge being ·9 inch, and it is wanted to know what quantity will pass when the water-gauge is 2·6 inches, remember that the quantity varies as the square root of the pressures. Therefore, as $\sqrt{\cdot 9} : \sqrt{2\cdot 6} :: 20{,}000 : 33{,}993$ cubic feet.

Example 12.—If it is desired to double the quantity of the air, the ventilating pressure must be increased $2^2 = 4$ times.

Example 13.—If with a water-gauge of ·1 inch 20,000 cubic feet of air per minute are obtained, and afterwards the quantity is increased to 60,000 cubic feet per minute, and it is desired to know the height of the water-gauge, remember that the square of the quantity of the air is proportional to the water-gauge, and as there are 3 times the quantity passing, the gauge will be $3^2 = 9$ times higher, and $9 \times \cdot 1 = \cdot 9$ inch as the height of water-gauge.

Example 14.—Supposing two airways (subject to the same pressure) of the same area, passing a total quantity of 100,000 cubic feet of air per minute, the resistances in the airways being in the proportion of 5 to 1, and it is desired to know the quantity going along each.

By the formula $R = \sqrt{\frac{a^3}{s}}$ for the one airway $R = \sqrt{\frac{a^3}{s}} = 1$, supposing a is 1 and s also 1; then for the second airway $a = 1$ and $s = 5$ $\therefore R = \sqrt{\frac{1^3}{5}} = \cdot 447214$. The sum of the two relative quantities is $1 + \cdot 447214 = 1\cdot 447214$, and the actual quantities going along each airway the total of which is 100,000, will be found by proportion, thus:—

As 1·447214 : ·447214 :: 100,000 : 30,901 cubic feet;
and as 1·447214 : 1· :: 100,000 : 69,098 „ „
Total 99,999 „ „

per minute, practically 100,000, the difference being due to error in neglecting decimals.

Example 15.—The junction of the two intakes at a colliery having 3 shafts (2 downcast and one upcast) is 300 yards from the downcast in one case and 600 yards in the other, and the distance from the junction to the upcast is 200 yards. The area of all the airways is the same, being 6 feet by 6, and all are subject to the same pressure. What total quantity of air will pass, and what quantity in each airway, the water-gauge at the bottom of the upcast being 1·5 inch?

Assume that v = the velocity of the air per minute through the longer intake,
and $v_{,}$ = the velocity do. do. do. the shorter do.

$$\text{then } p = \frac{\cdot 0000000217 \times 600 \times 3 \times 24 \times v^2}{36} \qquad (1)$$

$$\text{and also } p = \frac{\cdot 0000000217 \times 300 \times 3 \times 24 \times v_{,}^2}{36} \qquad (2)$$

Therefore the right hand side of (1) = the right hand side of (2), and by cancelling common factors $2v^2 = v_{,}^2$, $\therefore 1\cdot 414213v = v_{,}$. As the areas of the airways are equal, the velocity of the air in that part between the junction and the upcast will be equal to the sum of the velocities in the two intakes, that is, it will be $v + 1\cdot 414213v = 2\cdot 414213v$. As the water-gauge at the bottom of the upcast shaft is 1·5 inch, then $5\cdot 2 \times 1\cdot 5 = 7\cdot 8$ lbs. per square foot as the total pressure producing ventilation. If p represents that part of the pressure which is expended in overcoming the resistance met with by the air in passing through that part of the mine between the junction and the upcast,

Then $p = \frac{{\cdot}0000000217 \times 200 \times 3 \times 24 \times (2{\cdot}414213v)^2}{36}$.

The pressure for each of the intakes from the downcast to the point of junction will be $7{\cdot}8 - p$. Applying this to the longest intake, thus :—

$$7{\cdot}8 - p = \frac{{\cdot}0000000217 \times 600 \times 3 \times 24 \times v^2}{36}$$

and therefore $\frac{p}{7{\cdot}8 - p} = \dfrac{\dfrac{{\cdot}0000000217 \times 200 \times 3 \times 24 \times (2{\cdot}414213\ v)^2}{36}}{\dfrac{{\cdot}0000000217 \times 600 \times 3 \times 24 \times v^2}{36}}$

Cancel the common factors thus :—

$$\frac{p}{7{\cdot}8 - p} = \frac{2{\cdot}414213^2}{3} = 1{\cdot}9428$$

$$p = 15{\cdot}15387 - 1{\cdot}9428p$$

$$2{\cdot}9428p = 15{\cdot}15387,$$

$p = 5{\cdot}1495$, and the pressure in each of the two intakes is $7{\cdot}8 - 5{\cdot}1495 = 2{\cdot}6505$.

To find the velocity of air between the junction and the upcast, $v = \sqrt{\frac{5{\cdot}1495 \times 36}{{\cdot}0000000217 \times 200 \times 3 \times 24}} = 770{\cdot}23$ as the velocity in feet per minute, and therefore the total volume passing is $770{\cdot}23 \times 36 = 27{,}728{\cdot}28$ cubic feet.

To find the velocity in the longer intake, $v = \sqrt{\frac{2{\cdot}6505 \times 36}{{\cdot}0000000217 \times 600 \times 3 \times 24}}$ $= 319{\cdot}04$, and therefore the volume passing through the longer intake is $319{\cdot}04 \times 36 = 11{,}485{\cdot}44$ cubic feet.

Similarly for the shorter airway, $v = \sqrt{\frac{2{\cdot}6505 \times 36}{{\cdot}0000000217 \times 300 \times 3 \times 24}} = 451{\cdot}2$, and therefore the volume passing through the shorter intake is $451{\cdot}2 \times 36 = 16{,}243{\cdot}2$ cubic feet. The accuracy of these quantities is shown by adding together the intake volumes to the point of junction, and observing that they agree with the total thus, $11{,}485{\cdot}44 + 16{,}243{\cdot}2 = 27{,}728{\cdot}64$; and further, by multiplying the quantity 11,485·44 in the long airway by 1·414213, when, as a result, 16,243 cubic feet per minute is obtained as the volume for the shorter intake airway.

Example 16.—An aircourse is 8,000 feet long, 5 feet by 5 feet, and the quantity of air circulating is 6,250 cubic feet per minute in a district where 90 miners work; it is desired to convey 250 cubic feet per minute to each miner, and in order to do so, state what size the aircourse must be made, the ventilating pressure remaining the same.

Here there are 6,250 cubic feet per minute in an airway 8,000 feet long, its section being 5 feet by 5, and it is necessary to enlarge the area of the same length of airway to pass $250 \times 90 = 22{,}500$ cubic feet per minute with the same pressure.

In this case the relative quantities passing in two airways are given, the dimensions of one being stated, and the other has to be ascertained. By the formula $R = \sqrt{\frac{a^3}{s}}$, and using the value of o instead of s the relative volume going into the 5-foot airway is $\sqrt{\frac{25^3}{20}} = 27{\cdot}95$, and therefore if $a =$ the area of the required airway, and o its perimeter then $\sqrt{\frac{a^3}{o}} = 27{\cdot}95 \times \frac{22{,}500}{6{,}250} = 27{\cdot}95 \times 3{\cdot}6 = 100{\cdot}62$ as the relative volume for the altered airway.

The difficulty now is that the value of o is not known, but assume the airway to be square, then the value of o will be $4\sqrt{a}$ or $\sqrt{16\,a}$. Therefore $\sqrt{\frac{a^3}{\sqrt{16\,a}}} = \mathrm{R}$, where R represents the relative quantity.

$\therefore \frac{a^3}{\sqrt{16\,a}} = \mathrm{R}^2$; $a^3 = \mathrm{R}^2 \times \sqrt{16\,a}$ and $a^3 = \sqrt{16\,a\,\mathrm{R}^4}$; $a^6 = 16\,a\,\mathrm{R}^4$; $a^5 = 16\ \mathrm{R}^4$, and therefore $a = \sqrt[5]{16\ \mathrm{R}^4}$.

Substituting the value of R, which in this case for the altered airway was found to be $100{\cdot}62$, then $a = \sqrt[5]{16 \times 100{\cdot}62^4}$, and therefore (by using logarithms) $a = 69{\cdot}659$. $\sqrt{69{\cdot}659} = 8{\cdot}346$, or 8 feet 4·152 inches by 8 feet 4·152 inches as the size of the square airway necessary.

The result shows that it would be much easier to split the air than to attempt to carry it in one aircourse of the unusual size of 8 feet 4 inches square.

If instead of a square airway it had been desirable in the above example to have a circular one, then if a represented its area, its perimeter would be $3{\cdot}545\sqrt{a}$. In that case to find the area required proceed thus $\sqrt{\frac{a^3}{3{\cdot}545\ \sqrt{a}}} = \mathrm{R}$; $\frac{a^3}{\sqrt{12{\cdot}567\,a}} = \mathrm{R}^2$; $a^3 = \mathrm{R}^2 \times \sqrt{12{\cdot}567\,a}$; $a^3 = \sqrt{12{\cdot}567\,a\,\mathrm{R}^4}$; $a^6 = 12{\cdot}567\,a\,\mathrm{R}^4$; $a^5 = 12{\cdot}567\ \mathrm{R}^4$; $a = \sqrt[5]{12{\cdot}567\ \mathrm{R}^4}$; and substituting the value of R as before, $a = \sqrt[5]{12{\cdot}567 \times 100{\cdot}62^4}$, and therefore $a = 66{\cdot}375$, and the diameter of the airway would be $\sqrt{\frac{66{\cdot}375}{{\cdot}7854}} = 9{\cdot}193$ feet.

Again, if it had been desirable to have a rectangular road instead of a square or circular one, either the height or the width should be stated, probably the height of the seam would be known and given.

In the example being considered, if the height of the seam is 6 feet, proceed to find the width and area of the rectangular road thus :—

If $x =$ its width in feet. Then $6\,x =$ its area and $12 + 2\,x =$ its perimeter. Then $\sqrt{\frac{(6\,x)^3}{12 + 2\,x}} = \mathrm{R}$; and $\frac{(6\,x)^3}{12 + 2\,x} = \mathrm{R}^2$; $(6\,x)^3 = 12\,\mathrm{R}^2 + 2\,\mathrm{R}^2 x$,

$\therefore x^3 = \frac{12\,\mathrm{R}^2 + 2\,\mathrm{R}^2 x}{216}$ and substituting the known value of R $x^3 = \frac{12\,(100{\cdot}62)^2 + 2\,(100{\cdot}62^2)\,x}{216}$; $x^3 = \frac{121{,}500 + 20{,}250\,x}{216} = 93{\cdot}75\,x + 562{\cdot}5$; $x^3 - 93{\cdot}75\,x - 562{\cdot}5 = 0$.

Here is a cubic equation, and therefore proceed to find the value of x by the following trigonometrical formula,

$$x^3 - px - \mathrm{Q} = 0$$

$$\sin.\ a = \frac{p}{3\,\mathrm{Q}}\,2\sqrt{\frac{p}{3}}$$

$$\tan.\ \mathrm{B} = \sqrt[3]{\tan.\ \tfrac{1}{2}\,a}.$$

$$x = \frac{2\sqrt{\frac{p}{3}}}{\sin.\ 2\,\mathrm{B}.}$$

Substituting the values of p and Q in the formula

$$\sin. a = \frac{93.75}{1,687.5} \times 2 \times \sqrt{\frac{93.75}{3}}$$

$$a = 38^\circ 24'$$

$$\text{and } \tfrac{1}{2}\, a = 19^\circ 12'$$

$$\tan. B = \sqrt[3]{\text{Tan. } 19^\circ 12'}$$

$$\text{Log. tan. } 19^\circ 12' = 9.541875$$

$$\text{Less } 10$$

$$\text{which divided by 3 to cube root })\underline{\bar{1}.541875}$$

$$\bar{1}.847292$$

$$\text{add } \underline{10 \qquad\qquad}$$

$$9.847292$$

$$= \text{Log. tan. of } 35^\circ 8'$$

$$2\,B = 70^\circ 16'$$

$$x = \frac{2\sqrt{\frac{93.75}{3}}}{\sin. 70^\circ 16'}$$

$$x = 11.88$$

11·88 feet then is the width of the rectangular airway, and $6 \times 11.88 = 71.28$ square feet of sectional as compared with 69·659, the area of the square airway and 66·375 the area of the circular airway.

If the water-gauge in the above instance is required, then $p = \dfrac{.0217 \times 160{,}000 \times .25^2}{25}$ $= 8.68$ lbs. per square foot pressure and $\dfrac{8.68}{5.2} = 1.67$ inch as the water-gauge.

Observing now somewhat closely the examples just worked out, it is seen that to increase the quantity from 6,250 cubic feet per minute to 22,500 cubic feet per minute the airway must be increased from 5 feet square to one of 8·346 feet square, the pressure and length of airway remaining constant.

It may now be shown that the simplest way of finding the desired side of square for the altered airway is by taking the $\frac{2}{5}$th root of the ratio of the quantities of air and multiplying this result by the side of the known airway, thus in the present example $\left(\dfrac{22{,}500}{6{,}250}\right)^{\frac{2}{5}} \times 5 = 1.66925 \times 5 = 8.34625$ feet as the side of the square forming the altered airway, being the same result as already proved.

For circular airways this rule holds good also, applied to the diameters of the airways in the same way as applied to the side of a square airway. That this is true may be seen by another example.

Example 17.—If 8,000 cubic feet of air pass through a circular airway whose diameter is 5 feet, the length being 10,000 feet, and the quantity has to be increased to 20,000 cubic feet per minute by enlarging the airway in circular form, the pressure remaining the same, proceed thus :—

The ratio of the altered quantity to the old quantity would be $\dfrac{20{,}000}{8{,}000} = 2.5$

and $\left(2{\cdot}5\right)^{\frac{2}{5}} = 1{\cdot}4427$; $1{\cdot}4427 \times 5 = 7{\cdot}2135$, which is the diameter in feet that the altered airway should be. That this is so, can be shown by proving the value of p in each case, and it will be found to agree, thus :—

Before enlarging $p = \dfrac{{\cdot}0000000217 \times 157{,}080 \times 407{\cdot}43^2}{19{\cdot}635} = 28{\cdot}81$ lbs. per sq. ft.

After enlarging $p = \dfrac{{\cdot}0000000217 \times 7{\cdot}2135 \times 3{\cdot}14159 \times 10{,}000 \times \left(\dfrac{20{,}000}{7{\cdot}2135^2 \times {\cdot}7854}\right)^2}{7{\cdot}2135^2 \times {\cdot}7854}$
$= 28{\cdot}81$ lbs.

Referring to *Example* 16, the rule there given also holds good for rectangular airways applied to the height and to the width of the airways, that is, the $\frac{2}{5}$th root of the ratio of increased or decreased quantity must be multiplied by the height of the original airway to give the height of the altered airway, and by the width of the original airway to give the width of the altered airway.

That this is true, also may be proved by means of the following example.

Example 18.—An airway 5,000 feet long, 5 feet high, and 7 feet wide, passes 10,000 cubic feet of air per minute, and it is desired to enlarge the airway in the same rectangular form to pass 30,000 cubic feet of air per minute, the pressure remaining the same.

Here the ratio of increased quantity to the original would be $\dfrac{30{,}000}{10{,}000} = 3$ and $\left(3\right)^{\frac{2}{5}} = 1{\cdot}55184$. The height of the altered airway, therefore, must be $1{\cdot}55184 \times 5 = 7{\cdot}7592$ feet, and the width $1{\cdot}55184 \times 7 = 10{\cdot}86288$, so that the altered airway must be 10·86288 feet wide and 7·7592 feet high. Again prove the accuracy of this by testing the value of p in each case, thus :—

Before enlarging $p = \dfrac{{\cdot}0000000217 \times 120{,}000 \times 285{\cdot}7^2}{35} = 6{\cdot}073$ lbs. per sq. ft.

After enlarging $p = \dfrac{{\cdot}0000000217 \times 186{,}220{\cdot}8 \times 355{\cdot}925^2}{84{\cdot}287} = 6{\cdot}073$ do.

Example 19.—Suppose an airway as in the last example 5,000 feet long, 5 feet high, and 7 feet wide, passing 10,000 cubic feet per minute, and it is desired to enlarge the airway in a circular form to pass 30,000 cubic feet of air per minute, the pressure remaining the same.

Using the value of o for the 5 feet by 7 feet airway $R = \sqrt{\dfrac{35^3}{24}} = 42{\cdot}2665$, and if a = the area of the required airway and o its perimeter, then $\sqrt{\dfrac{a^3}{o}} = 42{\cdot}2665 \times \dfrac{30{,}000}{10{,}000} = 42{\cdot}2665 \times 3 = 126{\cdot}7995$ as the relative volume for the altered airway. Therefore, as in *Example* 16, $a = \sqrt[5]{12{\cdot}567\ R^4}$ and substituting the value of R for the altered airway $a = \sqrt[5]{12{\cdot}567 \times 126{\cdot}7995^4} = 79{\cdot}862$ feet; its diameter therefore is $\sqrt{\dfrac{79{\cdot}862}{{\cdot}7854}} = 10{\cdot}0838$ feet, and its perimeter $10{\cdot}0838 \times 3{\cdot}14159 = 31{\cdot}679$ feet. If the value of p be now worked out for this airway according to the formula $p = \dfrac{ksv^2}{a}$, it will be found as in the last example for a rectangular road to be 6·073 lbs. per square foot.

Example 20.—Supposing a 5-foot square airway with the conditions given in *Example* 16, of 6,250 cubic feet of air passing along it, and this quantity is to be increased to 22,500 cubic feet per minute with the same pressure along the same length of road, but finding it not practicable to use a road of 8·34625 feet square, it is decided to make an additional road or roads of the height of the seam, viz., 5 feet.

The old airway would continue passing its quantity of 6,250 cubic feet per minute, so that the additional road or roads must pass 22,500 — 6,250 = 16,250 cubic feet per minute.

Here it is evident that it would be better to use two additional roads to provide for the 16,250 cubic feet as the height is stated to be 5 feet. Adopting this plan of passing $\left(\frac{16{,}250}{2}\right) = 8{,}125$ cubic feet per minute along each of the extra roads, proceed to find the value of a side forming a square airway to pass 8,125 cubic feet thus; $\left(\frac{8{,}125}{6{,}250}\right)^{\frac{2}{5}} = 1{\cdot}11065$ as the side of square if that of the original airway is 1, and therefore $1{\cdot}11065 \times 5 = 5{\cdot}55325$ as the side of the square to pass 8,125 cubic feet, and its area would be 30·8385, and perimeter 22·213.

If, however, the height is to be restricted to 5 feet, the 5·55325-foot square airways will not be available, and, in that case, proceed as in *Example* 16 to find the size of a rectangular airway. There the value of R for the 5-foot airway is 27·95. Then if x = the width in feet of the rectangular road to pass 8,125 cubic feet per minute when subject to the same pressure as the 5 feet square airway; $5x$ = its area and $10 + 2x$ = its perimeter. Therefore $\sqrt{\frac{(5x)^3}{10 + 2x}} = R$ and $x^3 = \frac{10\ R^2 + 2\ R^2 x}{125}$. The value of R for the rectangular road is $27{\cdot}95 \times \frac{8{,}125}{6{,}250} = 36{\cdot}335$, and substituting it in the above; $x^3 = \frac{10\,(36{\cdot}335)^2 + 2\,(36{\cdot}335^2)x}{125}$, $\therefore x^3 = \frac{13{,}203 + 2{,}640{\cdot}6x}{125}$. $x^3 = 21{\cdot}125x + 105{\cdot}624$, $\therefore\ x^3 - 21{\cdot}125x - 105{\cdot}624 = 0$.

Proceed to find the value of x in this cubic equation by the trigonometrical formula given in *Example* 16.

$$\text{Thus sin. } a = \frac{21{\cdot}125}{316{\cdot}872} \times 2 \times \sqrt{\frac{21{\cdot}125}{3}}$$

$$a = 20^\circ 44'$$

$$\tfrac{1}{2}a = 10^\circ 22'$$

$$\text{tan. B} = \sqrt[3]{\text{tan. } 10^\circ 22'}$$

$$\begin{array}{rl}
\text{Log. tan. } 10^\circ 22' = & 9{\cdot}262292 \\
\text{Less} & 10 \\
\text{Divide by 3 to cube root }) & \overline{1}{\cdot}262292 \\
 & \overline{1}{\cdot}754097 \\
\text{Add} & 10 \\
 & 9{\cdot}754097 \\
\end{array}$$

$$= \text{Log. tan. of } 29^\circ 34'$$

$$2\ \text{B} = 29^\circ 34' \times 2 = 59^\circ 8'$$

$$x = \frac{2\sqrt{\frac{21{\cdot}125}{3}}}{\text{sin. } 59^\circ 8'}$$

$= 6{\cdot}182$, which is the width of the road required in feet.

So that the quantity may be increased as desired by substituting a square road 8·34625 high for the original road as shown in *Example* 16, or by the use of two additional roads as well as the old one, each of which may be either 5·55325 feet square or 5 feet high by 6·182 feet wide.

Example 21.—Referring to *Example* 3 in which is given and worked out a question of an airway 7 feet by 8 passing 63,480 cubic feet of air per minute, to a point from which it is split into 4 airways, No. 1 being 6 feet by 5 feet and 400 yards long; No. 2, 6 feet square 300 yards long; No. 3, 6 feet by 4 feet 280 yards long; No. 4, 5 feet by 4 feet, 240 yards long, and in which the quantities were shown to be as follow: No. 1, 15,381·15 cubic feet; No. 2, 22,352·83 cubic feet; No. 3, 13,796·53 cubic feet; No. 4, 11,949·5 cubic feet; now if it be desirable to ascertain the sectional area necessary to pass an equal quantity along each, proceed to find the size as follows.

A total quantity of 63,480 cubic feet divided amongst 4, means $\frac{63{,}480}{4} =$ 15,870 cubic feet along each. Dealing with No. 1, it is desired to substitute a road for one 6 feet by 5 which now passes 15,381·15 cubic feet. Therefore $\left(\frac{15{,}870}{15{,}381 \cdot 15}\right)^{\frac{2}{5}} = 1 \cdot 0126$ as the ratio of the height and width to those of the original airway, if the height and width of the latter is represented by 1. Therefore, 1·0126 × 6 = 6·0756 and 1·0126 × 5 = 5·063. Therefore, for the same length of No. 1 airway as before it must be 6·0756 feet by 5·054.

Dealing next with No. 2, the quantity of which must be decreased from 22,352·83 to 15,870. Here $\left(\frac{15{,}870}{22{,}352 \cdot 83}\right)^{\frac{2}{5}} = \cdot 87196$ and ·87196 × 6 = 5·23176 as the side of the square airway for No. 2. For No. 3, $\left(\frac{15{,}870}{13{,}796 \cdot 53}\right)^{\frac{2}{5}} =$ 1·0576 and 1·0576 multiplied by 6 and 4 respectively give 6·3465 and 4·2304, the dimensions in feet of the altered No. 3 roadway. For No. 4, $\left(\frac{15{,}870}{11{,}949 \cdot 5}\right)^{\frac{2}{5}} = 1 \cdot 1202$ and this number multiplied by 5 and 4 gives 5·601 and 4·4808 respectively as the dimensions in feet of No. 4 altered airway.

If tested the value of p according to Atkinson's formula will be found equal to 5·0196 lbs. per square foot in each case of altered road, the same as given before in *Example* 3 for the original roadways. Or, if tested by the formula $R = \sqrt{\frac{a^3}{s}}$ or $\sqrt{\frac{a}{\sqrt[3]{s}}}$, or since in cases where R must be the same in each on account of roads passing equal quantities the result will not be affected if the sign of the square root be removed and then,

$$\text{For No. 1} \quad R = \frac{30 \cdot 71}{\sqrt[3]{26{,}685 \cdot 12}} = 1 \cdot 028$$

$$\text{,,} \quad 2 \quad R = \frac{27 \cdot 37}{\sqrt[3]{18{,}834 \cdot 336}} = 1 \cdot 028$$

$$\text{,,} \quad 3 \quad R = \frac{26 \cdot 844}{\sqrt[3]{17{,}767 \cdot 68}} = 1 \cdot 028$$

$$\text{,,} \quad 4 \quad R = \frac{25 \cdot 097}{\sqrt[3]{14{,}517 \cdot 792}} = 1 \cdot 028$$

Example 22.—If two square airways each 14 feet area and therefore necessarily having the same perimeters, but one of them is 800 and the other 400 yards long, both subject to the same pressure; and if the total quantity in both airways is 10,000 cubic feet per minute and it is desired to know the quantity passing in each, and also what the area of the short airway should be so that each may pass an equal quantity, proceed first as in *Example* 1. The relative quantities passing into the two airways is as $\sqrt{2} : \sqrt{1}$ or as 1·4142 : 1. The sum of these relative quantities is 2·4142, and therefore the actual quantities passing are

For the 800-yard airway as 2·4142 : 1 : : 10,000 : 4,142 cub. ft.
„ 400 do. 2·4142 : 1·4142 : : 10,000 : 5,858 cub. ft.

Next, if each airway is to have an equal quantity passing through it with the same pressure, proceed by the formula $R = \sqrt{\frac{a^3}{s}}$, but as the quantities in the two airways are to be equal, the result is not affected if the square root sign be omitted. For the 14 feet area airway, which is 400 yards long $R = \frac{14^3}{17{,}960} = {\cdot}1527845$. If $a =$ the area of the square airway 800 yards or 2,400 feet long, which, being subject to the same pressure as the 14 feet area one, is to pass a similar quantity of 5,858 cubic feet per minute, then its perimeter is represented by $4\sqrt{a}$ or $\sqrt{16a}$. The value of R therefore for the proposed airway is $\frac{a^3}{2{,}400\sqrt{16a}} = {\cdot}1527845$ and $a^3 = 366{\cdot}68 \times \sqrt{16a}$; $a^3 = \sqrt{2{,}151{,}300\,a}$; $a^6 = 2{,}151{,}300a$; $a^5 = 2{,}151{,}300$ and $a = 18{\cdot}473$ which is the area the longer airway must be made if the shorter one is to remain 14 feet area and subject to the same pressure, or, if the long one is to remain 14 feet, then the shorter one would be

As 18·473 : 14 : : 14 : 10·61 feet area.

A square airway 10·61 feet area 400 yards long, and a square airway 14 feet area 800 yards long, both subjected to the same pressure, would always pass equal quantities of air, whether the pressure was increased or decreased, as would also a square airway 14 feet area 400 yards long and a square airway 18·473 feet area 800 yards long under the same circumstances.

If the pressure per square foot was maintained, the quantity passing in each airway would be 4,142 cubic feet per minute or a total of 8,284 cubic feet per minute, as a result of reducing the short airway from 14 to 10·61 feet area so that the quantity of air circulating would thus be reduced 10,000 − 8,284 = 1,716 cubic feet per minute. On the other hand as a result of enlarging the longer airway to 18·473 feet area, there would be 5,858 cubic feet passing in each airway if the pressure per square foot is maintained, or a total of 11,716 cubic feet for the two airways, showing in that case an increase of 11,716 − 10,000 = 1,716 cubic feet.

If it were desirable to have the same total quantity passing as before with the same pressure per square foot, then in that case observe that the areas of both airways must be altered. They would then pass 5,000 cubic feet each and the question to solve becomes; if a 14 feet area square airway 400 yards long passes 5,828 cubic feet per minute, what must its area be to pass 5,000 cubic feet if the pressure remain constant? The side of a 14 feet area square airway is $\sqrt{14} = 3{\cdot}74166$, and therefore the value of the side of the altered airway would be $\left(\frac{5{,}000}{5{,}858}\right)^{\frac{2}{5}} \times 3{\cdot}74166 = 3{\cdot}512$ and its area $3{\cdot}512^2 = 12{\cdot}334$ which is the area of the 400-yard airway; and similarly for the longer airway the question would become; if a 14 feet area square airway 800 yards long passes 4,142 cubic feet per minute, what must its area be to pass 5,000 cubic feet per minute with the same pressure?

Here $\left(\frac{5{,}000}{4{,}142}\right)^{\frac{2}{5}} \times 3{\cdot}74166 = 4{\cdot}0343$ and $4{\cdot}0343^2 = 16{\cdot}2755$ as the area of the 800-yard airway.

A check on the accuracy of these figures is given by proportion thus; As 18·473 : 16·2755 : : 14 : 12·334, showing that the relative proportions hold good for any quantities of air.

The questions bearing upon ventilation arising from different pressures have been considered; and it will now be well to give examples on the power producing ventilation. The student must remember that *power* and *pressure* are different things; pressure being the force per square foot producing the ventilation, while power (often expressed by *u*) is the quantity passing multiplied by the pressure.

The quantity of air passing is according to the cube root of the power applied, and the power necessary to produce ventilation as the cube of the quantity.

Example 23.—If 50,000 cubic feet of air circulate in a mine and it is to be trebled, or the quantity is to be increased to 150,000 cubic feet by increasing the power, the circumstances of the mine remaining unaltered, then the power must be increased 3^3 or 27 times to attain that object.

Example 24.—If 20,000 cubic feet circulate by the expenditure of 64,000 units of work, and the units of work be increased to 343,000, the quantity which would then pass will be found thus:—

As $\sqrt[3]{64{,}000} : \sqrt[3]{343{,}000} : : 20{,}000 : 35{,}000$ cubic feet.

Example 25.—It may be desirable to know the quantity which will circulate by the combined working of a furnace and a fan, or other means when the quantity obtained by each separately is known. Thus, if a mine circulates 20,000 cubic feet of air per minute by the furnace alone, and 30,000 cubic feet by the fan alone, proceed to find the quantity of air that will pass when the two act together as follows:—

As already said, the power required to ventilate a given mine varies as the cube of the quantity. If, therefore, a mine circulated 20,000 cubic feet of air per minute and 1 represent the power used, then when 30,000 cubic feet are circulated, whether by fan or other means, the power will be, As $20{,}000^3 : 30{,}000^3 : : 1 : 3{\cdot}375$, or it may be put $\frac{30{,}000}{20{,}000} = 1\frac{1}{2}$, and $\left(1\frac{1}{2}\right)^3 = 3{\cdot}375$. Therefore, it requires 3·375 times more power to circulate 30,000 cubic feet than 20,000 cubic feet. The combined powers would be then $3{\cdot}375 + 1 = 4{\cdot}375$, and as the quantity varies as the cube root of the powers, then for both furnace and fan.

As $\sqrt[3]{1} : \sqrt[3]{4{\cdot}375} : : 20{,}000 : 32{,}710$ cubic feet, which is the quantity circulating when the furnace and fan act together.

Example 26.—If two circular airways are each 1,000 yards long, but one of them is 4 feet in diameter, and the other 6, and it is required to know the relative powers for circulating a given quantity through each, say 10,000 cubic feet per minute.

If *u* represents the power and *q* the quantity in cubic feet per minute, then $u = \frac{ksv^2q}{a}$ because $u = q \times p$ and the value of *u* for the 4-foot diameter airway is

$$u = \frac{{\cdot}0000000217 \times 1{,}000 \times 3 \times 4 \times 3{\cdot}14159 \times \left(\frac{10{,}000}{4^2 \times {\cdot}7854}\right)^2 \times 10{,}000}{4^2 \times {\cdot}7854} = 412{,}250$$

units of work, and for the 6-foot diameter airway,

$$u = \frac{{\cdot}0000000217 \times 1{,}000 \times 3 \times 6 \times 3{\cdot}14159 \times \left(\frac{10{,}000}{6^2 \times {\cdot}7854}\right) \times 10{,}000}{6^2 \times {\cdot}7854} = 54{,}288$$

units of work.

If these units of work be taken and the proportion between them ascertained, thus, As 54,288 : 412,250 : : 1 : 7·593, observe that the same proportion exists between the 5th power of the larger airway's diameter and the 5th power of the smaller airway's diameter, thus, As $4^5 : 6^5 :: 1 : 7{\cdot}593$, therefore $\frac{412,250}{54,288} = \frac{6^5}{4^5}$ or $\left(\frac{6}{4}\right)^5 = 7{\cdot}593$.

From this is obtained the following rule. For circular airways of equal lengths to pass equal quantities of air, the relative powers applied to them must be in inverse proportion to the fifth power of their diameters; thus to pass an equal quantity through a 4 foot diameter and a 6-foot diameter airway it requires $\frac{6^5}{4^5} = \left(\frac{6}{4}\right)^5 = 1{\cdot}5^5 = 7{\cdot}593$ times the power for the 4-foot diameter than for the 6-foot diameter airway. For a 6-foot and an 18-foot diameter airway it would take $\left(\frac{18}{6}\right)^5 = 3^5 = 243$ times the power to pass the same quantity in the 6-foot as in the 18-foot diameter airway.

Example 27.—To find the value of p in the preceding example of a 4-foot and a 6-foot diameter airway, proceed thus, $p = \frac{u}{q}$, that is :—

For the 4-foot diameter airway $p = \frac{412,250}{10,000} = 41{\cdot}225$ lbs.

and „ „ 6 „ do, $p = \frac{54,288}{10,000} = 5{\cdot}4288$ lbs.

Example 28.—To ascertain the quantity that would pass in the 6-foot airway with the expenditure of the same power as in the 4-foot airway;

As $\sqrt[3]{54,288} : \sqrt[3]{412,250} :: 10,000 : 19,655$ cubic feet, or the quantity which will pass in circular airways with the same power is in inverse proportion to the cube root of the relative power, for taking the cube root of the relative power in *Example* 26, and multiplying it by the quantity there stated to be passing, thus $\sqrt[3]{7{\cdot}593} \times 10,000 = 19,655$ cubic feet, which agrees with the quantity ascertained in the present example.

Example 29.—Again, if the pressure per square foot of 41·225 lbs. applied in the 4-foot diameter airway were applied to the 6-foot diameter airway, then $v = \sqrt{\frac{pa}{ks}} = \sqrt{\frac{41{\cdot}225 \times 6^2 \times {\cdot}7854}{{\cdot}0000000217 \times 1,000 \times 3 \times 6 \times 3{\cdot}14159}} = 974{\cdot}62$, and therefore the quantity passing is $974{\cdot}62 \times 6^2 \times {\cdot}7854 = 27,557$ cubic feet, or the quantity passing may be found thus; since the quantity varies as the square root of the pressures; as $\sqrt{5{\cdot}4288} : \sqrt{41{\cdot}225} :: 10,000 : 27,557$ cubic feet. It is now shown that the quantity which will pass with a constant pressure is in inverse proportion to the square root of the relative power, for $\sqrt{7{\cdot}593} \times 10,000 = 27,557$ cubic feet as before given.

Example 30.—That the same rules are given in *Examples* 26, 28, and 29, for circular airways hold good for square airways may be proved by taking the airway given in *Example* 16 of 5 feet by 5, 8,000 feet long, and passing 6,250 cubic feet per minute. The value of p for this has been shown to be 8·68 lbs. in *Example* 16. Take a square airway 10 feet by 10, 8,000 feet long, and passing 6,250 cubic feet per minute, and the pressure $p = \frac{{\cdot}0000000217 \times 320.000 \times 62{\cdot}5^2}{100} =$

·27125 and $\frac{8{\cdot}68}{{\cdot}27125} = 32$ as the ratio between the pressures of the two airways. Now take the sides of the two square airways; divide one into the other and take the 5th power of the remainder; in other words take the 5th power of the ratio between the two lengths of sides forming the airways and the result is 32, thus $\frac{10^5}{5^5}$ or $\left(\frac{10}{5}\right)^5 = 32$.

Therefore the rule to find the relative powers for square airways of equal length to pass equal quantities of air is to remember that they are in inverse proportion to the power of their sides. Thus to pass an equal quantity through a 5-foot square and a 10-foot square airway it requires $\left(\frac{10}{5}\right)^5 = 32$ times the power.

Example 31.—Again, it may be shown that the same rules apply equally to rectangular airways, for take a road 4 feet high by 6 feet wide, 1,000 feet long, and passing 5,000 cubic feet of air per minute. Here $p = \frac{{\cdot}0000000217 \times 20{,}000 \times 208{\cdot}3^2}{24}$ $= {\cdot}7846$. Now find the value of p for a rectangular road 8 feet high by 12 feet wide to pass the same quantity along the same length of road. $p = \frac{{\cdot}0000000217 \times 40{,}000 \times 52{\cdot}083^2}{96} = {\cdot}02452$ and the ratio between the pressures in the two cases is $\frac{{\cdot}7846}{{\cdot}02452} = 32$. But take either the heights or widths of the two roads and find the ratio between them by dividing one into the other and then take the 5th power of the remainder, and the result is 32, thus $\left(\frac{8}{4}\right)^5$ or $\left(\frac{12}{6}\right)^5 = 32$. It is true that in using the rule for rectangular roads the form of the altered roadway must be similar to the original one, or in other words the rule is not applicable unless the proportion of the width to the height is the same in the altered airway as that prevailing in the original one. Bearing this in mind, the relative powers for rectangular airways of similar form and of equal length may be found to pass equal quantities of air by taking the proportion of the fifth power of either the two widths or the two heights of the roads.

Example 32.—If a mine be ventilated by one air current, and with a given power there are obtained 10,000 cubic feet through it, the quantities that would pass if the air be divided into 2, 3, 4, 5 or 10 equal splits, the power remaining the same would be in direct proportion to the area. That is with two splits there would be 20,000, with 3 splits 30,000, with 4 splits 40,000, with 5 splits 50,000, and with 10 splits 100,000 cubic feet. This is apparent when the rule to find the quantity that will pass along airways when the power u is given is observed closely. It is, $q = \sqrt[3]{\frac{u}{ks}} \times a$. In the case of the 2 splits, 3 splits, &c. the values of u, k, and s, are equal in each case, and only a would vary. It will be seen that this is a very different result from that obtained by splitting the air into a number of currents when the *pressure* remains the same as worked out in *Example* 6.

Example 33.—From the formula $q = \sqrt[3]{\frac{u}{ks}} \times a$ is deduced $u = ks \times \frac{q^3}{a^3}$ or $ks \left(\frac{q}{a}\right)^3$ thus:

$$q = \sqrt[3]{\frac{u}{ks}} \times a$$

$$\therefore q^3 = \frac{u}{ks} \times a^3$$

$$\frac{q^3}{a^3} = \frac{u}{ks}$$

$$u = ks \times \frac{q^3}{a^3} \text{ or } ks \left(\frac{q}{a}\right)^3 \text{ or } ksv^3.$$

Example 34.—The rule to find the horse-power producing the ventilation is to divide the units of work by 33,000. If the quantity of air passing in a mine is 50,000 cubic feet per minute, with a 1·5-inch water-gauge, the horse-power producing the ventilation will be found thus :—$\frac{50{,}000 \times 1{\cdot}5 \times 5{\cdot}2}{33{,}000} = 11{\cdot}82$ horse-power.

Example 35.—If 20,000 cubic feet of air pass in a circular airway of 12 feet diameter, what quantity will pass in one of 6 feet diameter ?

As $\sqrt[3]{\left(\frac{12}{6}\right)^5} : \sqrt[3]{1} :: 20{,}000$ or $\frac{20{,}000}{\sqrt[3]{32}} = 6{,}300$ cubic feet per minute.

Example 36.—If a furnace and steam jet combined produce 50,000 cubic feet of air per minute, and the furnace alone 44,000 cubic feet, and it is required to know what the steam jet alone would produce, set down the power of the furnace which alone produces 44,000 cubic feet per minute at 1, then the power of the furnace and steam jet combined will be in the ratio to the power of the furnace as $44{,}000^3 : 50{,}000^3 :: 1 : 1{\cdot}46741$ or $\left(\frac{50{,}000}{44{,}000}\right)^3 = 1{\cdot}46741 : 1$, therefore the power of the steam jet is ·46741. To find the quantity the steam jet alone will produce if a power of 1 produces 44,000 cubic feet proceed thus—As $\sqrt[3]{1} : \sqrt[3]{{\cdot}46741} :: 44{,}000 : 34{,}147$, or say if a power of 1·46741 produces 50,000, what will the power of ·46741 produce, thus—As $\sqrt[3]{1{\cdot}46741} : \sqrt[3]{{\cdot}46741} :: 50{,}000 : 34{,}147$, which is the quantity in cubic feet per minute the steam jet alone will produce.

Example 37.—If by applying 50,000 units of work 40,000 cubic feet of air are produced, and it is desired to know how many units will be required to circulate 75,000 cubic feet, then as $40{,}000^3 : 75{,}000^3 :: 50{,}000 : 329{,}590$.

Examples have already been worked out showing how the area of an air-course alters to pass different quantities of air with the same pressure, and in these examples the power must vary directly with the increased or decreased quantity.

Example 38.—Take for instance that of an aircourse 8,000 feet long, 5 feet by 5, passing 6,250 cubic feet per minute, and it is proposed to enlarge the area so as to pass 22,500 cubic feet per minute, the pressure remaining the same, which is once more using the figures of *Example* 16. The power u would be increased $\frac{22{,}500}{6{,}250} = 3{\cdot}6$ times. That this is so must be evident because $u = p \times q$. In the first case the power would be $8{\cdot}68 \times 6{,}250$ and in the second $8{\cdot}68 \times 22{,}500$. Also take the formula $q = \sqrt[3]{\frac{u}{ks}} \times a$ to find the quantities in each case, and

In the first, $q = \sqrt[3]{\dfrac{8{\cdot}68 \times 6{,}250}{{\cdot}0000000217 \times 160{,}000}} \times 25 = 6{,}250$ cubic feet.

In the second, $q = \sqrt[3]{\dfrac{8{\cdot}68 \times 22{,}500}{{\cdot}0000000217 \times 286{,}080}} \times 71{\cdot}28 = 22{,}500$ cubic feet.

If required to increase the quantity from 6,250 cubic feet per minute without altering the 5 feet by 5 feet airway the power must be increased thus :—

As $6{,}250^3 : 22{,}500^3 :: 1$ or $\left(\dfrac{22{,}500}{6{,}250}\right)^3 = 46{\cdot}656$ times.

In other words, if the quantity were increased by enlarging the airway as indicated from a 25-foot area one to a 6-foot by 11·88 or 71·28-foot area one, then by the consumption of 3·6 times the amount of coals previously consumed there would be an increase in the quantity of air from 6,250 cubic feet per minute to 22,500 cubic feet per minute ; but had it been attempted to increase the quantity of air to the same extent while the airway remained the same the increased consumption of coal would be 46·656 times.

If the airway had been altered from 5 feet by 5 to 6 feet by 11·88, and it became desirable to have the original quantity of 6,250 cubic feet per minute through the altered airway, the consumption of coals necessary to pass this quantity will be found as follows :

The value of p in the original aircourse has been shown to be 8·68 lbs. For the altered aircourse it would be $p = \dfrac{{\cdot}0000000217 \times 286{,}080 \times 87{\cdot}68^2}{71{\cdot}28} = {\cdot}66954$ lb. and since $u = p \times q$, in each case it would be $8{\cdot}68 \times 6{,}250$ and ${\cdot}66954 \times 6{,}250$.

The quantity 6,250 being common to both, the value of u is in direct proportion to $p \therefore \dfrac{8{\cdot}68}{{\cdot}66954} = 12{\cdot}964$ times less coal required with the enlarged airway to pass the same quantity of air.

Again, by allowing the power to remain the same after the alteration in the airway from 5 feet square to 6 feet by 11·88, then the quantity passing along the enlarged airway will be found thus: for the original airway $u = ks\left(\dfrac{q}{a}\right)^3$, and therefore ${\cdot}0000000217 \times 160{,}000 \times \left(\dfrac{6{,}250}{25}\right)^3 = 54{,}250$, and applying $q = \sqrt[3]{\dfrac{u}{ks}} \times a$ to the altered airway $\sqrt[3]{\dfrac{54{,}250}{{\cdot}0000000217 \times 286{,}080}} \times 71{\cdot}28 = 14{,}682$ cubic feet per minute, and the velocity would be $\dfrac{14{,}682}{71{\cdot}28} = 205{\cdot}977$ feet per minute, and in that case the value of p for the altered airway is $= \dfrac{{\cdot}0000000217 \times 286{,}080 \times 205{\cdot}977^2}{71{\cdot}28} = 3{\cdot}695$ lbs. as compared with 8·68 lbs. pressure for the original airway to pass 6,250 cubic feet per minute.

It must be borne in mind that the examples which have been worked out refer entirely to the overcoming of frictional resistances, because for simplicity the pressure necessary for velocity need not be considered—it is so little. The velocity of air without resistance is the same that a body would attain in falling the height of the motive column, so that if there is a difference of pressure equal to 36 feet of air column, the theoretical velocity of the air would be $8\sqrt{M}$, where $M =$ motive column in feet or $8\sqrt{36} = 48$ feet per second, because a falling body under the force of gravity would attain a velocity of 8 times the square root of 36 or 48 feet. The co-efficient of friction ·0000217 lb. per square foot of area of section is applicable only to the underground roadways. That for the shafts of different forms has not been definitely fixed by experiment. It is usual, however, to take ·00001 lb. per square foot for circular shafts, where the smooth surface gives little impediment to the passage of air.

As a further assistance, questions with answers are now submitted and worked out, bearing on the subject of ventilation.

Question 82.—There are 6,000 cubic feet of air passing, what power will be required to pass 12,000 cubic feet, the airway remaining in the same state ?

The velocity of the air will be doubled, necessitating four times the pressure on each square foot of area because the friction varies as the square of the velocity. Also this increased pressure per square foot has to be sustained at a double velocity, hence the power varies as the cube of the quantity. In this case the double quantity will require $2^3 = 8$ times the power.

Question 83.—A certain number of shots have to be fired in a mine, how should a fireman be instructed to proceed ? What other duties has he to attend to ?

In such cases a fireman should fire the shots in succession, beginning at the return side and travelling in the direction contrary to that of the current, so that any fire-damp liberated by a shot may not be exploded by the firing of a succeeding shot. His duties are to keep the airways free from all obstructions that would impede the circulation of the air. Further he should see that the regulators are in good condition, properly adjusted and locked; that all doors in use are made to fall to or close themselves without leakage of air, and that those not in use are taken off their hinges; that the roof and sides of the airways are safely timbered or walled up ; that all projecting or loose stones are removed or timbered, so that the road may be safe to travel in; that all stoppings, air-crossings and brattices are in good order and do not allow the air to escape ; he should examine all cavities in the roof and open spaces in the goaves for gas, and if gas be present adopt means to remove it. If there are any special features in the Colliery, such as gob fires, blowers of gas, &c., the fireman should receive special instructions concerning them.

Question 84.—The total quantity of air passing over a furnace is 200,000 cubic feet per minute, burning 5 tons of coal per 12 hours, how much increase of air would be obtained by adding 5 more tons of coal for the 12 hours to the furnace ?

The power increases in the same ratio as the coals burned are increased, which in this case is as 1 : 2, and as the quantity of air increases in direct proportion to the cube root of the powers, then as $\sqrt[3]{1} : \sqrt[3]{2} :: 200{,}000$, or as $1 : 1{\cdot}25992 :: 200{,}000 : 251{,}984$, which is the quantity circulating after the power is added to, therefore the net increase is $251{,}984 - 200{,}000 = 51{,}984$ cubic feet.

Question 85.—A regulator is placed in the return airway of a district, and a blower of gas is discovered in that return airway which comes against the air through the regulator. It is impossible to add more air to that district, how then must the gas be prevented from finding its way into the district ?

Under such unusual circumstances, reduce the area of the regulator until the velocity of the air is sufficient to resist the pressure of the gas, and as this act by itself will reduce the quantity of air in that district, if practicable enlarge the airway or provide an additional one on the outbye side of the blower. If this does not meet the case, additional aircourses must be devised.

Question 86.—If one ventilating fan produces 25,000 cubic feet of air per minute, what total quantity of air will be given if another fan of the same dimensions be added to it and worked with an equal power?

As $\sqrt[3]{1} : \sqrt[3]{2} :: 25{,}000$, or as $1 : 1{\cdot}25992 :: 25{,}000 : 31{,}498$ cubic feet per minute.

Question 87.—What is the general efficiency of fans? If the horse-power of an engine is 40, and the water-gauge is 1·5 inch, what quantity of air would you expect to get?

The general result obtained from a fan is about 50 per cent. of the indicated horse-power of the engine, realised in useful work done on the air. With an engine of 40 indicated horse-power the horse-power in the air therefore would be as 100 : 50 :: 40 : 20, and as the rule is; horse-power in the air $= \frac{\text{Quantity in cubic feet per minute} \times \text{water-gauge} \times 5{\cdot}2}{33{,}000}$ ∴ the quantity $= \frac{\text{H. P} \times 33{,}000}{\text{water-gauge} \times 5{\cdot}2}$ ∴ $\frac{20 \times 33{,}000}{1{\cdot}5 \times 5{\cdot}2} = 84{,}615$, which is the quantity in cubic feet per minute we may expect to get.

Question 88.—The temperature in the downcast is 50° F., that in the upcast is 100° F. What would be the increase in the velocity of the air by doubling the depth of the pits, the temperature remaining the same?

The power of a furnace consists in the difference it can produce between the pressures per square foot at the bottom of the two shafts, multiplied by the velocity at which this difference is maintained, and as the said difference of pressure depends on the difference between the weights of a cubic foot of air in the two shafts multiplied by the depth of the upcast shaft, therefore if the temperature in the shafts remains constant, so also will the weight of given volumes of air, and hence the pressure produced by the furnace will then vary as the depth of the upcast shaft.

By doubling the depth of the upcast shaft the pressure produced will be doubled, and as the velocity varies as the square root of the pressure giving rise to it, the velocity will be increased $\sqrt{2} = 1{\cdot}4142$ times. The power of the furnace would be $1{\cdot}4142 \times 2 = 2{\cdot}8284$ times as great in the latter case. The quantity of coals consumed will be a little more than 1·4142 times, for the quantity of air to be heated is increased to that extent, and the initial temperature will need to be higher in the last case to get the same average temperature in the deeper shaft. This is disregarding the extra friction added by lengthening the upcast shaft, which would reduce the quantity, and this may or may not be to a serious extent according as the shaft friction is or is not a large proportion of the friction of the whole mine.

Question 89.—A barometer registers 30 inches at the surface, what will it register at a point 1,890 feet below the surface?

By the formula given under the remarks on barometer in Chapter XI., $I = \frac{D \times B}{26{,}216}$; $I = \frac{1{,}890 \times 30}{26{,}216} = 2{\cdot}16$, therefore the barometer would register $30 + 2{\cdot}16 = 32{\cdot}16$ inches at a point 1,890 feet below the surface. Or say that a barometer rises $\frac{1}{10}$th of an inch for every 88 feet of descent. Therefore $\frac{1890}{88} = 21{\cdot}5$, which divided by 10 gives 2·15 as the rise of the barometer for the depth given, almost as before.

Question 90.—Give the size of two airways whose perimeters are equal; the area of one being one and a half times larger than that of the other.

Here 9 feet × 6 feet = 30 feet perimeter and area 54 feet.
12 „ × 3 „ = 30 „ „ „ 36 feet and
36 × $1\frac{1}{2}$ = 54 feet.

Question 91.—How much more resistance will a current of 600 feet per minute meet with than one 500 feet per minute, the aircourse being the same? The latter has a water-gauge of ·76 inch, what will that of the former be?

The resistance is as the square of the velocity while the aircourse remains the same, therefore if the velocity be increased from 500 feet to 600 feet per minute the resistance will be as $5^2 : 6^2 = \frac{36}{25} = 1{\cdot}44$ times greater, and the water-gauge would be ·76 × 1·44 = 1·0944 inch.

Question 92.—The quantity of air which must be made to flow in a new mine is 116,640 cubic feet per minute. What size ought the shaft to be for the air to have a mean velocity in the shaft of $7\frac{1}{2}$ feet per second.

Here $7\frac{1}{2}$ feet per second = 450 feet per minute and $\frac{116{,}640}{450} = 259{\cdot}2$ square feet area of shaft. Its diameter therefore is $\sqrt{\frac{259{\cdot}2}{{\cdot}7854}} = 18{\cdot}166$ or 18 feet 2 inches.

Question 93.—There are two airways; the first is 8 feet square, the second 10 feet by 4. The same quantity of air is wanted to be passed in each airway; the water-gauge stands at ·2 inch in the first, what will it register in the second case?

By Atkinson's formula $p = \frac{ksv^2}{a}$, and as k is common to both airways omit it and for s substitute the value of o. The relative velocities will be in proportion to the areas of the roads, viz., as 40 : 64, or as 5 : 8. The value of p in the two roadways will then be in the following proportion:—

As $\frac{32 \times 5^2}{64} : \frac{28 \times 8^2}{40} = 12{\cdot}5 : 44{\cdot}8$, and therefore the pressure in the smaller airway is $\frac{44{\cdot}8}{12{\cdot}5} = 3{\cdot}584$ times that of the larger, and the water-gauge in the larger being ·2 inch, that of the smaller is ·2 × 3·584 = ·7168 inch.

Question 94.—What horse-power would an engine exert when yielding 60 per cent. of duty to move 100,000 cubic feet of air a minute; the water-gauge stands at 1 inch?

$\frac{100{,}000 \times 5{\cdot}2}{33{,}000} = 15{\cdot}75$ horse-power effective ∴
As 60 : 100 :: 15·75 : 26·25 horse-power exerted by engine.

Question 95.—In an airway 10 feet by 6 passing 30,000 cubic feet of air per minute, with a velocity of 500 lineal feet per minute, the water-gauge stands at ·5 inch. What will the water-gauge show if the same quantity be passed through an airway 8 feet by 5?

The perimeters are 32 and 26; therefore, the rubbing surface is as 16 : 13; and the areas are 60 and 40 feet, or in the proportion of 3 to 2. The velocity in the small airway would be as 2 : 3 :: 500 : 750 feet per minute. The pressures, then, would be in proportion thus: as $\frac{16 \times 5^2}{60} : \frac{13 \times 7{\cdot}5^2}{40}$ or as 6·6 : 18·28, and therefore the pressure in the smaller airway is $\frac{18{\cdot}28}{6{\cdot}6} = 2{\cdot}744$ times more than that of the larger, and the water-gauge of the larger being ·5, that of the smaller would be $\cdot 5 \times 2{\cdot}744 = 1{\cdot}372$ inch to pass equal quantities of air.

> *Question* 96.—20,000 cubic feet of air per minute are required through an airway 4,000 feet long by the application of 5 horse-power. Of what area must it be to give the desired result, (*a*) if square, (*b*) if circular, and (*c*) if rectangular, supposing the seam to be 6 feet high?

By Atkinson's formula $a = \frac{ksv^2}{p}$, and in the case of a square airway if a be its area, then its perimeter is $4\sqrt{a}$, and as the quantity of air is found by multiplying the area by the velocity, $v = \frac{q}{a}$ where q represents the quantity. For $a = \frac{ksv^2}{p}$, then substitute $a = \frac{k(4\sqrt{a}\,l) \times \left(\frac{q}{a}\right)^2}{p}$; l, being the length of air-course in feet. By collecting terms $\frac{a^3}{\sqrt{a}} = \frac{4\,k\,l\,q^2}{p}$, and by squaring both sides $\frac{a^6}{a} = \left(\frac{4\,k\,l\,q^2}{p}\right)^2 \therefore a^5 = \left(\frac{4\,k\,l\,q^2}{p}\right)^2$ and $a = \left(\frac{4\,k\,l\,q^2}{p}\right)^{\frac{2}{5}}$.

Since the power is the quantity passing multiplied by the pressure $a = \left(\frac{4\,k\,l\,q^3}{pq}\right)^{\frac{2}{5}} = \left(\frac{4\,k\,l\,q^3}{u}\right)^{\frac{2}{5}}$ where u is the power expressed in units of work.

Substituting the known values in the equation, thus

$a = \left(\frac{4 \times {\cdot}0000000217 \times 4{,}000 \times 20{,}000^3}{33{,}000 \times 5}\right)^{\frac{2}{5}} = 49{\cdot}0314$ square feet as the area, or rather more than 7 feet square.

In the case of a circular airway, if a represent its area, the perimeter will be $3{\cdot}545\sqrt{a}$, and the formula will be $a = \left(\frac{3{\cdot}545\,k\,l\,q^2}{p}\right)^{\frac{2}{5}}$ or $\left(\frac{3{\cdot}545\,k\,l\,q^3}{u}\right)^{\frac{2}{5}}$, and substituting the known values in this equation

$a = \left(\frac{3{\cdot}545 \times {\cdot}0000000217 \times 4{,}000 \times 20{,}000^3}{33{,}000 \times 5}\right)^{\frac{2}{5}} = 46{\cdot}72$; or, knowing the size of the square airway, proceed to find the size of a circular one to pass the same quantity, thus:—

The perimeter of the square airway is $7 \times 4 = 28$ feet, and $R = \sqrt{\frac{49^3}{28}}$.

Therefore, $\sqrt{\frac{49^3}{28}}$ or omitting the sign of the square root, since the value of R is the same in each case, $\frac{49^3}{28} = 4{,}201{\cdot}75$. Then, as per *Example* 16, but allowing for the square root sign removal, the area a of the circular airway is $= \sqrt[5]{12{\cdot}567\,R^2}$. $\therefore a = \sqrt[5]{12{\cdot}567 \times 4201{\cdot}75^2} = 46{\cdot}7$ as before.

In the case of a rectangular road whose height is given as 6 feet let $x =$ the width of the road in feet. Then $6\,x$ = its area, and $12 + 2\,x$ its perimeter. Substituting these in the formula $a = \frac{ksv^2}{p}$

$$6\,x = \frac{kl\,(12 + 2\,x) \times \left(\frac{q}{6\,x}\right)^2}{p}$$

dividing both sides by $\left(\frac{q}{6\,x}\right)^2$,

$$\frac{\frac{6\,x}{q^2}}{36\,x^2} = \frac{kl\,(12 + 2\,x)}{p}$$

$$\therefore \frac{216\,x^3}{q^2} = \frac{kl\,(12 + 2\,x)}{p},$$

$$216\,x^3 = \frac{klq^2\,(12 + 2\,x)}{p},$$

$$x^3 = \frac{12\,klq^2 + 2\,klq^2x}{216\,p} \text{ or } \frac{12\,klq^3 + 2\,klq^3x}{216\,u}.$$

Now substitute the known values of k l q and u, thus

$$x^3 = \frac{\{12 \times {\cdot}0000000217 \times 4{,}000 \times (20{,}000)^3\} + \{2 \times {\cdot}0000000217 \times 4{,}000 \times (20{,}000^3\}\,x}{216 \times 33{,}000 \times 5}$$

$$x^3 = \frac{8{,}332{,}800{,}000 + 1{,}388{,}800{,}000\,x}{{\cdot}35.640{,}00},$$

$$x^3 = 38{\cdot}9\,x + 233{\cdot}8,$$

$$x^3 - 38{\cdot}6\,x - 233{\cdot}8 = 0.$$

Or the same cubic equation may be obtained another way.

Knowing the size of the square airway, proceed to substitute a rectangular one for it to pass the same quantity; thus $R = \frac{(6\,x)^3}{12 + 2\,x}$, and therefore, as shown in *Example* 16, but allowing for the sign of the square root there used and now omitted, $x^3 = \frac{12\,R + 2\,R\,x}{216}$, and substituting the known value of R

$$x^3 = \frac{(12 \times 4{,}201{\cdot}75) + (2 \times 4{,}201{\cdot}75\,x)}{216},$$

$$x^3 = \frac{8{,}403{\cdot}5\,x + 50{,}421}{216} = 38{\cdot}9\,x + 233{\cdot}4$$

$$x^3 - 38{\cdot}9\,x - 233{\cdot}4, = 0.$$

Proceed to work out this cubic equation by the trigonometrical formula,

$$x^3 - px - Q = 0.$$

$$\text{Sin. } a = \frac{p}{3\,Q}\,2\sqrt{\frac{p}{3}}$$

$$\text{Tan. } \beta = \sqrt[3]{\text{tan. } \tfrac{1}{2}\,a}$$

$$x = \frac{2\sqrt{\frac{p}{3}}}{\text{Sin. } 2\,\beta}$$

Substituting the values of p and Q in the formula,

$$\text{Sin. } a = \frac{38{\cdot}9}{700{\cdot}2} \times 2 \times \sqrt{\frac{38{\cdot}9}{3}}$$

$$a = 23°36'$$

$$\text{and } \tfrac{1}{2}\,a = 11°48'$$

$$\text{Tan. } \beta = \text{tan. } \sqrt[3]{11°48'}$$

Log. tan. 11°48′ = 9·319961
less 10
divide by 3 to cube root) $\bar{1}$·319961
$\bar{1}$·773320
add 10
9·773320
= log. tan. of 30°41′
$2\,\beta = 61°22'$

$$x = \frac{2\sqrt{\frac{38{\cdot}9}{3}}}{\text{Sin. } 61°22'}$$

$x = 8{\cdot}3$

The size of the rectangular road to pass the required quantity therefore is 6 feet × 8·3, or 49·8 square feet area.

Question 97.—The temperature of the air in the downcast is 60° F., and in the upcast 100° F.; the quantity of air entering the mine by the downcast is 20,000 cubic feet per minute, and the size of each shaft is 15 feet in diameter. What will be the velocity of air in feet per minute in each pit respectively?

The velocity in the downcast would be $\frac{20{,}000}{15^2 \times {\cdot}7854} = 113{\cdot}176$ feet per minute. Since 460 cubic feet of air at 0° expand a cubic foot for each degree of heat (Fahrenheit) added, therefore 460 cubic feet at 0° become 460 + 60 = 520 cubic feet at 60° and 460 + 100 = 560 cubic feet at 100°. Therefore, as 520 : 560 :: 20,000 : the volume occupied by 20,000 cubic feet at 100°, or since the areas of the upcast and downcast are the same, the velocity in the downcast may be proportioned to that in the upcast thus:—As 520 : 560 :: 113·176 : 121·88, which is the velocity in the upcast shaft.

Question 98.—How far can you drive a pair of levels without showing a "cap" in the safety lamp at the upcast? The current of air is 4,000 cubic feet per minute, and assuming that the coal gives off 1 cubic foot of fire-damp for every 30 yards of each level, give the proportions of air and gas in the mixture.

Light carburetted hydrogen gas can be detected by the lamp when it exists in the air in the proportion of 1 in 30. In 30 yards of the level the quantity of gas in the air will be one-four-thousandth. ∴ As 30 : 4,000 :: 30 : 4,000, but as there is a pair of levels divide by 2 = 2,000. That is, the levels can be driven 2,000 yards from the pit before the return air will show a cap on the flame at the upcast if there is no change in the quantity of gas emitted during the driving. The proportions of the mixture at the upcast will be $\frac{1}{30}$th of gas in the air current.

Question 99.—A pair of winning places 100 yards long each have stentons connecting them 20 yards apart, the winning places being separated by 10 yards of coal. The winning places and stentons are all filled with undiluted gas CH_4; current of air passing 6,000 feet per minute. What is the least possible time to clear this gas out without fouling the air, except with small cap on top of a safety-lamp? Places all are 8 feet wide and 5 feet high.

Here $\frac{100}{20}$ = 5 stentons 10 yards long = 50 yards, and the winning places 100 × 2 = 200 yards. 200 + 50 = 250 × 3 × 40 the area = 30,000, which is the total number of cubic feet of undiluted gas.

If $\frac{1}{30}$th be taken as the proportion of gas that shows a slight cap on the safety lamp flame, the quantity of gas that can be taken away per minute will be $\frac{6,000}{30}$ = 200 cubic feet, ∴ the time occupied in removing the whole will be $\frac{30,000}{200}$ = 150 minutes = $\frac{150}{60}$ = $2\frac{1}{2}$ hours.

Question 100.—An explosive mixture of air and gas at the highest explosive point passes along an airway 4 feet by 5 at a velocity of 500 feet per minute. What quantity of fresh air must be added so that you cannot detect it on the flame of a safety lamp?

Here the area of the airway is 4 × 5 = 20 feet × 500 = 10,000 feet of air and gas. When at the highest explosive point the mixture is 1 of gas to 8 or 9 of air, but taking it at one of gas to 9·5 of air, as proportioned by some, then

As 10·5 : 9·5 :: 10,000 : 9,047·6 cubic feet of air,
and as 10·5 : 1 :: 10,000 : 952·4 „ „ firedamp.

When the mixture is 31 of air to 1 of the gas its presence cannot be detected, so that 952·4 × 31 = 29,524 cubic feet as the quantity of air required, but as there are already 9,047 cubic feet of air it would take an additional quantity of 29,524 − 9,047 = 20,477 cubic feet per minute of fresh air to dilute the gas so that it could not be detected by means of the lamp.

Question 101.—How long would it take an air-current of 20,000 cubic feet per minute to make the circuit of an airway 4,000 yards long, 10 feet by 10 feet?

10 feet × 10 feet = 100 feet sectional area, and $\frac{20,000}{100}$ = 200 feet per minute as the velocity of the current. ∴ $\frac{4,000 \times 3}{200}$ = 60 minutes or 1 hour to make the circuit.

Question 102.—A current of 2,000 cubic feet a minute is at explosive point. How much fresh air must be mixed with it to prevent its showing a "cap"?

When the firedamp forms as much as one part out of thirteen of an air-and-gas mixture it becomes explosive.

The gas evolved, therefore, would be $\frac{1}{13}$ of 2,000 = 153·846 cubic feet per minute, and the quantity of air is 2,000 − 153·846 = 1,846·154. In a mixture of one part of firedamp to 30 of air the flame will not show a cap. 153·846 × 31 = 4,769 cubic feet per minute as the total ventilation, when the requisite change has taken place. Then the fresh air required to accomplish this will be 4,769 cubic feet, but as already 1,846 cubic feet of fresh air are passing, an additional quantity of 4,769 − 1,846 = 2,923 cubic feet per minute will be required.

Question 103.—The depth of downcast is 100 fathoms, with a temperature of 50° F., the depth of upcast is 150 fathoms, with a temperature of 100° F. What water-gauge should this show?

Here the state of the barometer is not given, but, assuming it to be 30 inches, the weight of a cubic foot in the downcast would be $\frac{1\cdot3253 \times 30}{459 + 50} = \cdot07811$ lb.; that of the upcast would be $\frac{1\cdot3253 \times 30}{459 + 100} = \cdot07112$ lb.; and the difference is $\cdot07811 - \cdot07112 = \cdot00699$. The depth of the upcast being 900 feet gives $\cdot00699 \times 900 = 6\cdot291$ lbs. as the pressure due to the difference in the mere weight of the two columns. The water-gauge would therefore read $\frac{6\cdot291}{5\cdot2} = 1\cdot21$ inch, and the motive column is as $\cdot07811 : 900 :: \cdot00699 : 80\cdot54$ feet.

> *Question* 104.—In a colliery with two shafts, one downcast and the other upcast, each 1,000 feet deep and 15 feet in diameter, temperature of downcast 60° F., average pressure in the shafts 30 inches. Quantity of air going down the downcast, 150,000 cubic feet per minute, the ventilation being produced by a furnace. Required the average temperature of the upcast, and total horse-power necessary to produce the ventilation, assuming that of the workings (total, minus shafts) to be 50 horse-power.

By using the formula $u = ks\left(\frac{q}{a}\right)^3$ to find the power necessary to circulate 150,000 cubic feet of air per minute through the two shafts, it gives $\cdot0000000217 \times (15 \times 3\cdot14159 \times 2{,}000) \times \left(\frac{150{,}000}{15^2 \times \cdot7854}\right)^3 = 1{,}250{,}792$ units of work. $\therefore \frac{1{,}250{,}792}{33{,}000} = 37\cdot9$ horse-power. But, as stated before, the value of k has not been fixed for shafts, and it is manifestly unfair to assume it will be the same for them as for the rough and uneven passages of a mine; probably the value for the smooth surfaces of the shafts will not exceed one-half that of the underground roads.

Assume the power used in the two shafts to be 20 horse-power instead of the above worked-out 37·9, then the total horse-power required for both shafts and mine will be $20 + 50 = 70$, and $70 \times 33{,}000 = 2{,}310{,}000$ foot pounds or units of work. That is, the furnace must be capable of performing that work per minute.

At a temperature of 60° F., and barometer at 30 inches, a cubic foot weighs $\frac{1\cdot3253 \times 30}{459 + 60} = \cdot0766$ lb., and as there are 150,000 cubic feet of air circulating, its weight at the stated temperature and pressure is $\cdot0766 \times 150{,}000 = 11{,}490$ lbs. Therefore, the duty of the furnace is to lift this weight of air through a height of $\frac{2{,}310{,}000}{11{,}490} = 201$ feet in one minute. Proceed to find the temperature that must be maintained in the upcast shaft to circulate the air through both the shafts and mine, by the formula

$$T = \frac{h\,(459 + t)}{l - h} + t$$

where h = the height through which the whole of the air which traverses the mine has to be lifted

t = the temperature of the downcast shaft

T = do. do. upcast do.

l = the length of the heated column of air in the upcast.

Substituting the known values in the above formula $T = \frac{201 \times (459 + 60)}{1{,}000 - 201} + 60 = 190°$.

A column of air 1,000 feet high, and at a temperature of 190° F., in the upcast shaft, would occupy a height represented by x when cooled down to 60° F. of $1{,}000 \times \frac{459 + 60}{459 + 190} = 799$; or $x\left(1 + \frac{190 - 60}{459 + 60}\right) = 1{,}000$, and so $x = 799$. Therefore $x = l - h$. The value of h is the height of a column of air which will balance the difference of weight between the two volumes of air in the downcast and upcast, usually called the motive column.

Question 105.—If 40,000 cubic feet of air enter a downcast shaft 14 feet diameter, and the temperature is 60° F., what must be the diameter of the upcast shaft to pass that quantity of air, the velocity being the same in each case, and the temperature of the upcast being 100° F.?

The sectional area of the downcast is $14^2 \times {\cdot}7854 = 153{\cdot}9384$ square feet. $\therefore \frac{40{,}000}{153{\cdot}9384} = 259{\cdot}8442$ feet per minute as the velocity of the current in the downcast.

The velocity in the upcast would be, therefore, $259{\cdot}8442 \times \frac{460 + 100}{460 + 60} = 279{\cdot}83$ when both shafts have the same diameter. Hence $\sqrt{\frac{279{\cdot}83 \times 153{\cdot}9384}{259{\cdot}8442 \times {\cdot}7854}} = 14{\cdot}528$ feet diameter of the upcast when the velocity of the current is the same in both shafts.

Question 106.—If 30,000 cubic feet of air entered a shaft 14 feet diameter, at a temperature of 60° F., what diameter of upcast is necessary to pass the same quantity, the velocity being the same in each case, and temperature of upcast 90° F.?

A volume of air increases $\frac{1}{460}$ of its bulk for each F. degree of increased temperature. The diameter varies as $\sqrt{q}$, or it may be found by $d = \sqrt{\frac{D^2 \times Q}{q}}$ where D = diameter of downcast, d = diameter of upcast, Q = quantity in the upcast, q = quantity in the downcast.

Taking the former formula, $\frac{30{,}000}{1} \times \frac{460 + 90}{460 + 60} = \frac{16{,}500{,}000}{520} = 31{,}730$ cubic feet to pass up the upcast shaft, and as $\sqrt{30{,}000} : \sqrt{31{,}730} :: 14 : 14{\cdot}398$ = diameter of upcast shaft. Or by the latter formula $14^2 = 196 \times 31{,}730 = 6{,}219{,}080 \div 30{,}000 = 207{\cdot}3$ and $\sqrt{207{\cdot}3} = 14{\cdot}398$, as before.

Question 107.—Supposing the existence of an airway passing 20,000 cubic feet of air per minute, what would be the increased quantity after putting in an additional airway of the same dimensions, the power remaining the same?

By the formula $u = ks\left(\frac{q}{a}\right)^3$, assuming any value for a, say 30 feet, and s, say 1,000 feet.

Then $u = {\cdot}0000000217 \times 1{,}000 \times \left(\frac{20{,}000}{30}\right)^3 = 6{,}429{\cdot}7$.

After putting in the additional airway, the units of work are to remain the same as before, but the value of a and of s will be doubled, therefore

$q = \sqrt[3]{\frac{6{,}429{\cdot}7}{{\cdot}0000000217 \times 2{,}000}} \times 60 = 31{,}748$, which is the number of cubic feet per minute circulating after putting an additional airway of the same dimensions as the original one.

Or $q = \sqrt[3]{\frac{u}{ks}} \times a$.

The value of u, s, and a, may be set down at 1 each when 20,000 cubic feet pass, and the altered conditions will then necessitate u being made 1, and s and a 2 each. The value of k may be omitted, as it is common to both airways, and will not affect the result. The quantity that would pass then is as $\sqrt[3]{\frac{1}{1}} \times 1 : \sqrt[3]{\frac{1}{2}} \times 2$ :: 20,000. $\therefore$ 1 : ·7937 × 2 :: 20,000, and 20,000 × ·7937 × 2 = 31,748.

> *Question* 108.—A steam jet and fan, both acting together in an upcast shaft, produce 50,000 cubic feet of air per minute; when the fan is stopped the jet gives 10,000 alone; what would be the result were the jet removed?

The power required to ventilate a given mine varies as the cube of the quantity; if therefore the power of the steam jet which gives 10,000 cubic feet is set down at 1, then the power of the two combined which produces 5 times the quantity will be in the ratio to the power of the steam jet as $5^3 : 1^3$ or as 125 : 1. Therefore the power of the fan is 124, that of the steam jet being 1. To find the quantity the fan alone will produce—if a power of 1 will produce a quantity of 10,000 what quantity will a power of 124 produce? or, if a power of 125 produce a quantity of 50,000 what will be produced by a power of 124? It is equally true that the quantity varies as the cube root of the power; therefore, As $\sqrt[3]{1} : \sqrt[3]{124}$ :: 10,000 : 49,866;
or, as $\sqrt[3]{125} : \sqrt[3]{124}$ :: 50,000 : 49,866;
that is, the fan alone would give 49,866 cubic feet per minute.

> *Question* 109.—There are two airways in a mine whose lengths are 2,000 feet and 3,000 feet respectively, subject to the same ventilating pressure. The shorter is 5 feet × 5, what size must the longer be made so that each may pass an equal quantity?

Here it is possible to proceed by the formula $R = \sqrt{\left(\frac{a^3}{s}\right)}$, but a simpler way of getting the same result is to remember that the side of the square airway will vary in the proportion of the 5th root of the lengths, when the roads are subject to the same pressure and are to pass the same quantities; thus, As $\sqrt[5]{2{,}000} : \sqrt[5]{3{,}000}$ :: 5; or,

As $\sqrt[5]{2} : \sqrt[5]{3}$ :: 5 : 5·42235 as the side of the longer airway and its area would therefore be $5{\cdot}42235^2 = 29{\cdot}4$.

Similarly for a circular airway the diameter varies in the same ratio. Thus if the airways were circular in form, the diameter of the shorter would be $\sqrt{\frac{25}{{\cdot}7854}} = 5{\cdot}6419 \therefore \sqrt[5]{2} : \sqrt[5]{3}$ :: 5·6419 : 6·1185 as the diameter of the 3,000-foot airway and its area therefore would be $6{\cdot}1185^2 \times {\cdot}7854 = 29{\cdot}402$, the same as that already worked out for the square airway.

So also for a rectangular airway the height and the width of the airway vary in the same ratio. Thus if the airway had been rectangular and 4 feet high, its width would be $\frac{25}{4} = 6{\cdot}25$ feet. For the 3,000-foot airway then As $\sqrt[5]{2} : \sqrt[5]{3}$ :: 4 : 4·3379 which is the height of the longer airway : $\sqrt[5]{2} : \sqrt[5]{3}$ :: 6·25 : or put it as 4 : 4·3379 :: 6·25 : 6·7779 as the width of the longer airway, and therefore the area would be 4·3379 × 6·7779 = 29·4 as before.

Airways passing equal quantities subject to the same pressure must have an equal ventilating power, whether their lengths be equal or not.

The answer to this question may be found by assuming any quantity, say, 2,500 cubit feet per minute, to be passing along each airway.

Then proceeding by the formula

$$u = ks \times \left(\frac{q}{a}\right)^3 \text{ for the short airway}$$

$$u = \cdot0000000217 \times 40{,}000 \times 100^3$$

$$u = 868$$

If a = the area of the long airway then $4\sqrt{a}$ or $\sqrt{16a}$ = its perimeter

Then since $\sqrt[3]{\frac{u}{ks}} \times a = q$

$$\sqrt[3]{\frac{868}{\cdot0000000217 \times 3{,}000 \times \sqrt{16a}}} \times a = 2{,}500$$

$$\frac{868a^3}{\cdot0000000217 \times 3{,}000 \times \sqrt{16a}} = 15{,}625{,}000{,}000$$

$$\frac{868a^3}{\sqrt{\cdot0000000678a}} = 15{,}625{,}000{,}000$$

$$\frac{753{\cdot}424a^6}{\cdot0000000678a} = 244{,}140{,}625{,}000{,}000{,}000{,}000$$

$$11{,}112{,}448{,}377{,}581a^5 = 244{,}140{,}625{,}000{,}000{,}000{,}000$$

$$a^5 = 21{,}970{,}102.$$

$$a = 29{\cdot}4$$ being the same result as obtained in the previous working.

Question 110.—There are three airways in a colliery
A, being 3,000 feet long and 6 feet × 5 feet = 30 feet area
B, „ 4,000 „ „ 6 feet × 6 feet = 36 do.
C, „ 2,000 „ „ 5 feet × 5 feet = 25 do.

The total quantity of air passing in the three airways is 50,000 cubic feet per minute. What is the quantity passing along each?

These airways must be subject to a common pressure, and the relative volume for each will be found by the formula $R = \sqrt{\left(\frac{a^3}{s}\right)}$ ∴

$$\text{For A,} \quad R = \sqrt{\frac{30^3}{66{,}000}} = \cdot6396$$

$$\text{„ B,} \quad R = \sqrt{\frac{36^3}{96{,}000}} = \cdot69714$$

$$\text{„ C,} \quad R = \sqrt{\frac{25^3}{40{,}000}} = \cdot625$$

Total relative volumes 1·96174

Therefore the actual volumes going into each must be
For A As 1·96174 : ·6396 :: 50,000 : 16,302 cubic feet
„ B „ 1·96174 : ·69714 :: 50,000 : 17,768 do.
„ C „ 1·96174 : ·625 :: 50,000 : 15,930 do.

Total 50,000 do.

Question 111.—Compare the friction in the following roads:—No. 1 is 8 feet by 5; No. 2 is 6 feet by 12; (*a*) for the same quantity and (*b*) for three times the quantity of air.

First, by the formula $R = \sqrt{\left(\frac{a^3}{s}\right)}$, or as R must be the same in each case, omit the sign of the square root, and the result is not affected. Therefore, there are, as the relative resistances in the two roads for the same quantity of air, As $\frac{40^3}{26} : \frac{72^3}{36}$, or as 2,461·5 : 10,368 :: 1 : 4·212; that is, the resistance offered by No. 1 airway will be 4·212 times as great as that offered by No. 2 airway for the same quantity of air.

If 3 times the quantity of air is to circulate through No. 2 as compared with No. 1, then proceed by the formula $u = ks\left(\frac{q}{a}\right)^3$ or ksv^3 to find the relative resistances.

The value of k will be the same in each case, and being a common factor, may be omitted, and the relative resistances then become sv^3. There being no definite quantity of air assigned as passing through the airways, for convenience make the quantity passing through No. 1 a multiple of its area, say 40, then the quantity passing through No. 2 will be $40 \times 3 = 120$. The relative velocities must then be $\frac{40}{40} = 1$ and $\frac{120}{72} = 1{\cdot}\dot{6}$. The relative pressures will be

For No. 1 airway $26 \times 1^3 = 26$ (substituting o for s).
" " 2 " $36 \times 1{\cdot}\dot{6}^3 = 166{\cdot}\dot{6}$ do.

Therefore $\frac{166{\cdot}\dot{6}}{26} = 6{\cdot}41$ times as much friction in No. 2 as in No. 1 for 3 times the quantity of air.

Question 112.—Given 10,000 cubic feet of air per minute through an airway 10 feet by 8, with a pressure of 10 lbs. per square foot, what is the length of the airway?

By the formula $s = \frac{pa}{kv^2}$, $s = \frac{10 \times 80}{{\cdot}0000000217 \times 125^2} = 2{,}359{,}446$ square feet of rubbing surface, and since the perimeter is 36, the length of the airway is $\frac{2{,}359{,}446}{36} = 65{,}540$ feet.

Question 113.—What precautions should be adopted where candles and safety lamps are used in different parts of the same mine?

The practice of having "mixed lights" in a mine is one to be condemned, but if some extraordinary event made it necessary for a temporary purpose, the points to be observed are to see that no return air after having passed through the workings lighted by safety lamps should be allowed to pass through the workings lighted by candles or open lights. Each district should be ventilated by means of a separate intake and return airway. Proper precautions should be taken to prevent persons using candles from entering that part of the mine lighted by safety lamps.

By General Rule 8 of the Coal Mines Act, 1887—"When it is necessary to work coal in any part of a ventilating district with safety lamps, it shall not be allowable to work the coal with naked lights in another part of the same ventilating district situated between the place where such lamps are being used and the return airway."

Question 114.—If 1 cubic foot of gas should explode, how many cubic feet of flame would it make?

The law of the expansion of gases is that every 460 volumes of gas measured at 0° F., and at a pressure of 30 inches, expand to 461 volumes for each degree of temperature on Fahrenheit's scale the gas is raised.

The proportion of oxygen to nitrogen in the air is about $\frac{1}{5}$th. As oxygen combines with twice its volume of hydrogen, $\frac{1}{2}$ a cubic foot of oxygen would be required to unite with a cubic foot of hydrogen, necessitating about $2\frac{1}{2}$ cubic feet of air. There would then be a mixture of $3\frac{1}{2}$ cubic feet. Taking the heat of the explosive mixture of gas and air to be equal to that of 2,348° F., every 460 volumes become by explosion equal to $2{,}348 + 460 = 2{,}808$. $\therefore$ as $460 : 2{,}808 :: 3\frac{1}{2} : 21{\cdot}3$ cubic feet; that is, 1 cubic foot of gas becomes 21·3 cubic feet of flame.

Question 115.—At what velocities do the Davy (without shield), Stephenson and ordinary Clanny lamps become unsafe?

The ordinary unprotected Davy explodes in an inflammable mixture of gas and air when the current moves at a velocity of 6 feet per second, the Stephenson at from 9 to 12 and ordinary Clanny lamps at from 6 to 10 feet per second. If these lamps are to be used it must be in currents considerably below their exploding velocities.

Question 116.—Supposing a 40 horse-power fan gives 120,000 cubic feet per minute, what quantity of air will a fan of 32 horse-power give?

As $\sqrt[3]{40} : \sqrt[3]{32} :: 120{,}000 : 111{,}398$ cubic feet of air per minute or $\sqrt[3]{\left(\frac{32}{40}\right)} \times 120{,}000 = 111{,}398$ cubic feet.

Question 117.—If 10,000 cubic feet of air pass per minute along an aircourse 7 feet square, how much will pass through the aircourse if it be reduced to 4 feet by 3, the power remaining the same?

Here the formula $q = \sqrt[3]{\frac{u}{ks}} \times a$ may be used, but as no length of airway is stated, substitute o for s, and the relative values will not be affected if k also, which is common to both airways, be omitted. Then $q = \sqrt[3]{\frac{u}{o}} \times a$. No actual value is assigned to u, so for convenience make it a multiple of o, say 28, and the proportion then is

$$\text{As } \sqrt[3]{\frac{28}{28}} \times 49 : \sqrt[3]{\frac{28}{14}} \times 12 :: 10{,}000, \text{ or}$$

$$\text{As } 49 : \sqrt[3]{2} \times 12 :: 10{,}000 = \frac{10{,}000 \times 1{\cdot}25992 \times 12}{49} = 3{,}085{\cdot}5 \text{ cubic feet}$$

per minute for the smaller aircourse.

Question 118.—Supposing the workings of a colliery to be ventilated by a Guibal fan $21\frac{1}{4}$ feet in diameter, going at 40 revolutions per minute, how many cubic feet would the fan produce? The air is to pass through 4 separate airways, 2 of them being 6 feet by 5 feet and 1,000 fathoms in length, and 2 are 8 feet by 5 feet and 1,200 fathoms in length: what would be the height of the water-gauge in each airway, and also in the main airway at the bottom of the downcast pit, which is 12 feet × 6 feet, and 60 fathoms in length?

The theoretical water-gauge (close to the fan) that a Guibal fan 21¼ feet in diameter, going at 40 revolutions per minute, would produce, is ·902 inch (the height of barometer and thermometer slightly affects the water-gauge).

Thus, by the formula $h = \frac{v^2}{64{\cdot}4}$, where v = speed of the tips of vanes in feet per second, and h = the height of column necessary to create such velocity in a falling body,

$$\therefore h = \frac{\left(21{\cdot}25 \times 3{\cdot}1416 \times \frac{40}{60}\right)^2}{64{\cdot}4}$$

$h = 30{\cdot}7$, and therefore the water-gauge would be $\frac{30{\cdot}7 \times 2}{68} = {\cdot}902$ inch.

The depth and size of the shafts are not stated, and even if they were it could only be told approximately what proportion of the ventilating pressure will be spent in forcing the air through them. Assume it to be ¼th of the total pressure; for overcoming the friction due to the shafts, then ·902 — ·226 = ·676 as the water-gauge due to the underground workings between the two shaft bottoms; and as the airways must all leave the main intake at 60 fathoms from the downcast, and all come into the return at the bottom of the upcast, the water-gauge due to each of the 4 split airways will be the same.

The relative pressures for overcoming the friction are as follow:—

	Area in feet.	Rubbing surface in square feet.
Main Intake	72	12,960
No. 1 airway	30	132,000
" 2 "	30	132,000
" 3 "	40	187,200
" 4 "	40	187,200

Assuming now that any quantity, say 50,000 cubic feet, is passed along the main airway, the relative quantities going into the four airways, which must get the total among them will be found by the formula $R = \sqrt{\left(\frac{a^3}{s}\right)}$, thus,—

$$\text{For No. 1 airway } R = \sqrt{\frac{30^3}{132{,}000}} = {\cdot}45226$$

$$\text{" " 2 " "} = \sqrt{\frac{30^3}{132{,}000}} = {\cdot}45226$$

$$\text{" " 3 " "} = \sqrt{\frac{40^3}{187{,}200}} = {\cdot}5847$$

$$\text{" " 4 " "} = \sqrt{\frac{40^3}{187{,}200}} = {\cdot}5847$$

$$\text{Total } 2{\cdot}07392$$

For a total of 50,000 cubic feet circulating, the proportion of volumes going into each airway must be,

For No. 1 As 2·07392 : ·45226 : : 50,000 : 10,904
" 2 " 2·07392 : ·45226 : : 50,000 : 10,904
" 3 " 2·07392 : ·5847 : : 50,000 : 14,096
" 4 " 2·07392 : ·5847 : : 50,000 : 14,096

By the formula $p = \frac{ksv^2}{a}$ now proceed to find the relative pressures for the main intake and any one of the four airways due to the above quantities, since p is the same in all four,

then for the main intake $p = \frac{\cdot0000000217 \times 12{,}960 \times 694\cdot4^2}{72} = 1\cdot8837$ lb.

„ No. 1 airway $p = \frac{\cdot0000000217 \times 132{,}000 \times 363\cdot5^2}{30} = 12\cdot614$ lbs.

so that the total pressure for the quantity assumed to pass would be $12\cdot614 + 1\cdot8837 = 14\cdot4977$ lbs. per square foot.

But there is actually only a water-gauge of ·676 inch or $\cdot676 \times 5\cdot2 = 3\cdot5152$ lbs. per square foot pressure due to the underground workings between the shaft bottoms. The proportion of the pressures, however, will hold good between the main intake and the airways so that the actual pressures will be,

For the 4 airways As 14·4977 : 12·614 : : 3·5152 : 3·0584 lbs.
„ main intake As 14·4977 : 1·8837 : : 3·5152 : ·4568 lb.

and these pressures as indicated by the water-gauge would be $\frac{3\cdot0584}{5\cdot2} = \cdot588$ inch and $\frac{\cdot4568}{5\cdot2} = \cdot088$ inch, that is the water-gauge due to friction in the main airway is ·088 inch, and that due to each of the four split airways is the same, viz., ·588 inch.

Now proceed to find the actual quantity of air circulating in the mine.

By the formula $v = \sqrt{\frac{pa}{ks}}$ and substituting values for the main airway $v = \sqrt{\frac{\cdot4568 \times 72}{\cdot0000000217 \times 12{,}960}} = 341\cdot9$ feet per minute, therefore the quantity actually circulating is $341\cdot9 \times 72 = 24{,}616$ cubic feet per minute.

Or say, as the water-gauge to produce 50,000 cubic feet per minute would be $\frac{14\cdot4977}{5\cdot2} = 2\cdot788$ inches, the quantity that would circulate with a water-gauge of ·676 inch is, As $\sqrt{2\cdot788}$: $\sqrt{\cdot676}$: : 50,000 : 24,620 cubic feet, which is almost the same result as before, the difference being due to loss in dropping decimals.

Question 119.—What are the chief points to bear in mind in the adoption of a fan which is to supersede a furnace that produces 30,000 cubic feet of air per minute with a water-gauge at the pit bottom of ·3 inch? Give the difference in water-gauge when the fan produces 60,000 cubic feet of air per minute, the circumstances of the mine remaining unaltered.

The chief points to be borne in mind in choosing a fan to supersede a furnace are that a certain water-gauge, whether caused by a fan or by any other machine, only circulates a certain quantity of air through the mine.

That at a given speed a fan can never get more than a certain constant water-gauge, but that if the fan be not properly proportioned, or if it be too small for the airways, the air will be throttled at the fan, which will then get less than the water-gauge due to its speed. In choosing a fan one should be decided on that will do the work required with as small a consumption of fuel as possible, and for that reason it should be driven by a compound engine if a new engine has to be purchased for the purpose. In answer to the latter part of the question, if the quantity of air be doubled the water-gauge is increased $2^2 = 4$ times, and it would therefore read $\cdot3 \times 4 = 1\cdot2$ inch.

Question 120.—If 20,000 cubic feet of air are produced in an airway 500 fathoms in length, 8 feet in breadth, and 4 feet in height, how many feet would be produced if the air was split into three divisions, the first airway being as the above, the second 600 fathoms in length, 9 feet in breadth, and 5 feet in height, the third 800 fathoms in length, 10 feet in breadth, and 6 feet in height, the power being the same?

Use the formula $u = ks\left(\frac{q}{a}\right)^3$ or ksv^3 and for the airway passing 20,000 cubic feet at a velocity of $\frac{20,000}{32} = 625$ feet a minute $u = \cdot 0000000217 \times 72,000 \times 625^3 = 381,444 \cdot 7$ units, or $\frac{381,444 \cdot 7}{33,000} = 11 \cdot 5589$ horse-power.

Then, for the power to remain the same for the 3 airways,

	Area in feet.	Rubbing surface in square feet.
For the 1st 8 feet × 4 feet × 3,000 feet long	32	72,000
,, 2nd 9 feet × 5 feet × 3,600 ,,	45	100,800
,, 3rd 10 feet × 6 feet × 4,800 ,,	60	153,600
Total	137	326,400

Then by the formula $q = \sqrt[3]{\frac{u}{ks}} \times a$; $\sqrt[3]{\frac{381,444 \cdot 7}{\cdot 0000000217 \times 326,400}} \times 137 = 51,736$ cubic feet as the quantity which would circulate with the same power.

Question 121.—If in a heading 7 feet 6 inches by 6 feet 8 inches the air travels 40 yards in 12 seconds, what would be the quantity of air passing per minute? If the water-gauge was 2·5 inches, what would be the horse-power?

Here $7\frac{1}{2} \times 6\frac{2}{3} = \frac{15}{2} \times \frac{20}{3} = \frac{300}{6} = 50$ feet area of airway. 40 yards = 120 feet, and if the air travels that distance in 12 seconds, its velocity must be as 12:60::120:600 feet per minute, and the quantity passing is 600 × 50 = 30,000 cubic feet per minute. With a 2·5-inch water-gauge the horse-power of ventilation is $\frac{30,000 \times 5 \cdot 2 \times 2 \cdot 5}{33,000} = 11 \cdot 82$.

Question 122.—If with a ventilating fan running at 65 revolutions per minute 1·02 inch of water-gauge is produced, what will be the water-gauge if the fan speed be increased to 96 revolutions per minute?

The water-gauge varies as the square of the quantity and also, since the quantity is proportional to the speed of the fan, as the square of the revolutions. $\therefore \left(\frac{96}{65}\right)^2 \times 1 \cdot 02 = 2 \cdot 22$ inches.

Question 123.—If with a ventilating fan running at $93\frac{1}{2}$ revolutions per minute 1·3 inch of water-gauge is produced, what will be the water-gauge if the fan speed be altered to 82 revolutions per minute?

As the water-gauge varies as the square of the revolutions, then as $93 \cdot 5^2 : 82^2 :: 1 \cdot 3$ or $\left(\frac{82}{93 \cdot 5}\right)^2 \times 1 \cdot 3 = 1$ inch of water-gauge.

Question 124.—If a ventilating fan is running at 40 revolutions per minute with 1·5 inch of water-gauge, and it be altered so that the water-gauge reads 2·6 inches, what will be the fan speed?

The quantity of air passing varies as the square root of the water-gauge, and as the quantity is in direct proportion to the fan speed, the latter also varies as the square root of the water-gauge $\therefore \sqrt{\frac{2{\cdot}6}{1{\cdot}5}} \times 40 = 52{\cdot}67$ revolutions.

Question 125.—If a ventilating fan is running at 80 revolutions per minute with 3·75 inches of water-gauge, and the speed be altered so that the water-gauge reads 1·82 inch, what will be the fan speed?

Here, As $\sqrt{3{\cdot}75} : \sqrt{1{\cdot}82} :: 80$ or $\sqrt{\frac{1{\cdot}82}{3{\cdot}75}} \times 80 = 55{\cdot}73$ revolutions.

Question 126.—In an explosion of gas at 70° F., what would be the difference of expansion in volume, the combustion taking place at 9,564°?

A gas expands $\frac{1}{460}$th of its volume at 0° F. for each degree it is raised above that point under a constant pressure.

Therefore any 460 volumes at 0° become 460 + 70 = 530 volumes at 70°, and at 9,564° the volume would be 460 + 9,564 = 10,024. In other words, the relative volumes occupied by a gas at the respective temperatures of 70° F. and 9,564° F. will be represented by the figures 530 and 10,024, so that the difference of expansion in volume is as 530 : 10,024 or as 1 : 18·9, that is, every cubic foot of the explosive mixture at 70° becomes 18·9 cubic feet at 9,564°.

Question 127.—How would you ventilate a mine giving off CH_4 and CO_2 freely, and what kind of airways would you adopt, and what proportion should they be to one another?

I should make no distinction in ventilating a mine giving off CH_4 or light carburetted hydrogen gas at one portion and CO_2 or carbonic acid gas at another, except not allowing the air from the one portion to return so as to mix with the other. By the first general Rule of the Mines Act, 1887, we are bound to provide an adequate amount of ventilation in every mine to dilute the noxious gases so as to render them harmless. Both CH_4 and CO_2 are noxious gases, and the means of diluting them the same, viz., by providing and coursing round the districts of the mine such quantities of pure air as to ensure the rendering harmless of these gases. Air containing 3 or 4 per cent. of CO_2 is unfit to be breathed, and therefore we must be sure that a district giving off that gas has air in the proportion of 100 to 3 of the quantity of CO_2 given off, or about 33 to 1, and as also CH_4, when mixed with the air in the proportion of 1 in 30, will show a "cap" on the flame of a lamp, we shall require 30 parts of air in any district giving off CH_4 to every 1 part of CH_4 so given off. I do not mean to say that it is sufficient merely to dilute these gases so that the one may be just in a breathable state and the other be just beyond the point of its showing a cap; the figures are given merely to show that the relative quantity for this purpose is nearly the same. Therefore the proportion of air required in the different districts of a mine, taken in consideration simply of these gases, will be practically the same, but other considerations, such as the number of workmen employed in each, &c., may affect the relative quantities we should send into each. I would, then, ventilate any mine by a judicious arrangement of splits, and the kind of airways would be governed to some extent by the thickness of the seam worked. Theoretically, the most effectual shape of airway is the circular: the circular, however, is not practicable in our underground roadways, but a square one, which is the next best form, frequently is, and where practicable, I should adopt it, and where impracticable I should be careful to make the airways of a large sectional area, and if limited as to height, of a width to ensure the sectional area being sufficient, but I should prefer the square form. The proportions of airways for districts giving off CH_4 and CO_2, as already shown, apart from other considerations, may be the same.

CHAPTER XIII.

THE PRIESTMAN OIL ENGINE: PETROLEUM AND NATURAL GAS.

Application of the Oil Engine to Mining—Description of its Action—Cost of Working—Its Advantages in certain Positions—Rules for the Prevention of Accident from its Use—Particulars of its Application as a Hauling Engine—Different Nature of Work Performed by Oil Engines—Quality of the Oil Used—Character of Petroleum—Geological Formations in which Found—Possibility of a Boring first Tapping Petroleum, Water, or Gas—A Theoretical Mode of Production—Professor Mendeleeff's Theory of Petroleum Formation—Comparison of Manufactured and Natural Petroleum—Possibility of the Exhaustion of Coalfields and Continuance of Oil-fields—Chemical Composition of Petroleum—Natural Gas in Commercially Profitable Quantities—Where Found—Particulars of the Findlay Gas Well—Increase of Capital Employed in the Use of Natural Gas—Shrinkage in its Supply—Burning the Gas on the Surface of River Water and on the Ground—Analysis of Pittsburg Natural Gas—Its Occurrence in the United Kingdom—Petroleum in Europe with the Number of Wells bored and their Depth in the Baku Oil-field—Known Oil Regions of the United States, Canada, and Mexico—Number of Wells Bored and their Average Depth in America—Oil-fields of South America, Australia, New Zealand, North Africa, South Africa, Persia, Burmah and India—Petroleum in China, Sumatra, Java, Borneo, and Japan.

THE PRIESTMAN OIL ENGINE.

During the past few years the use of mineral oil as a source of power has increased considerably, and oil engines are rapidly gaining ground among colliery proprietors for pumping, haulage, winding and other purposes.

Messrs. Priestman Bros., Ltd., of Hull, may be said to have been the first to introduce an engine which works satisfactorily.

Fig. 423D is a drawing of this engine. The oil is placed in the tank, Y; it is then put under a pressure of air by working the hand pump lever, D, after which the stopcock is turned so that the air and oil pass along the copper pipes to the heating lamps, E, E. One, two or more heating lamps are used according to the size of the engine. A three-way cock, the lever for which is at A, is fixed so that the oil and air can be turned either to the heaters or through small copper pipes to the spray-maker. As the oil will not ignite in the working cylinder, Z, unless the temperature is raised, the vapouriser, O, is heated for a few minutes by a flame from the heaters, E, E, which are kept alight by working the hand pump lever, D, or by other means which can be employed in order to save the manual labour of pumping. When the vapouriser is sufficiently heated, the stopcock is turned by means of the cock lever, A, so that the oil and air are shut off from the heaters, E, E, and admitted through pipes to the spray-maker, and so into the central passage of the vapouriser. The spray-maker and regulator are of special construction. The air and oil come into contact in a nozzle, both being forced through by the air-pressure in the oil tank and pass into the vapouriser in the form of spray. A further supply of air is drawn through small holes in the cover of the vapouriser. Each charge of oil mixed with air is correctly regulated by means of the governor in proportion to the amount of work to be done, and the valves are so carefully adjusted as to ensure constant proportions of air and oil and great regularity in the running. During the working of the engine the air pressure is maintained in the oil-tank by means of an air-pump worked by an eccentric from the

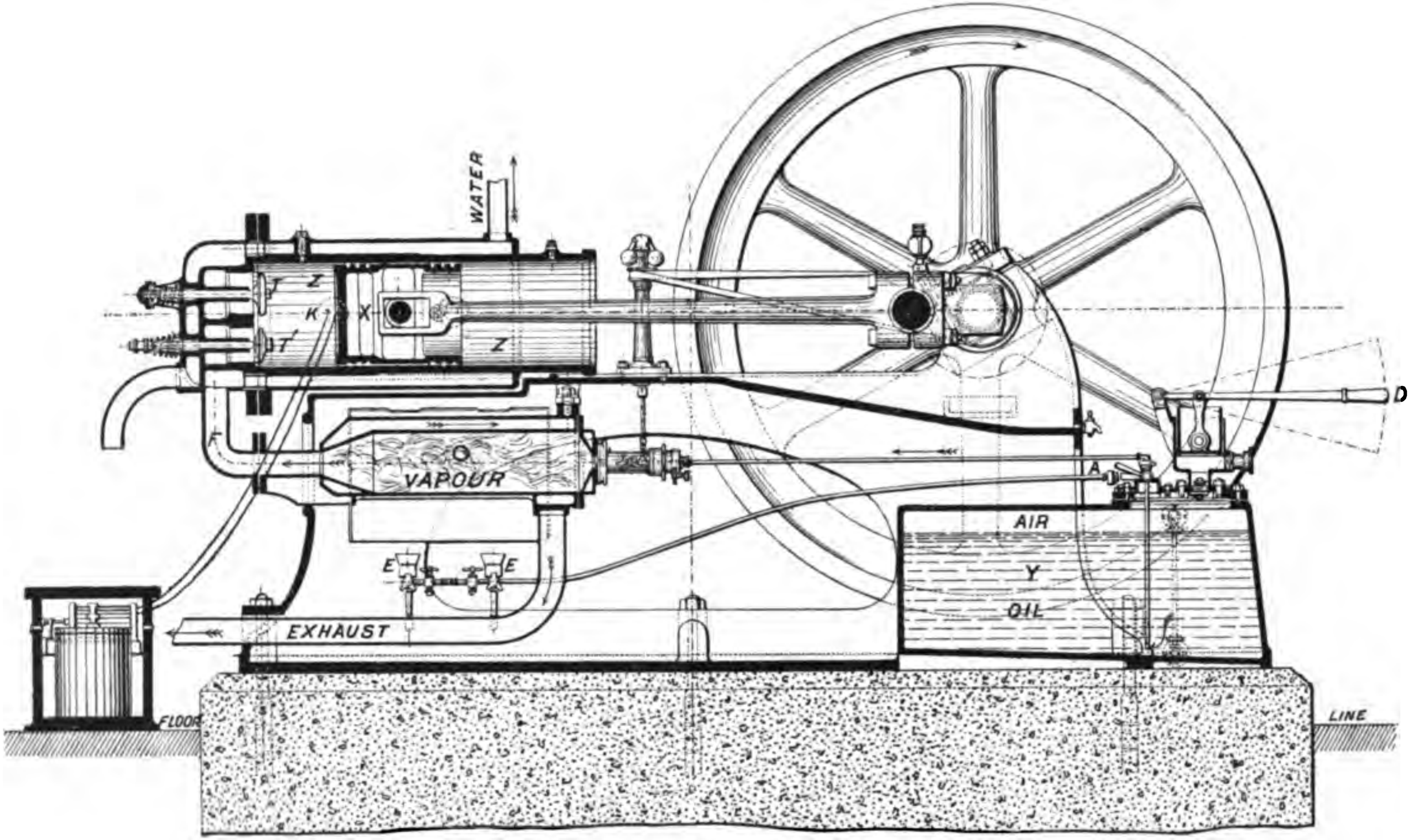

Fig. 423D.—Sectional Elevation of the Priestman Horizontal Oil Engine.

counter-shaft of the engine. Sprayed oil is thus admitted into the vapouriser and is immediately warmed and vapourised, so that when the piston, X, is drawn out by taking a turn of the fly-wheel, this warmed vapour, with air, is sucked through the passage, F, and top valve, T, into the cylinder, Z. The fly-wheel being further turned forces the piston back, which constitutes the second stroke, thus compressing the vapour in the chamber of the cylinder; it is then ignited by means of the spark at K, which immediately expands the charge and drives the piston forward and sets the fly-wheel in motion. The velocity of the fly-wheel forces back the piston, which drives out the spent vapour through the lower valve T′, which is opened by a rod operated by the counter shaft. The vapour which is forced out, being hot, passes down one of the pipes at F, round the outer passage of the vapouriser, O, and into the atmosphere, and so the vapouriser is kept sufficiently hot, by the exhaust gases in their passage through the jacket, to warm the new spray coming from the tank, and the engine therefore works automatically. The next stroke of the piston draws in a charge as before described, when it is again compressed, and after ignition gives a further impulse to the piston. The cylinder is water-jacketed, and water can be conveyed through the jacket either by gravitation or by means of a small pump worked from the countershaft of the engine.

The ignition is caused by electricity; an ordinary bichromate cell is used with induction coil. On one side of the cylinder, at K, there is a screw plug which carries two porcelain insulators, through which the terminal wires pass, and on the other side they are connected to a brass fork or spring, the circuit being completed at the proper moment by the brass ball or contact finger fixed on the eccentric rod passing through the fork.

The engine has only one motion of the crank-shaft, so that when it has been necessary to work it the reverse way, separate gearing has been applied for the purpose and is found to answer remarkably well.

The price of oil in various seaport towns in Great Britain is about 4*d.* per gallon. The oil used by the engine is rather less than 1 pint per brake H.P. for the large engines, and slightly exceeds this for the small engines, so that the cost per brake H.P. per hour is about one halfpenny.

The oil engine has been successfully applied to underground pumping, to hauling in mines, for driving ventilating fans, and for drilling. Being a prime motor and having the advantages of a gas engine without being dependent on a supply of gas, which is not available in such positions as the distant point of an underground roadway, there are many classes of work requiring motors of moderate size to which the engine can be well applied. It has great advantages, under favourable conditions, over other motive powers placed at long distances from the shaft in the underground workings, and may be easily placed in position or readily removed when necessary.

Many varieties of oils are suitable for these engines, but as the oils usually to be obtained vary much in their flashing temperature, it is best to make a careful selection of that intended for use, which, unless a well-known brand, should be tested before using.

The safe use of oil in the engines placed underground is of such importance that rules have been drawn up by the Board of Trade to ensure careful handling of the oils underground.

The flashing point of an oil is defined as being that temperature at which vapour is given off, sufficient to ignite and burn with a blue flash when brought into contact with a small flame.

A further raising of the temperature causes the oil to reach burning point, when the oil itself becomes ignited.

The temperature at which the oil shall flash has been settled at 100° Fahr. for the open test, but as the Scotch shale oils of high flashing points are admirably suited for these engines, no difficulty need be experienced in complying with the rules laid down by the Board of Trade. The oil used in the Priestman engine is heavy and has a high flashing temperature. It does not require the attention of specially skilled men, no expense is necessary for boiler or in the erection of chimney, the fuel is cleanly to handle, and the engine can be started in a few minutes.

It is desirable in underground workings that a good ventilation should be maintained, and that where it is possible the exhaust gases from the engine should be conveyed direct to the main return airway. The gases will then quickly mix with the return air and be reduced in temperature; any vapour passing into the airway from incomplete combustion in the engine will thus be rendered inoffensive and the smell of petroleum be carried away by a good current of air before it becomes objectionable. The products of combustion vary slightly when the engine is working under varying loads but are inexplosive. Under ordinary working conditions there is no risk of petroleum passing unconsumed into the exhaust.

A large number of these engines have already been put to work in underground workings in England and Scotland, New South Wales, and Victoria. In a mine in South Wales, the engine was coupled up direct to a set of three-throw pumps 6 inches × 12 inches, and the water pumped a distance of about 300 yards.

A set of pumps in the county of Durham, which were originally worked on the tail-rope haulage system, were replaced by a Priestman oil engine and a double-acting pump. These were placed 2,400 yards from the shaft, and at a point 165 vertical feet in the dip. The engine is of 5 nominal H.P. and drives the pump by a belt. The pump has a 6-inch barrel and a stroke of 18 inches. The water is forced a distance of 1,320 yards a total height of 72 feet.

Mr. W. H. Wain, speaking before the North Staffordshire Institute of Mining Engineers on the 21st March, 1892, said:—

"At the Midland Coal Co.'s colliery some time ago, having no available power within a thousand yards of a point where it was desired to drive down a pair of exploring places, it was decided to have one of the Priestman oil engines. The work done was the haulage of the coal from the bottom of the dip, and water from the different points at which it was caught. The distance hauled was 491 yards, the weight being about 37 cwts., and the average time 5 minutes 55 seconds. The total cost per hour for stores and wages was *2s.* 1·58*d.*"

The oil engine, slightly altered, has been largely employed for rock drilling in ironstone mines in the Cleveland district, this having been found to effect a great saving, both in first cost and in maintenance, when compared with hand labour, compressed air, or electricity.

From the fact that the British and Norwegian Governments have placed repetition orders for this engine for printing and fog-signalling, that it is in use in several of the Lighthouses of Great Britain, and is adopted by the Chinese and United States Governments for the same purpose, and further that the Governments of Victoria, New South Wales, New Zealand, and Spain, are using it for electric lighting, pumping, sawing, &c., it may fairly be assumed that it is perfectly satisfactory.

PETROLEUM AND NATURAL GAS.

Petroleum, or rock-oil, has been found in many parts of the world and probably exists in many others. In character it resembles both naphtha, which is more fluid, and asphaltum, which is solid. None of them are soluble in water or in alcohol, but the more dense can be dissolved in the more fluid of the allied substances. The term petroleum or rock-oil is understood to include the hydrocarbon products yielded by the distillation of bituminous shales, Boghead coal, and peat, besides the mineral oils which flow or are raised from the crust of the earth. Petroleum has been found in many if not all the geological formations from the Silurian upwards. In the United States it occurs for the most part in sandstones at the upper extremity of the Devonian epoch ; in Canada in the Silurian limestone. It exists in soft or other porous rock which may be continuous over a wide area or be contained in cavities in the regular formation of rocks. The cavities were probably formed by disturbances of the strata. The more violent the disturbances, the larger will be the cavities, which, becoming filled with loose material, form receptacles in which the petroleum collects. Petroleum is found in the greatest abundance where the strata have not been disturbed to any extent, and where receptive beds of porous sandstone or limestone, at considerable depth below the surface of the earth, favour an accumulation over extensive areas, while at the same time the cover is of a compact, unfissured nature, or nearly so, forming an impassable upward barrier to the gas and oil. Sometimes the oil rises spontaneously to the surface when pierced by a bore-hole ; at other times it requires pumping. Water and gas are frequently, but not always found with the oil, and the following sketch shows how either may be tapped by a bore-hole. Fig. 424 shows a fissure in a stratified rock which is filled with water, oil, and gas, arranged in the order of their specific gravities, the water at the bottom, the oil above, and the gas at the top. A bore-hole on reaching the cavity would tap one of these in accordance with its position at A, B, or C. If water rise in the bore-hole, the pressure of the oil and gas may cause it to issue in a jet to a considerable height above the surface. After a time, if the pressure remains sufficient, the water will be all ejected from the cavity, and oil will appear in the bore-hole. The diminished pressure of the gas may now be unable to force the oil to the surface, and so pumping is resorted to. If the bore-hole had reached that portion of the cavity containing oil, the fluid might have reached the surface unaided, owing to the pressure of the gas. The oil on being struck usually rises to the surface and overflows around the boring site, but occasionally it issues

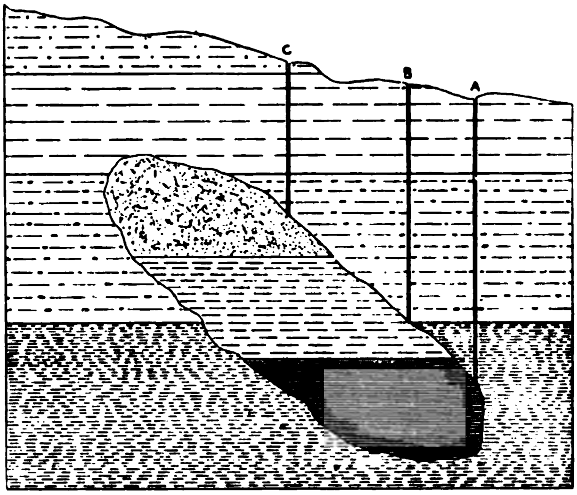

Fig. 424.—Fissure in Stratified Rocks, containing Water, Oil, and Gas.

with such force as at first to reach a height of 40 or 50 feet above the surface, this being diminished as the force grows less. In a discharge of this kind the yield of the well is large for a time, during which its commercial prosperity is at its greatest. After a short time the ejection ceases, but the oil may afterwards be pumped from the well and yield a steady supply for many years. If the gas is first tapped by the boring it is allowed time to escape, after which the oil can only be got by pumping. If, however, the gas is present in a considerable quantity, it is not allowed to escape, but is piped away for use.

In some instances the gas has been securely retained and conveyed a distance of a quarter of a mile in pipes, and has then been utilized to drive a steam engine just as though it were compressed air or high pressure steam produced on the spot.

The American oilfields often occur where the strata are very flat over extensive areas, and where the absence of faults favours the circulation of imprisoned fluids. Where there are great flat anticlines and synclines it is easy to understand that the wells drilled along the crests of the former yield gas, those in the troughs only water, while a zone between them may be expected to yield oil. The effect of boring into the different zones formed by flexures in the strata is similar to that described for borings in a fissure, but occurs on a larger scale. It is thought that although the oil is confined to particular strata, the beds are not uniformly porous, and a well may strike a close-grained rock through which no fluid can find its way.

Bore-holes drilled on an oil-belt whose direction is known vary in production according as the strata are more or less disturbed. A productive oil-well may exist close alongside one yielding comparatively little.

The crude petroleum is useful for some purposes, but it must be distilled in order to convert it into other commercial products; these again often requiring purification.

From the first distillation crude naphtha, and burning and paraffin oils are obtained, but sometimes the crude naphtha is got rid of before distillation begins. Further distillation causes the crude naphtha to yield gasolene, benzine, and refined naphtha. The burning oil usually requires treatment and purification before it is fit for use. The presence of naphtha, even in small proportions, lowers the flashing temperature of the burning oil, and, in larger proportions, renders it still more dangerous in use.

It has been assumed that petroleum is of the nature of coal, and has been formed out of the turpentine oils and resins of the coniferous trees of primeval forests which entered into the composition of coal seams, or out of the remains of marine animals, and that in course of time the deposits will be exhausted. No satisfactory theory has been suggested to account for the decomposition, at great depths beneath the surface, of vegetable and animal remains, and no explanation as to whether or not the process is still going on.

Professor Mendeleeff, a distinguished Russian chemist, has propounded the theory that petroleum is constantly being formed by the action of water on metallic deposits in the heated interior of the earth. Professor Mendeleeff states, that the oil-bearing regions generally lie parallel to mountain ranges, such as the Caucasus, in Russia, whence the greatest European supply comes, the Alleghanies, in the United States and Canada, and the Andes in Peru, and that petroleum does not appear to belong to any particular geological formation. In Europe it usually occurs in rocks of the Tertiary formation, while in the United States it is found in the Devonian and Silurian periods. He also shows that, on account of the volatile nature of rock-oil, it could not have been transferred from a distance, like many other deposits, but must have been formed very near the spot where it is found. The formation of mountain chains and terrestrial disturbances have fissured the earth's crust, making openings through which surface water penetrates to the interior; there, coming in contact with the glowing metals and their carbides, chemical reactions have been set up, resulting in hydrogen taking up the carbon to form petroleum in a state of vapour, and then of steam. As these vapours are given off they flow through fissures into the upper and cooler regions, upon reaching which they are either wholly or partially condensed, forming deposits of petroleum very commonly associated with water; the gases which resist condensation by cold escape to the surface. The precise compounds which are formed depend upon the temperature and pressure, and are afterwards found as gas, oil, asphalte or mineral pitch, ozokerit or earthwax, &c. The persistence of the oil wells in an average yield points to the probability of its formation almost as fast as it is removed. Professor Mendeleeff's theory is supported by the fact, which indeed suggested it, that petroleum which cannot be distinguished even by experts from the natural product, has been produced artificially by the action of steam at high pressure and high temperature upon the carbides of metals, more especially those of iron; the water is decomposed into its constituent gases, upon which the oxygen combines with the metal, and the hydrogen, in part at least, with the carbon.

If this theory be true others are not necessarily false, for petroleum varies in its constituent elements, and may therefore be formed variously.

In view of the certainty that our coalfields will sooner or later be exhausted, even if coal be only used at the present rate, it is reassuring to fall back on such a theory as that of Prof. Mendeleeff, which regards petroleum as inexhaustible because always in process of formation and that in all parts of the world.

Petroleum is an almost pure mixture of hydro-carbons, the American variety consisting of hydro-carbons homologous with marsh gas or fire-damp, CH_4, in their chemical formulæ. Other varieties differ in composition. Heat causes a succession of vapours to be given off in the distillation of petroleum. In some districts some constituents of the rock-oil issue from the ground in the form of gas, even at ordinary temperatures and pressures.

In America natural gas is yielded in profitable quantities over certain small and intermittent areas. The chief centres of production are Olean, New York; Oil City, Pennsylvania; several places between Pittsburg and Wheeling; and others near the boundary between West Virginia and Kentucky, all being in a coal region. The gas is found in rocks, not far below the coal measures, from 1,200 to 1,500 feet below the surface.

A gas-well was discovered in Findlay, Ohio, about 50 miles South of Toledo, on January 20th, 1886. The gas was taken 48 feet above the ground, through a 6-inch iron pipe. On being lighted the flame reached a height of from 20 to 30 feet above the pipe. The noise of the escaping gas was such as to be heard at a distance of 5 miles, and the whole town was brightly illumined by the light of the flame. The amount of gas yielded by this well has been estimated to be as much as 40,000,000 cubic feet per day. Its pressure was not measured, but in some wells of Western Pennsylvania the pressure has been estimated at 750 or 800 pounds to the square inch. This is probably an extreme estimate, but on being tapped the gas often rises with a pressure of from 100 lbs. to 500 lbs. to the square inch.

The Findlay well is fully 100 miles outside of the coal region and is supplied by gas from the Trenton limestone, at a depth of about 1,200 feet.

In 1882, the total capital employed in the use of natural gas for fuel in America was estimated at only 215,000 dollars (£44,791). Two years later the amount had increased to 1,500,000 dollars (£312,495). In Sept. 1885, 1,500 dwellings and 150 factories and mills in the city of Pittsburg only, were depending upon natural gas for fuel. In Western Pennsylvania, several cities are supplied with natural gas conveyed from 20 to 50 miles through wrought iron pipes. In Pittsburg it is said that 1,000 cubic feet of gas are equivalent in heating capacity to 82 lbs. of the local coal. In 1889 the average daily consumption of natural gas in the United States was said to be over 1,500,000,000 cubic feet.

The use of natural gas as fuel at steelworks and in rolling mills in the United States was for a time increasingly prevalent; in 1892, owing to a smaller supply, its use declined. In that year there were 74 mills wholly or partially depending upon this product, this being a reduction of 30 in two years, notwithstanding the development during that period of the gas-bearing region in the central part of Indiana, where several iron and steel industries have arisen. Bituminous coal, gas coal, and petroleum are used as substitutes at those works which have been obliged to abandon the use of natural gas. Fresh bores are continually being made in order, if possible, to maintain a gas supply.

From the bed of a diminutive river called the Cat Fish Run, in the petroleum regions of Pennsylvania, the natural gas rises in bubbles, which explode on the surface. Not many years ago, some visitors to the neighbourhood, by way of experiment, floated some burning rags down the stream towards the gas, which then ignited with an explosion. Other instances have occurred in which the gas has been lighted on the surface of the water at night, giving the magnificent appearance of a sea on fire.

Such a fire was observed in 1845 on a small scale on the river Wear, near the city of Durham, and from its singular appearance attracted much notice. The *Durham Advertiser* in August of that year relates that at certain points in the river near Framwellgate Bridge on perfectly calm days large numbers of air or gas bubbles issued from below as though the water were boiling. A gentleman in the neighbourhood, thinking that it might be due to the escape of light carburetted hydrogen through a fissure from a seam of coal under the bed of the river, determined to make an experiment. For this purpose, choosing a day when there was no wind, he moored a boat alongside the disturbed surface of the water and set fire to the gas. Subsequently a pipe with a bell-mouthed end was fixed over the supposed fissure in the river bed, and the gas collected and conveyed through it to a reservoir floating on the surface. The receptacle was provided with a burner and glass chimney. A brilliant jet of flame could then be commanded at pleasure. There were several other places beside the one at which the experiment was conducted, the total escape from which amounted to many cubic feet of gas per minute. In favourable weather these could be ignited by holding a light close to the water until the ascending gas came in contact with it. In the absence of wind, rain, &c, separate clusters of gas jets could thus be lighted on the water, the extreme clusters being distant about 100 yards from one another. This series of burning jets scattered about in irregular patches on the surface of the water when seen from the bridge at night looked like a burning river. No coal workings were known to exist within two miles which could cause a subsidence and the occurrence was probably due to a large natural accumulation of firedamp at a fault, which extended to the surface with interstices at the lines of broken strata.

In the neighbourhood of Baku, on the shores of the Caspian, gas issues from the ground in great abundance, and over large tracts, as though the soil were completely impregnated. The land glare of the fires by night is an amazing sight, as may well be imagined. In some instances, the flame is utilized for domestic purposes.

The analysis of the gas at Pittsburg is as follows:—

Marsh gas	67 %
Hydrogen	22 ,,
Ethylic hydride	5 ,,
Nitrogen	3 ,,
Olefiant gas	1 ,,
Carbonic acid, carbonic oxide, oxygen	2 ,,
	100

Natural gas in the United Kingdom occurs only in limited quantities. That which is found in coal mines has been fully dealt with elsewhere in this work. Gas given off at the Wallsend Colliery was burned on the surface for several years, and in many other collieries, "blowers" of firedamp have been conveyed to the surface and used there. In two of the three series of experiments made with safety lamps in explosive mixtures by the Royal Commissioners on Accidents in Mines, 1886, two natural gases were made use of, one given off at a powerful blower at the Garswood Hall Colliery, near Wigan, and the other at the Llwynypia Colliery, South Wales. In the former case, the gas issued from the Wigan 9-foot seam which had just been reached in the newly sunk shaft, and was quite unworked at the time of the experiments; in the latter, the gas was obtained from the sandstone about 60 yards above the 6-foot seam of the Rhondda valley, and piped to the surface. Gas occurs in the jet rock of the Upper Lias in East Yorkshire, together with some heavy liquid bitumen; and it also finds its way down into the ironstone mines worked in the Middle Lias. A blower was burned for over 20 years in the Grag Hall ironstone mine, a few miles south-east of Saltburn. In some of the first borings for salt at Middlesborough, gas was found, and oil at Seaton Carew, both being as it is believed, from the upper beds of magnesian limestone. A deep boring at Port Clarence was continued 150 feet below the salt, in order to prove the magnesian limestone. The limestone yielded traces of bitumen, and there was also a constant escape of gas, which contained 83·2 per cent. of hydro-carbons, and 16·8 per cent. of nitrogen. Gas was found afterward at the same place in greater quantities under considerable pressure, together with some petroleum.

Petroleum has also been found at Worsley (near Wigan), and West Leigh in Lancashire, in Shropshire, in Derbyshire, in North Staffordshire, and in Lanarkshire, but never in sufficient quantities to justify its working. The latest discovery is that in Somersetshire.

In Scotland, the working of the bituminous shales of the coal-measures is a very important industry. The oil shales are found in calciferous sandstone measures situated between the carboniferous limestone measures and the old red sandstone. They occur in small basins in the centre of Scotland. The seams of shale vary in thickness and quality, and are usually won by mines commencing at the outcrops on the surface and following the seam to the full dip. The thick seams are worked by pillar and stall method, the thin ones by longwall. The small shale which passes through one-inch riddles is stowed underground. Fire-damp is given off in the shale mines, which, however, are worked with open lights.

On distillation the shales yield naphtha, sulphate of ammonia, and crude oil. The crude oil when treated in the refinery produces burning, lubricating, and paraffin oils.

It has been suggested that although the strata of the United Kingdom are as fossiliferous as other regions in which petroleum and gas are largely produced, the palæozoic strata have been subjected at various ages to repeated foldings and denudations, leaving exposed edges which have facilitated the escape of any gas or petroleum which might else have been retained.

In Europe, petroleum has been found in Germany, France, Italy, West Sweden, in Spain in small quantities, Austro-Hungary, Hungary, Roumania, and Moldavia. There are also several deposits in Russia, beside that of Baku, which is one of the largest and oldest known. The oil is here found nearly colourless in soft sandstones interstratified with impervious clays of different colours, and requires no purification before being burned in lamps. Out of 1,600 square miles of oil-bearing territory, only about 5 square miles have been developed. Up to 1890 some 500 wells had been bored on that area, the deepest being about 850 feet, while the average depth is about 500.

The known oil-fields of the United States are situated chiefly in New York, Pennsylvania, Ohio, West Virginia, Kentucky, Tennessee, and California. Between 1860 and 1890, about 50,000 wells had been bored, some being 2,500 feet in depth, while the average was 1,500 feet. The bore-holes vary in size, but are usually from 3 to 4 inches in diameter. At the 1893 World's Fair at Chicago, the Exhibition authorities used petroleum exclusively for the boilers there. It was brought from oil-wells in Ohio and Indiana. The oil-fields of Canada are near Lake Erie, and also on the coast and near the Rocky Mountains. The most important of these are in the valley of the Mackenzie and Athabasca rivers, and, according to the report of the Senate of Canada about 1886, they constituted the largest petroleum field in the world. About 10,000 square miles of this petroleum territory has been marked out as a reserve to be constituted as a Crown domain. The whole area is quite undeveloped. In Mexico, there are large deposits which have not yet been extensively worked.

Petroleum occurs in many of the West Indian Islands.

Turning to South America, Venezuela and Peru have very extensive oil-fields; although no great efforts have yet been made to develop them. Those of Peru are likely to be of great importance. Petroleum occurs also in the Argentine Republic, and is said to have been found in Bolivia and New Granada.

Oil-fields have been discovered in South Australia on the banks of the Coorang, a little to the north of Salt Creek, and in New South Wales.

In New Zealand, there are similar fields undeveloped near Poverty Bay, Auckland, and in the Taranaki district. The latter are interesting from the fact that the oil bubbles through the pulverised iron ore that forms the beach along the west coast.

In North Africa, petroleum has been found both in Egypt, and, according to report, in Algiers.

In South Africa, there are oil-fields in the Transvaal and in the Orange Free State. Little is known of these as yet, and there is room for speculation as to the future of Africa as an oil-producing country.

The Persian oil-fields are situated in the valley of the river Karun, but little has been done to develop them.

In Burmah and India petroleum occurs, and may some day lead to the development of important industries there.

It is also stated to have been discovered in China, and is known in Sumatra, Java, Borneo, and other islands in the eastern seas.

Petroleum deposits have also been worked in Japan.

The oil-bearing regions of the world, therefore, are vast and practically inexhaustible, even apart from the theory of constant production. The use of paraffin, both in a crude state and greatly purified, has made enormous strides during the last few years for public and private illumination, and for fuel and manufactures. The application of mineral oil as a fuel for ocean-going vessels, locomotives, and stationary engines is likely to undergo extensive development.

CHAPTER XIV.

SURVEYING AND PLANNING.

General Use of Working Plans—Chaining Distances—Construction of the Spirit Level—Of the Levelling Staff—Method of Taking Levels—Ordnance Bench Marks—The Adjustments of the Spirit Level—Forms of Levelling Book—Mr. Wells's Hints on Levelling Operations—The Miner's Dial—Davis's Improved Hedley Dial—The Hoffman Joint—Stanley's Improved Miner's Dial—Stanley's Tripod for Dials—The Reflecting Cup—Method of Needle Surveying—Vernier or "Fast" Needle Surveying—Reducing Angles to an Original Base Line—Stanley's Hanging Dial—Hanging Clinometer—Stanley's Mining Survey Lamp—Henderson's Rapid Traverser—Declination, Diurnal Fluctuation, and Dip of the Needle—Construction of the Transit Theodolite—Method of Using the Theodolite Underground—The Plain Theodolite—The Adjustments of the Theodolite—The Protractor and its Use—The Parallel Ruler and its Use—Scales—Ogle's Protractor—Survey Book—Paper for Colliery Plans—Meridianal Lines on Plans—Separate Plan of each Seam's Workings—Desirability of Placing Full Information on Colliery Plans—Plotting the Surveys—"Tieing" Surveys—Plotting Sections—Colouring Colliery Plans—Computation of Areas and Produce of Coal from them—Setting Out Railway Curves—Making Geological Sections—Louis's Improved Davis's Clinometer—The Celluloid Slide Rule—Practical Questions and Answers.

A VERY important item in colliery operations is the making and maintaining, in compliance with the Mines Act, 1887, accurate plans of the workings in the different seams.

Plans are indispensable as a guide to the mining engineer or colliery manager in directing the workings, designing the best means of ventilation, guarding against old workings containing water or gas, keeping well within and leaving a proper barrier against the boundaries defined in the lease of the colliery, and for giving notice to the proprietors of railways, canals, &c., when the workings have approached within a specified distance of their surface property. In some instances the royalty rents are chargeable on the amount of coal worked; such amount being calculated from the areas as proved by survey and plotted on the working plans.

Surveying instruments are divided into three classes:—1. Instruments for measuring distances. 2. Instruments for measuring angles. 3. Instruments for plotting the survey.

Those ordinarily used for Class No. 1 are the chain or tape, the spirit level and levelling staves. Those for measuring angles, which are applicable to mine surveys, are the miner's dial and the theodolite. Under the head of No. 3, the different forms of protractor, the parallel ruler, the T-square, the set-square, and scales, will be considered.

Gunter's Chain is that which is almost universally adopted for measuring distances. It is 66 feet or 4 poles in length, and is divided into 100 links joined together by rings. The length of each link together with the rings connecting it with the next is $\frac{66 \times 12}{100} = 7{\cdot}92$ inches. To every tenth link are attached pieces of brass of different shapes for readily counting the distances which may be fractional parts of a chain.

With the chain, on the surface, 10 or 11 arrows also are used, the latter number being preferable, but they are not often used in underground measuring as they will not stick in the floor, and the custom is to mark the end of a chain with chalk on the floor or the rails. Care is required to measure straight between the two marks set up, which on the surface are usually rods, and underground must be lights.

The best land chains are of steel, and made light and strong. Fig. 425 shows a land chain and arrows. The chain, whether made of steel or iron, has long links formed by turning up the ends of a length of wire. Three small oval links are placed between each pair of long links. Three interval links are found to cause the chain to *kink* less than when only two are used. Each oval link is sawn through at the meeting line, which is brought up on one flat side of the oval in bending it from the wire. The small links are used for the adjustment of the chain, as they may be closed to shorten, or forced open to lengthen it. There are generally four *swivels* in the length of the chain, two of which are at the handles: these prevent the chain from becoming twisted in turning the handles over in use.

The length of an ordinary measuring-chain is always tested, and, if necessary, adjusted by a standard chain before it is used in a survey of any importance.

Standard chains are of the same form as the ordinary steel chain, but all the links are hard-soldered after being adjusted link by link.

Standard chains are used at collieries only for testing and adjusting ordinary chains.

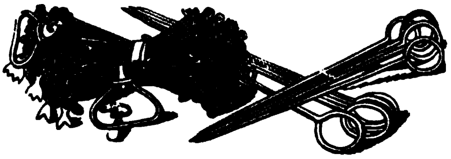
Fig. 425.—Land Chain and Arrows.

Horizontal distances only are shown on the plan, and these will not be the actual distances measured where the ground rises or falls. On the surface, approximate horizontal measurement may be obtained by holding one end of the chain up, so as to keep it in a horizontal position, and plumbing from the handle to the ground; but where the ground is very steep, it is impossible to hold out a whole chain-length in this way, and it has to be done by using a part of the chain, say 40, 30, or 20 links at a time, according to the fall of the ground. It is, however, difficult to tell when the chain is being held horizontally, and it is much better instead of adopting this method, to measure the distances along the surface of the ground; and by taking the angles of elevation or depression over the several inclined parts of the line with the instrument used for measuring the angles, the correct horizontal distances may be computed. The underground measuring admits of no other method. Tables prepared from calculations, may be obtained, showing this correction for every 100 links for any angle whatever. For measuring the depths of shafts, or in making plans whose areas are required in square feet, or it may be for plans of surface buildings or other purpose, a chain 100 feet in length, divided into links a foot long, is to be preferred. Chains made of steel are lighter to use than those made of iron.

The tape may be made of any length, but those mostly in use are 33, 50, 66, or 100 feet long, and there are different ways of dividing them. The most generally useful one is divided into feet and inches on one side and links on the other, so that it can be used to measure either. In surveying it is used to measure long offsets from the chain, the lengths of buildings, &c., on the surface, where its use is a necessity in addition to the chain, but it is seldom required in underground surveying. Tapes shrink in use, and should be frequently tested by standard measurement.

The Spirit-Level is an instrument used in measuring the vertical distances between different places. It consists of a glass tube which is not quite cylindrical in form, but having its diameter largest in the middle and decreasing slightly and with great regularity from the middle to the ends. The tube is nearly but not quite filled with spirits of wine, thus leaving in it a bubble of air, which rises to the highest part of the tube and which, when the instrument is adjusted, has the two ends of the bubble equally distant from the middle. A scale is generally scratched on the glass to guide the operator in adjusting the instrument. This spirit-level is fixed to the telescope by a joint at one end and a capstan-headed screw at the other so as to raise or depress it for adjustment.

The oldest form of surveyor's level is that termed the *Y-Level*, a modern form of which is shown in Fig. 426. It is but little used in Great Britain now, although it has much to recommend it, because of the facility its construction affords of verifying its adjustments before commencing work. It receives its name from the Y-formed bearings YY″ which support the telescope. These in turn rest upon the limb L. The telescope has two circular collars soldered upon it at positions exactly corresponding with the Y's. The collars are turned perfectly cylindrical, and parallel on the surface with the axis of the telescope, and ground in a gauge-plate to exact size, so that the telescope may be turned end for end in the Y's without altering the lineal direction of its axis in reversing it. The telescope is prevented from shifting longitudinally in its Y's by a pair of flanges placed on the inside of the collar-pieces.

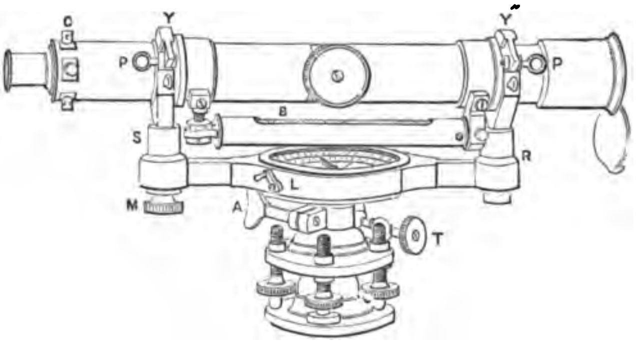

Fig. 426.—SURVEYOR'S Y-LEVEL.

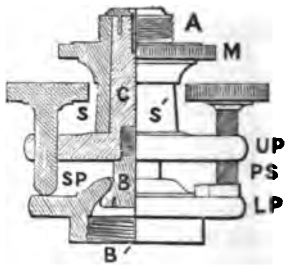

Fig. 427.—SECTION OF PARALLEL PLATE AND VERTICAL AXIS—ARRANGEMENT OF Y- AND OTHER LEVELS.

The Y's are erected upon the limb, to which they are each fixed firmly by a clamping-nut at one end R, and a milled-head clamp at M. The telescope is held down by strap-pieces, each of which has a joint at one end and a loose pin at the other PP. The pin is attached to the instrument by a piece of cord and to a loop formed in its head, so that it dangles, but cannot be lost, when out of use. At the top of the inner side of the strap-piece under YY″ a piece of cork is inserted in a cave. The cork, by its elasticity, keeps an equal but light pressure upon the collar of the telescope. It will be seen that by the above plan of holding the telescope, it is so far free that it may be revolved on its axis, by which perfect adjustment of the diaphragm may be made in any direction.

The general construction of the vertical axis and parallel plate of the Y-level is shown in Fig. 427, the left-hand side being a half-section. A is a screw by which the parallel plate-foot is attached to the limb of the instrument; M, a large milled head, by means of which the screw can be brought up firmly to its collar; SS′, the socket which is ground to fit the cone C; C forms a part of the upper parallel plate UP; B, a ball pin which screws firmly into C; LP, lower parallel plate, part of which forms the ball socket, so that the whole instrument rocks about the ball B as a centre by the action of the parallel plate-screws PS; B′, female screw for fixing this part, which is called altogether the parallel plates, to the tripod head. In the old Y-level there was usually a clamping-screw upon the axis for slow motion, which generally caused a strain upon it. Modern instruments are made without clamping-screws. The parallel plate-screws are tapped, that is, have female threads cut into the upper plate, UP, and their points press the lower parallel plate, LP, at certain points, there being a stop-piece placed round the point of one screw to prevent rotation.

The diaphragm of the telescope, Figs. 428 and 429, is formed of a stout disc of brass, having a central hole of about ·3 inch diameter. Upon the side which is placed next the eye-piece the hole is brought to a thin edge by an internal bevel or countersink, which leaves the hole much larger at its off-surface, Fig. 429. The disc is held in its place and adjusted by four capstan-headed screws, termed collimating-screws, two of which are shown in section as C C′, their points being tapped into the rim of the diaphragm frame P. The screws are placed through a stout collar. The diaphragm has generally three spiders' webs crossed in the manner shown in the centre of Fig. 428. The eye-piece is screwed into the thick plate, Fig. 429, and adjusts to the focus of the webs. The webs are fixed to lines drawn on the diaphragm.

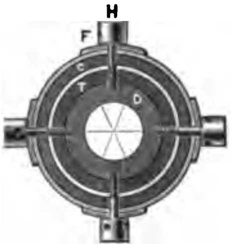

Fig. 428.—Elevation of Diaphragm.

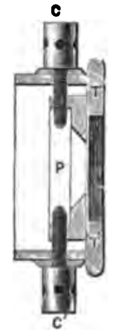

Fig. 429.—Section of Diaphragm.

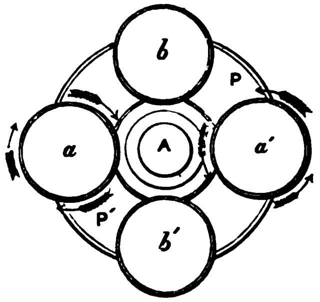

Fig. 430.—Diagram Plan of Parallel Plate Screw Milled Heads.

To set up the Y or other level with parallel plates, the tripod stand is opened out to form a firm support for the instrument; the toes of the legs are then each separately pressed into the ground sufficiently to make the instrument stand quite firmly. The instrument is then screwed down tightly upon the tripod head.

The eye-piece is then adjusted by sliding it gently in and out until the webs can be seen most distinctly. On a bright day a white pocket-handkerchief or other light covering may with advantage be thrown singly over the object-glass, to prevent any confusion from objects in the field of view during the focussing of the eye-piece.

For the setting-up adjustment of the telescope, it is brought to be directly over one pair of parallel plate-screws, PS, Fig. 427. The milled heads only of these screws can be seen in the diagram plan, Fig. 430, *a a′* being the opposite pair over which the telescope will be assumed to be at first placed. The level tube is now brought to adjustment by bringing the bubble to the centre of its run by means of the parallel plate-screws, *aa′*, by taking the milled heads of these screws, one between the balls of the thumb and forefinger of each hand, and rolling them simultaneously, the one in one direction and the other in the reverse. This action gives the axis of the telescope motion in one direction or the other. Thus, by the screws being rolled inwards, as shown by the direction of the arrows in the diagram, the left-hand side of the instrument would be raised; if turned the reverse way, the right-hand end. The opposite end always requires to be raised from that to which the bubble runs. If the legs are suitably placed on rather soft ground, one or other of them may be pressed down to assist in the adjustment.

When the level tube is adjusted over the screws *aa'*, it is then placed over *bb'*, and adjusted in a similar manner, returning again to the position *aa'* for final adjustment. When the instrument is in perfect adjustment the bubble will remain in the centre of its run throughout a complete revolution of the instrument upon its vertical axis.

Some practice in setting-up is necessary in order to avoid undue pressure of any of the screws upon the parallel plates. When the instrument is correctly set-up the staff may be sighted, and the telescope brought to true focus upon it by moving its milled head until the divisions of the staff are as sharp as possible, and then moving the eye upwards and downwards, to be sure there is no error of parallax. After this the final adjusting of the bubble should be made, care being taken that it is brought to the centre of its run by noting the number of divisions scratched on the glass on each side from the centre.

To adjust the axis of the telescope truly parallel with the periphery of its supporting collars in its Y's, the four capstan-headed screws which adjust the diaphragm are used, one being marked C, Fig. 426. Having the adjustment of the eye-piece in focus for the webs, and the object-glass focussed upon a distant distinct small object or mark, and without parallax, the instrument which carries the telescope is then exactly adjusted to make the intersection of the webs cut the mark. The telescope is now turned half round on its axis, so that the lower part becomes the upper, and observation is again made of the distant mark. If the same intersection of the webs falls on the same point of the object, the collimation adjustment is perfect. If it does not so fall, the upper capstan-headed screw at C, or the under opposite one, is loosened by means of the small pin provided with the instrument, and the opposite screw tightened until the webs are brought over a point situated half-way between the points cut by the first and second observation. The telescope is again directed to the point first observed, and the adjustment checked to see if it has been done correctly, that is, if on being reversed it cuts the same point, or whether it requires further adjustment by the same process as before. The other web of the diaphragm, at right-angles to the first, is adjusted in a similar manner to the first, but with the other pair of capstan-headed screws.

To adjust the vertical axis the eye-piece is first brought to focus on the webs. The telescope is then brought over one pair of parallel plate-screws opposite each other, and the instrument is levelled. The Y's are then opened out and the telescope is directed so that the intersection of the webs cuts or covers any distinct small mark upon a distant object, or preferably upon the centre reading of a foot-line upon a levelling-staff. There is no objection to adjust slightly to this by the parallel plate-screws, because this adjustment is independent of the level of the instrument. The telescope is then lifted out of its Y's, turned end for end, and replaced. The telescope is now turned half a revolution on its vertical axis, and the webs are again brought to read on the staff, if one is used. If they now fall upon the same spot or foot-line the vertical axis is perfectly perpendicular to the axis of the telescope in this direction. If the webs do not fall upon the first reading or point, the amount of difference of reading is recorded and this space is bisected; so that now, if the telescope be adjusted at its bearings upon the limb supporting it by the milled head, M, Fig. 426, for the webs to cut the bisection, the axis will be perfectly perpendicular in the direction of its bearing-socket. The telescope is now turned until it is over the other pair of parallel plate-screws which were not used at first, so that it is at right-angles to its first position while the same process is repeated. As the handling of the instrument in one adjustment nearly always disturbs the other, the whole of the process may have to be repeated until the instrument reverses in any direction; but this for final adjustment is better deferred until the adjustment of the level tube, to be next described, has been made.

To adjust the level tube, the telescope is placed as before over an opposite pair of parallel plate-screws, and these are adjusted until the bubble is in the centre of its run. The telescope is then turned half a revolution, so that it is placed over the same pair of screws in the reverse direction, and the displacement of the bubble from the centre is now noted. The capstan-headed bubble-screws at the end of level B, Fig. 426, are then adjusted to one-fourth of the difference observed, and the parallel plate-screws are adjusted for the other fourth, so that by these two adjustments the difference of the run in two positions, obtained by sighting a staff, is bisected. The same process is repeated over the second opposite pair of parallel plate-screws. If this be very carefully done with a correctly-divided bubble, the Y's of the telescope may be opened out and the telescope be reversed end for end in its Y's, and the bubble will remain true. As a check, however, the adjustments should be all gone over a second time.

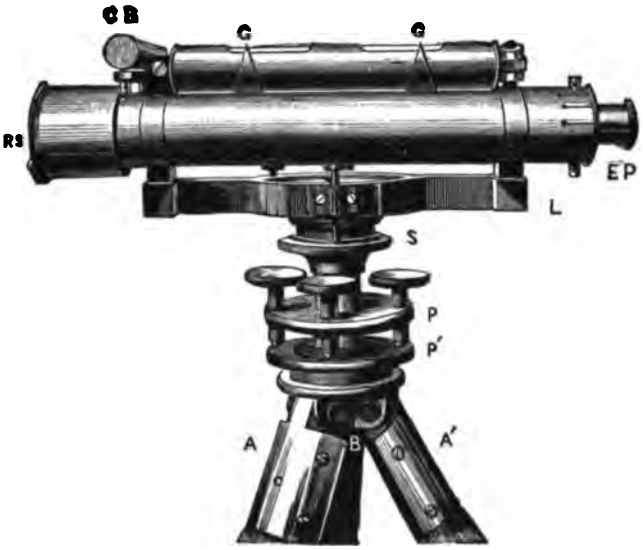

Fig. 431.—Dumpy Level.

A modern form of *Dumpy Level* is shown in Fig. 431. The telescope carries a ray-shade, RS, at the object-glass end, to work in the field to eastward or westward, facing a low sun. The eye-piece, EP, is adjustable to the webs in the telescope by pressure in and out. Two straps or bands are accurately fitted and soldered round the tube of the telescope, one of which carries a hinge joint, and the other a pair of locking-nuts to support the level-tube, GG, which at the same time permit its adjustment. The level casing-tube has two three-quarter bands, which slide upon it, pointed at one end, GG; these adjust to the length of the bubble for changes by temperature. The lower part of each strap-piece is left a solid block of metal, to give very firm support to the telescope as it rests upon the limb, L, beneath. The limb may be either a casting with a socket-screw only in its centre, or a compass-box may be formed in the centre and the socket-screw be placed under this, as it is shown in the figure. The attachment of the telescope support to the limb is made by three screws, two of which draw the limb down and one in the centre presses it upwards, as shown in the section, Fig. 432—CC′, telescope; TT′, drawing-screws; P, pressing-screw. By this means firm adjustment may be made either by raising or lowering one end of the telescope, as also by a lateral rocking motion should the web or bubble not be quite to position. The general construction of the vertical axis, parallel plates, tripod head, and tripod is the same as that of the Y-level already described. The cross-level, CB, Fig. 431, at right-angles to the principal level, allows the setting-up of the instrument to be completed approximately, without turning the level a quarter revolution backwards and forwards several times during the operation, as was necessary in the setting-up of the old form of Y-level. Modern Y-levels are, however, made with cross-levels.

The diaphragm of the dumpy-level is generally webbed with two vertical webs and one horizontal. In use, the staff is viewed through the vertical webs, and these indicate whether it is held upright sideways, although they cannot be used to detect a backward or forward leaning of the staff. The upper margin of the portion of the horizontal web between the two vertical ones is the index of level to which all readings are made, either for adjustment or for reading the levelling-staff in the field. The four-screw adjustment for a level used in rough work with capstan-head screws, shown in Fig. 428, which is necessary for the adjustment of the telescope in Y's, has been abandoned for many years in the construction of the dumpy-level, and given place to that shown in Fig. 433. In this plan there is no lateral adjustment; the diaphragm is carried as a frame in a dovetail slide, and is adjustable by vertical screws only. The figure shows the face of diaphragm BB′ slide-pieces. A is the slide moved by capstan-head screws.

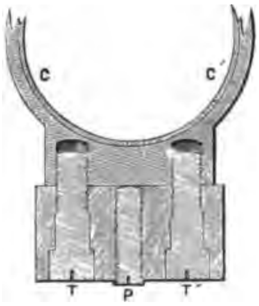

Fig. 432.—Attachment of Telescope Block to Limbs.

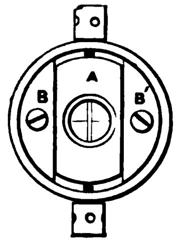

Fig. 433.—Diaphragm of Dumpy Level with Webbed Stop.

A 12-inch dumpy level should read the ·01 foot on a Sopwith staff at 5 chains with a webbed diaphragm; a 14-inch dumpy should read the ·01 foot at 10 chains.

The Dumpy level sometimes has a circular level placed immediately under the telescope in the place of a cross level.

Fig. 434 shows Messrs. Davis & Son's Dumpy level, with Hoffman patent joint. It has achromatic lenses, and all parts are of gun-metal. Two eye-pieces are provided with the instrument, and a tripod of polished mahogany, the head of which carries a screw about 1½ inches in diameter with coarse thread, which fits into a socket on the lower parallel plate of the level. The three legs shut together in one round staff or may be opened out to stand firmly on the ground, however uneven it is. The telescope has a focussing arrangement, and near the eye-piece is the diaphragm carrying cross-webs, by means of which the observer reads the staff.

The construction of the level staff in common use is that due to the late Thomas Sopwith, sometimes called the *telescopic staff*, the face view of which is shown in Fig. 435. For ordinary open field work this is made 14, 16, or 18 feet in its extended length; but generally, except for levelling on mountainous land, the 14 feet is used. This staff when closed is about the same length as the tripod—5 feet 4 inches, which is convenient for carriage. As at first made it was liable to become somewhat shaky when extended. It is now usual to make the boxes slightly diminished in size towards the upper part, so that they form a fair fit when opened out, but are very free when closed, which prevents their getting jammed if put away damp from rain.

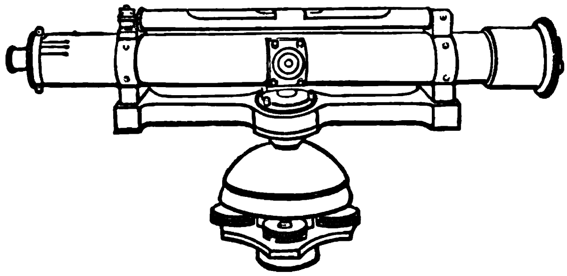

Fig. 434.—Dumpy Level with Hoffman Joint.

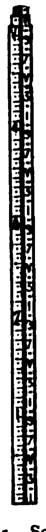

Fig. 435.—Sopwith Pattern Staff.

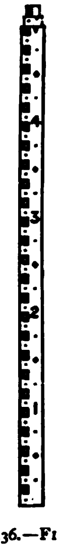

Fig. 436.—Field's Pattern.

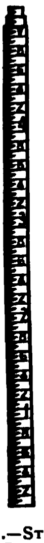

Fig. 437.—Stanley's Old Pattern.

The ordinary construction of Sopwith's staff and the best mode of manufacture are shown, with the joints grooved together, in section, Fig. 438. The outer tube or case A, which in the 14-foot staff is 5 feet in length, is made of mahogany $\frac{5}{16}$ inch thick, the front being $\frac{1}{4}$ inch. The outer dimensions of the section are

$3\frac{1}{8}$ inches by 2. The second tube, B, is 5 feet 1 inch long, of outer dimensions $2\frac{3}{8}$ inches by $1\frac{1}{4}$. The inner slide, C, is solid, 5 feet 2 inches long, $1\frac{3}{4}$ inches by $\frac{3}{4}$. All the slides are sunk on the face about $\frac{1}{16}$ inch to prevent the divisions from being rubbed by exposure in sliding together. The slides have each brass shoes and cap. They are held, when extended, by a spring catch, the detail of which is shown in Fig. 439, section of Fig. 438—S, spring of T form screwed firmly to the edges of the box. The catch is made at A, over the edge of the brass cap *a*. The spring should be of very hard rolled brass. It is well to have one or two brass bands round the body of the outer casing to secure this as far as possible from being split by accident.

In Fig. 438 is shown an arrangement of Mr. Stanley's for the protection of the front of the staff when it is carried or handled, or when it is strapped with the tripod. The two light shutters, E, E′, may be folded over the face, or opened right back to the sides of the staff when it is in use. In the illustration, one shutter, E, is shown closed, and the other, E′, opened back, leaving the face of the staff exposed to view. At the inner edges pieces of leather are attached to the shutter, so that when the tripod is strapped upon it the shutter does not bruise the face or tend to crack it, but the pressure comes upon the sides of the face, where it is strong to resist it. This arrangement makes the staff heavier.

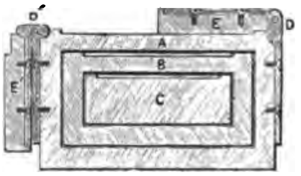

Fig. 438.

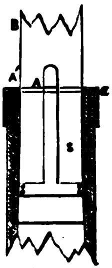

Fig. 439.

Sections of Sopwith's Staff.

With the diminished width of reading to $1\frac{1}{4}$ inches in the upper length of the Sopwith staff it is not so easily read when sighted through the level as are forms of staves which maintain a wider reading-surface right to the top; but this objection is overcome in Mr. Stanley's staves by placing dots in the upper extension instead of the narrow figures. The form of reading is shown in Fig. 435, the dots at the ends of the lines being introduced by Mr. Stanley to render the staff more distinct than when lines only are used, as in the ordinary pattern.

Sopwith's pattern is sometimes printed on paper for pasting on the staff, and in this manner the staff can be made cheaper than by drawing the readings in solid paint. Paint, however, wears much better, keeps cleaner, and the paint, when varnished, adds very much to the durability of the staff, exposed as it must be to rainy weather.

For reading the Sopwith staff the foot readings are taken from the tops of the red figures. The decimal 1-foot figures are in black, and are all odd numbers, 1, 3, v, 7, 9. These read also from the top. The height of the figure is exactly ·1 foot, so that the bottom of each figure reads the lower even number—thus, the bottom of 3 reads 2, of v reads 4, of 7 reads 6, and of 9 reads 8. The 6 and 9-foot figures, if made alike, from effect of telescopic inversion, may cause error. To avoid this Mr. Stanley makes the head of the 9 a solid black block.

Many surveyors design their own staff-readings. A good plan to avoid the necessity of having the staff raised from the ground in order to sight a red figure when it is in use very close to the level, is to have small repeat red figures marked every 3 inches apart on the staff, between the larger red figures marking the feet. Thus, between the top of the first foot and the second there would be three small red 1's marked, between the top of the second foot and the third three small 2's, and so on throughout the staff. There is no confusion in this, because the extra red figures are made so much smaller than the others.

Figs. 436 and 437 show forms of Field's and Stanley's staves which have been designed to render staff-readings clearer, but they do not seem to have come much into use. Many other staff-readings have been designed.

A Sopwith mining-staff for underground levelling has the reading arranged precisely the same as that of the surface-staff, but as the height in which it has to be used is limited, it is commonly made from 2 feet 3 inches to 3 feet 6 inches in length in its closed form, and from 5 to 9 feet when fully opened out.

Mr. Stanley has designed a portable staff which is made in reading-lengths of 18 inches, somewhat like a French folding rule. The staff may be formed of three, four, five, or six lengths, opening out respectively 4 feet 6 inches, 6 feet, 7 feet 6 inches, and 9 feet—Fig. 440. The separate pieces are flat boards, slightly sunk on the face to prevent the divisions from being scratched in opening and closing, but left solid at the joint-ends. The boards are attached together with a kind of rivet at each joint. A strong spring at the end of each piece, with a catch and notch, keeps the length opened or closed with sufficient rigidity. The entire length of the staff when closed is 20½ inches. The same kind of staff forms a very useful builders' or drainage-staff, divided in this case in feet and inches; and it is convenient for carrying about.

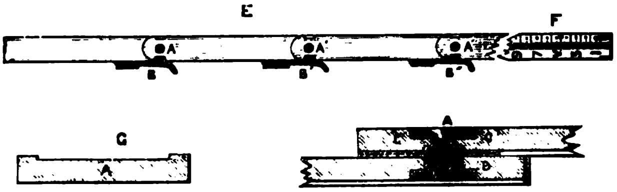

Fig. 440.—Stanley's Patent Mine Staff.

A mining staff brought out by Mr. James R. Linsley, the surveyor at Cramlington Collieries, consists in dividing the staff into an upper and lower division, the former sliding into the latter. To the top of the sliding portion is fixed a spring drum, on which coils a tape, whose lower end is attached to the top of the lower division of the staff. The figures and divisions are painted on this tape the same as on the staff, and in roadways of varying height the upper portion of the staff may be opened until close to the roof, and when this is done the reading of the staff is continuous from the floor upwards.

Mr. G. J. Jee's underground levelling staff is somewhat similarly constructed, but is made in two sizes by Messrs. J. Davis & Son, Derby, the 9-foot size closing down to 3 feet 6 inches, and the 6-foot to 2 feet 6 inches. Fig. 441 (A) shows a front and Fig. 441 (B) a side view of the staff. Both sizes consist of three divisions. The 9-foot staff has a 2-inch band attached to the top of the lowest division, which is 3 feet long, and as the middle division is raised the band uncoils off the brass drum shown in the figure. This drum is placed just below the top of the top division of the staff, and in unwinding off it the band is carried through a small brass roller on top of the division. The band is graduated upwards, and when the middle division of the 9-foot staff is out, reads continuously up to 5 feet 8 inches. If the roadway permit of it, after the middle division has been opened to its full extent, the top one may be drawn out, and the same band further uncoiled till it reads continuously to 9 feet.

Another form of levelling staff for use in mines has been introduced. In it the figures are painted in dark colours on a background of ground glass or other transparent material. When being used a light is placed behind the staff, and this gives the surveyor a better opportunity of making his observations.

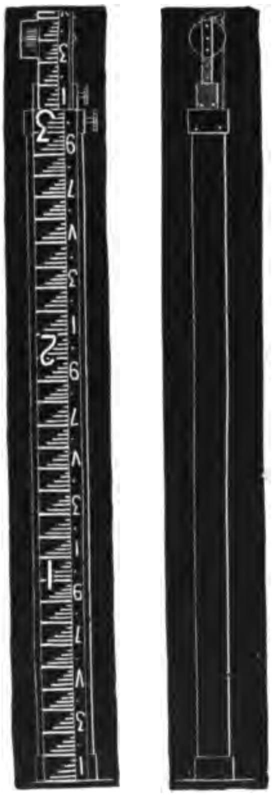
A. Front view. B. Side view.

Fig. 441.—Jee's Underground Levelling Staff.

To prevent injury to the staff during carriage, a convenient plan is to have two pads formed of stout ox-hide butt, each pierced with two slots near their ends at the exact distance apart of the width of the staff—Fig. 442. The strap, of calf leather, is passed from one slot round the staff into the other slot, and then passed round the tripod and pulled up tightly and buckled. The pad of course protects the front of the staff from grazing by the friction of the tripod against it. One pad is placed at the top, the other at the bottom of the closed staff and tripod.

Stanley's spring pads, Fig. 443, form a better protection against the cylindrical tripod pressing against the front of the staff than ordinary pads. In these a hard-rolled German-silver spring covered with leather distributes the pressure, which is thus greatest near the edges of the front, where the staff is strongest to resist. The section of this spring is shown at E, with the general arrangement of the staff S, and the tripod D, fixed upon it by the strap F.

For the entire protection of the staff a leather-bound sailcloth case is very generally used. This may be divided in two compartments for the staff and the tripod, with pads between. The whole case has a neat appearance, and forms a protection from slight bruises and dirt, either in travelling or when set up in an office corner for future use.

A triangular plate of iron, as represented in Fig. 444, may be used with advantage to rest the staff on during an observation. It is trodden down firmly by the staff-holder before he places the staff upon it. In use it gives a certain base to turn the staff upon from fore to back sight, and removes the necessity for the staff to be held continuously in one spot while the level is changed from one position to another.

The staff-level consists of a small circular level, shown in section, Fig. 445, the upper surface of which is formed of a glass worked slightly concave and fixed over a short cylindrical box. The glass is generally burnished into the box and cemented round with elastic cement. The box is nearly filled with spirit from a hole covered by a large-headed screw, which is fitted with an india-rubber washer. The circular level is mounted on a plate with studs. The studs fit in two holes with bayonet slots in the holding-plate, which is attached to the back of the staff. In use the staff-holder has to observe when the bubble under the concave glass is in its centre. A very little practice is required to hold the staff vertically by means of this little contrivance, which only weighs, with its pocket case, about 2 ozs.

From the unequal expansion of the brass and glass by heat, the spirit will slowly evaporate and the bubble will enlarge (say after a few months' use), but it is easily refilled by removing the screw and adding a little whisky or other spirit.

The staff-holder, Fig. 446, is very generally used in Germany and other parts of the Continent. The staff is sunk into one side of a hardwood block The block is turned at one end to form a handle. A second similar handle screws into the end of the block. This screw-handle by turning brings up a following-piece, shown inside next the staff, which is covered with leather. When it is screwed up, the staff may be held firmly by the handles only, without risk of the fingers coming in front. With this apparatus it is also held more easily and truly vertical.

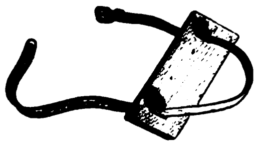

Fig. 442.—Pad for Holding a Staff and Tripod.

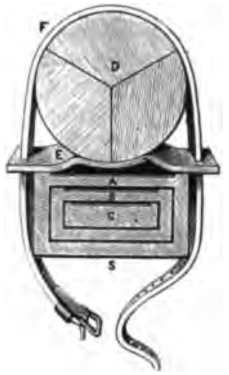

Fig. 443.—Spring Pad for Staff and Tripod.

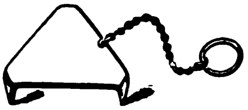

Fig. 444.—Iron Triangle to Support a Staff.

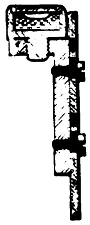

Fig. 445.—Section Staff Level, ½ Scale.

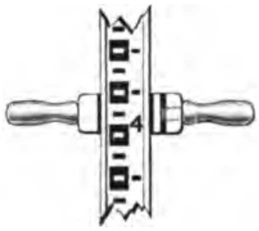

Fig. 446.—Staff-holder, $\frac{1}{10}$ Scale.

The operation of levelling is conducted as follows:—

Let A and B be two points (Fig. 447), and the difference of level between them is required to be known. Place the instrument at C and adjust it in a truly horizontal position. Then a levelling staff, divided in feet and decimals (to 2 places), being placed at A, read the height of the staff there and remove it to B. The telescope being turned round, read the height of the staff at B, and if now one reading be deducted from the other it gives the difference of level between A and B. The staff at A is called the back-station, and the reading taken there is called the back-sight, whilst the staff at B is called the fore-station, and the reading there is called the fore-sight. If the back-sight is greater than the fore-sight the difference of level is called a rise; if the back-sight is less than the fore-sight, the difference of level is called a fall. If in the above case the back-sight read 6·42 feet and the fore-sight 10·17 feet, there would be a fall of 10·17 − 6·42 = 3·75 feet, which, in other words, would mean that B was 3·75 feet below the level of A. When from the nature of the ground it is not possible to see from one point or position of the instrument the two extremes, a series of simple levels like the one indicated above must be made, the fore-station at each plant of the instrument being made the back-station at the next, and from the combination of all the results is obtained the required difference of level between the desired points.

When it is required not only to find the difference of level between two distant points, but to make observations that will enable a section to be drawn, showing the undulations of the ground along some specified route, Fig. 448, the stations must be chosen so that they are at the commencement of each change in the inclination of the ground, the distances between each station must be carefully measured, and these measurements must be reduced to horizontal measurements when plotting the section.

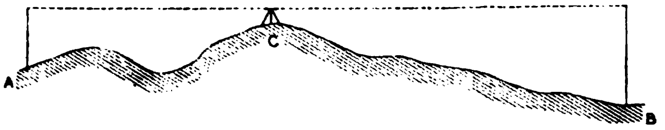

Fig. 447.—Illustrating the Operation of Levelling.

A number of sights may be taken with the instrument from the same position if the ground be very undulating and it is desired to note every change of inclination; in that case, it is usual to call all but the fore and back stations intermediate stations, and book their staff-readings as intermediate sights. It is usual also to assume that the starting point is so many feet (frequently 100) above datum line, unless some well-defined datum line, such as would be obtained from an Ordnance bench mark, is used; the reason for assuming the starting point to be a certain number of feet above datum line being the convenience both of keeping the book and of plotting the levelling.

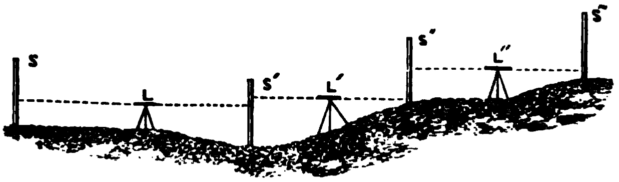

Fig. 448.—Practice of Levelling.

Ordnance bench marks are the broad arrows cut on stone walls, window-sills, coping-stones, or other prominent and convenient places by the ordnance surveyors. The ordnance plan-sheets of any locality have similar arrow-marks on them at the places shown on the plans where the marks have been cut, and, in addition, a number printed distinctly at the arrow, thus B.M. 562·25, which indicates that that particular arrow is 562·25 feet above mean sea-level at Liverpool. These marks serve as a record by means of which subsidence or elevation of land may be measured. The counties of Lancashire and Yorkshire, *e.g.*, were re-surveyed by the Ordnance surveyors in 1890, and many changes in the surface level since the original survey of 1852 have been brought to light. At a point on the line between Warrington and Adlington, the surface is as much as 6·523 feet lower, while the bench mark in Wigan old church has sunk 5·825 feet. In the latter case the surveyors observed a subsidence of one foot in a year. In the neighbourhood of Barnsley, Doncaster, and Rotherham, the ground has sunk, over an area of about 100 square miles, in some cases as much as 5 feet. Of the original bench marks on the principal lines in the two counties, 47 per cent. have been destroyed or have disappeared, 11 per cent. have been moved more than one-tenth of a foot, and only 42 per cent. remain undisturbed, that is, differ less than one-tenth of a foot from the published levels. These changes are no doubt the result of mining operations.

In taking levels of the surface in South Wales, we have frequently observed a difference of a foot, or rather more, in the relative levels of bench marks at short distances apart, proving subsidence of the land in that district.

The level is a delicately made instrument, unable to stand rough usage, and even with care in its use is easily deranged. Before using in the field or the mine the instrument should be thoroughly examined and its adjustment tested.

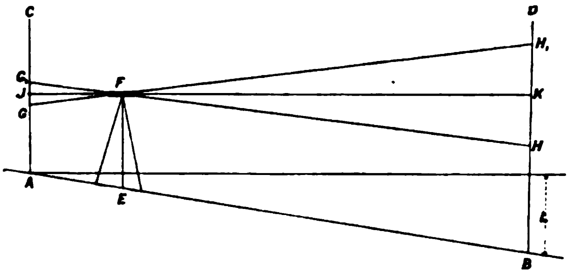

Fig. 449.—Collimating the Level.

The adjustments are for parallax, collimation, and the perpendicularity of the telescope-axis to the vertical axis round which the instrument moves.

Parallax is shown to exist when by moving the eye about on looking through the telescope, the cross-webs in the diaphragm change their position and are flittering and undefined. To correct this error, adjust the eye-piece by drawing out or in the movable eye-piece tube, till the cross-webs appear clear and well defined. Then move the internal tube, by means of the milled-headed focus-screw, until the distant object to be sighted and the intersection of the wires are clearly and sharply defined. It may not always be sufficient to set the eye-glass first and then the object-glass; it may be necessary to adjust the eye-piece again. This adjustment is made continually throughout the time of taking levels, so far as the object-glass is concerned, as it will have to be altered at every change of distance at which the staff is observed.

The adjustment for collimation becomes necessary owing to the diaphragm carrying the cross-webbing getting deranged, the consequence of which is to cause every point bisected by the webs to be either above or below the true level point.

To examine and correct the collimation; on a gently and uniformly sloping ground (Fig. 449), measure off a distance, A B, about 160 feet, and place a levelling-staff, A C and B D, at each end of this distance. Plant the instrument exactly half-way between the staves, and after levelling it, read off the fore- and back-sights.

Whether the collimation be out of adjustment or not, the difference in the readings obtained will be the *true* difference of level.

At E, about 12 feet from A C (or as near to it as it is possible to clearly distinguish the figures and obtain a reading of the staff), set up the instrument in line between the staves and take back- and fore-sights. In Fig. 449, J K is the horizontal line through the axis of the telescope. F G, F H, and F G_1, F H_1, represent the lines of collimation for the back- and fore-sights respectively, the two former if the cross-webs in the diaphragm are too high, and the two latter if they are too low.

The true difference of level of the ground at the two staves or $L = KB - JA$ (1); but $KB = BH, \pm KH$, and $JA = AG, \pm JG$. Substitute these values in equation 1, then $L = BH \pm KH - AG \mp JG$ (2), and by similar triangles $\frac{JG}{JF} = \frac{KH}{FK}$ $\therefore$ $JG = KH \frac{JF}{FK}$. Instead of this last expression take $KH \frac{AE}{EB}$, the ratio being the same, and A E and E B are the distances as actually measured.

Substitute this value of J G in equation 2 and $L = BH \pm KH - AG \mp KH \frac{AE}{EB}$.

By transposing and reducing

$$KH = \frac{L - BH + AG}{\pm 1 \mp \frac{AE}{EB}}$$

Having thus found K H, the diaphragm must be moved by the capstan-headed screws provided for the purpose until the reading on the staff, B D, is $BH \pm KH = BK$; then the line of collimation will be horizontal.

If $BH - AG$ is $\left\{\begin{matrix}\text{greater}\\ \text{less}\end{matrix}\right\}$ than L, use the $\left\{\begin{matrix}\text{lower}\\ \text{upper}\end{matrix}\right\}$ sign in the above formula. To take an example, let $AE = 12$; $EB = 120$; L, or the true difference of level, 1·53; A G 3·21, and B H 5·15.

$$\text{Then } KH = \frac{1{\cdot}53 - 5{\cdot}15 + 3{\cdot}21}{-1 + \frac{12}{120}} = \frac{-{\cdot}41}{-{\cdot}9} = {\cdot}4\dot{5}.$$

Now move the diaphragm till the reading of B D is 5·15 — ·45 = 4·70, when A C will read 3·17, and the instrument is properly collimated.

On being now set up and levelled by using the two pairs of parallel plate-screws, so as to bring the air-bubble to the centre of its run, whatever direction the telescope is turned in, the line of collimation moves round in a horizontal plane, unless the instrument itself has defects in its construction.

With the instrument in this carefully adjusted state, the operator may with confidence commence the necessary observations, the accuracy of which as they are taken will depend only on his own skill in manipulating the instrument and keeping his levelling-book.

There are many methods of keeping a levelling-book, and three different examples are here submitted.

LEVELS OF PROPOSED BRANCH RAILWAY AT ——— COLLIERY, FEBRUARY 10TH, 1886.

Distance in Links.	Sights.			Rise.	Fall.	Reduced Rise.	Remarks.
	Back.	Inter.	Fore.				
0	8·59					100·00	On — Railway 150 feet below the Red Lion Inn.
100		7·27		1·32		101·32	
156		10·15			2·88	98·44	
193		8·76		1·39		99·83	
235		9·58			·82	99·01	
280	12·43		6·04	3·54		102·55	
345		9·78		2·65		105·20	
460		7·56		2·22		107·42	
524		6·82		·74		108·16	
596		8·45			1·63	106·53	
620							Crosses hedge.
694		9·52			1·07	105·46	
753		10·89			1·37	104·09	
806	7·56		11·53		·64	103·45	
920		8·21			·65	102·80	
985		9·40			1·19	101·61	
1,050		7·79		1·61		103·22	
1,124		9·65			1·86	101·36	
1,243		11·33			1·68	99·68	
1,297		12·94			1·61	98·07	
1,365		13·84			·90	97·17	
1,500	6·45		14·00		·16	97·01	
1,582		8·42			1·97	95·04	
1,644		9·81			1·39	93·65	
1,700		11·32			1·51	92·14	
1,800			12·47		1·15	90·99	
1,900	7·84		9·42		1·58	89·41	
	42·87		53·46 42·87	13·47	24·06 13·47		
	Difference	...	10·59		10·59		

and 100 — 10·59 = 89·41, the reduced rise.

The following is the same levelling in another form :—

Distance in Links.	Sights.		Height of Instrument.	Reduced Rise.	Remarks.
	Back.	Fore.			
				100·00	On —— Railway, 150 feet below the Red Lion Inn.
0	8·59		108·59		
100		7·27		101·32	
156		10·15		98·44	
193		8·76		99·83	
235		9·58		99·01	
280	12·43	6·04	114·98	102·55	
65		9·78		105·20	
180		7·56		107·42	
244		6·82		108·16	
316		8·45		106·53	
414		9·52		105·46	At 340 crosses hedge.
473		10·89		104·09	
526	7·56	11·53	111·01	103·45	
114		8·21		102·80	
179		9·40		101·61	
244		7·79		103·22	
318		9·65		101·36	
437		11·33		99·68	
491		12·94		98·07	
559		13·84		97·17	
694	6·45	14·00	103·46	97·01	
82		8·42		95·04	
144		9·81		93·65	
200		11·32		92·14	
300	7·84	12·47	98·83	90·99	
100		9·42		89·41	
1,900	42·87	53·46 42·87			
Difference being a fall of ...		10·59			

Starting point above datum ... 100 feet.
Deduct fall between start and finish 10·59
Finishing point above datum ... 89·41

Another form, requiring only four columns, is as follows:—

Distance Links.	Level Readings.	Reduced Levels.	Remarks.
		108·59	
0	8·59	100·00	On —— Railway, 150 feet below the Red Lion Inn.
100	7·27	101·32	
156	10·15	98·44	
193	8·76	99·83	
235	9·58	99·01	
280	6·04	102·55	
	12·43	114·98	
65	9·78	105·20	
180	7·56	107·42	
244	6·82	108·16	
316	8·45	106·53	
414	9·52	105·46	At 340 crosses hedge.
473	10·89	104·09	
526	11·53	103·45	
	7·56	111·01	
114	8·21	102·80	
179	9·40	101·61	
244	7·79	103·22	
318	9·65	101·36	
437	11·33	99·68	
491	12·94	98·07	
559	13·84	97·17	
694	14·00	97·01	
	6·45	103·46	
82	8·42	95·04	
144	9·81	93·65	Starting point above datum ... ... 100 feet.
200	11·32	92·14	
300	12·47	90·99	Deduct fall between start and finish ... 10·59
	7·84	98·83	Finishing point above datum ... ... 89·41
100	9·42	89·41	

1,900 53·46 Total Fore-sights.
42·87 Total Back-sights.
Difference being a fall of ... 10·59

On comparing the three forms, it will be observed that the first requires eight columns, as compared with six in the second, and four in the last. That the method of filling in the distances is continuous from the starting to the finishing point in the first, whilst in the other two the distances are only continuous for one "plant" of the instrument, and an addition made of these at the end. An advantage claimed for the first form is that it checks every level filled in when added up at the bottom of the page, whilst there is no such check on the intermediate levels in the other methods. Against this it may be fairly claimed for the second and third forms that there is less figuring required, and that they are more simple than the first. It will be observed that the Reduced Levels and Remarks are the same in all three cases; that in the first form the first row of intermediate figures have to be subtracted from the figures under the head of back-sight, or the latter are subtracted from the former, the result being placed as a rise if the intermediate sight is less than the back, and as a fall if more. The next figures in the Inter. column are either deducted from those previously placed in the Inter. column, or else the previous Inter. column figures are subtracted from them, and the result placed as a rise or fall. These rises or falls are added to or subtracted from the figures in the Reduced column all the way down the page. At the bottom of the page, or sooner termination of the levelling, an addition is made of the Back, Fore, Rise and Fall columns, as shown, and the difference between the Back and Fore must agree with the difference between the Rise and Fall shown in the addition, and this difference must also agree with the difference of the Reduced Level at the finish and at the commencement. In the second form, taking the starting point as being 100 feet above the datum line, and the reading for the back-sight being 8·59, it is obvious that the height of the instrument itself must be 100 + 8·59 = 108·59 feet above datum, and placing these figures in a column to itself, called Height of Instrument, every other reading of the staff placed on the chosen stations during the first "plant" are entered as fore-sights, and subtracted from the figures in the Height of Instrument column, the result being the Reduced Level. On fixing the instrument again, we read for the back-sight 12·43, which must be added to the figures 102·55 in the Reduced Level column, and the result placed in the Height of Instrument column. All other readings whilst the instrument remains in its second position are booked as fore-sights, and these are each subtracted from the figures last placed in the Height of Instrument column. For the sake of clearness, the figures that have to be added at the bottom of the page are underlined, because all that are in the Distance and Fore columns are not to be added together, but only those indicated by the underlining which forms a part of those columns. In practice this underlining is unnecessary in the levelling book, because the eye easily rests on the figures to be added together, as they are in a line with the figures in the Back and Height of Instrument columns. Besides the underlined figures in the Fore and Distance columns, the Back column figures must be added at the foot of the page, and the method afterwards explains itself. If the levelling is to be continued further, the totals at the foot of one page are carried forward to the next.

The third form is simply a modification of the second, whereby two of the columns there used are dispensed with. Instead of having two columns for back- and fore-sights, all the level readings are entered in one column, and the first entered after each "plant" of the instrument is a back-sight, and the last the fore-sight. A line is drawn across the first three columns after taking the fore-sight and removing the instrument. These lines serve to distinguish the proper figures to add together at the foot of each page. Thus, only the figures in the Level Readings column immediately over the lines ruled are added together for the total fore-sights, and only those immediately under the lines are added together for the total back-sights, the rows between these extremes not forming a part of either addition. The Height of Instrument column is dispensed with, and the figures really forming the height of the instrument are entered in the Reduced Level column. These are not to be plotted, but are merely used as a basis each time for ascertaining the levels of other stations which have to be plotted. There is little fear of confusing these figures with those in the Reduced columns, which have to be plotted, as the line drawn immediately over them, together with the blank space in the Distance column alongside, sufficiently distinguishes them.

The plotting of the levelling just given will be shown later on, after plotting instruments have been explained.

A very interesting pamphlet on "Hints on Levelling Operations, as applied to the reading of distances by the law of perspective, and the saving thereby of chainmen in a level survey," by W. H. Wells, C.E., was published in 1879 by E. & F. N. Spon, from which the following remarks are drawn.

The image of the staff, or any portion of it, seen in the telescope of a level will diminish as the distance the staff is moved from the telescope increases, according to the laws of perspective.

If when the staff is held, say, 100 feet from the telescope, the portion of the image of the staff contained within the space A B, Fig. 450 (being the horizontal cross web), and C (being the lower edge of the diaphragm), be 1·5c foot, a datum is obtained from which can be ascertained any distance that the staff is afterwards moved, by reading the portion of its image enclosed within the vertical space D C. For instance, if the staff be now moved, and, on directing the telescope to it, it is found that the portion of the staff image contained within the space D C is 2·10; the distance to the staff will be found by a simple proportion sum, thus:—As 1·50 : 2·10 :: 100 : 140; that is, the staff in its new position is 140 feet distant from the telescope.

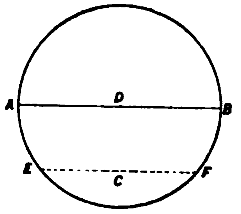

Fig. 450.—Shifting Arrangement in the Diaphragm of a Telescope.

Instead of using the lower edge of the circular diaphragm, a more accurate outline of a portion of the staff image is obtained by fixing a second horizontal web either near the bottom or top of the diaphragm, as shown dotted at E F.

If this second web is a fixture in the diaphragm, the surveyor must ascertain his own datum at a distance of 100 feet, by carefully measuring out that distance on the ground, the staff being placed at one extremity and the telescope at the other, and then reading the portion of the staff image enclosed between the lines A B and E F. A note of this reading is made in the survey book, and all further levelling operations may be proceeded with without using the chain. All the surveyor has to do after reading the staff in the ordinary way is to read and enter in a proper column in his survey book the portion of the staff image contained between the lines A B and E F. The distances can be worked out after the day's operations are finished, and this is done by a series of proportion sums.

The form of book must necessarily be altered to suit the particular method. The following would appear to be quite clear, although it is open to the objection of having many columns.

Datum distance, reading, say, 1·50 of the staff image = 100 feet.

	Distance Readings.				Level Readings.						
	Reading of 2nd cross web.	Back.	Fore.	Distance in feet worked out.	Back.	Inter.	Fore.	Rise.	Fall.	Reduced Rise.	Remarks.
A	10·32	1·73	......	115·33	8·59	......	......	......	......	100·00	....
B	...8·01	·74	......	49·33	......	7·27	......	1·32	......	101·32	....
C	10·34	·19	......	12·66	......	10·15	......	......	2·88	98·44	....
	....	......	......		......	......	......	......	......	... .	....
D	...8·93	......	·17	11·33	......	8·76	......	1·39	......	99·83	....
E	10·17	......	·59	39·33	......	9·58	......	......	·82	99·01	....
F	...7·08	......	1·04	69·33	......	......	6·04	3·54	......	102·55	....
	....	......	......		......	......		......	......	....	....
F	15·20	2·77	......	184·67	12·43	......	......	......	......	....	....
G	11·91	2·13	......	142·00	... ..	9·78	......	2·65	......	105·20	....
H	...8·55	·99	......	66·00	......	7·56	......	2·22	......	107·42	....
J	...7·18	·36	......	24·00	......	6·82	......	·74	......	108·16	....
	....	......	......		......	......	......	......	......	....	....
K	...8·80	......	·35	23·33	......	8·45	......	......	1·63	106·53	....
L	10·84	......	1·32	88·00	......	9·52	......	......	1·07	105·46	....
M	12·80	......	1·91	127·33	......	10·89	......	......	1·37	104·09	....
N	13·96	......	2·43	162·00	......	......	11·53	......	·64	103·45	....

In this form, a line is drawn between the distances at C and J, to show the position of the instrument, and these are ruled in the survey book, whilst the operations are going on ; the distances may afterwards be filled in. A line is also drawn under the Level Readings Fore column at F, as well as in the Distance column, and this distinguishes between the position of the instrument and the last sight taken by it each time of fixing.

The example above shown is the first two "plants" of the level in the Levelling given previously in other forms of keeping the level book, and taking distances in the ordinary way.

It will be noticed that the reading of the second cross web at F, after moving the instrument, is 15·20, and with a 14-foot staff the instrument could not have been used in this way for distance reading, unless it had been placed at a lower level.

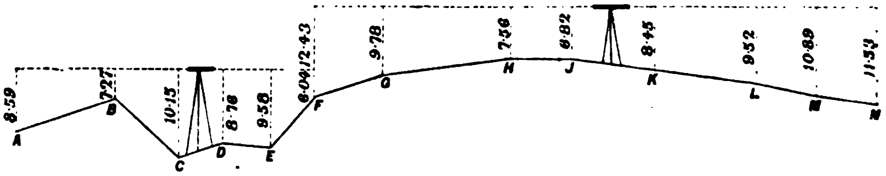

Fig. 451.—STAFF READINGS OBTAINED FROM A LEVELLING.

There are, no doubt, times when shorter sights must necessarily be taken as a consequence of working by this system, but there is a much greater advantage obtained from having no chainage to do, and all the distances read are horizontal ones. Where great accuracy is desired, the distances may be chained and checked by the staff distance readings.

Fig. 451 shows the position of the instrument when the preceding readings were taken.

It is plain that the distances, as worked out in the level book, will be those between the position of the instrument and the different stations on which the staff was held on either side of it, and this must be borne in mind when plotting. If the distance between B and C was required, then 49·33 — 12·66 = 36·67 feet. Similarly the distance between any other two stations may be obtained.

An objection to having the second cross web fixed in the diaphragm at any convenient distance from the usual horizontal cross web is the fact that a series of proportion sums must be worked out before the distances are obtained. But this objection may be removed if the instrument maker fixes to the instrument a properly arranged diaphragm, in which the second cross web can be moved and regulated by the surveyor himself, by means of a thumb-screw placed either above or below the diaphragm of the instrument.

Having this shifting arrangement, the surveyor may bring 1 foot of the staff image in the space between the two cross webs at 100 feet distance, or any other portion convenient to himself. If he make the former arrangement, there will be no proportional sum to find the distances, or only such as can be made mentally in a moment. For instance, if the space between the cross webs read 2·15 after such an adjustment were made, the distance from the staff will be 215 feet; if ·65, it will be 65 feet.

If by means of the regulator two or more whole feet of the staff image is contained between the two horizontal webs at 100 feet distance, more accurate readings will be obtained.

Another advantage of this regulator is in the facility it affords for reading the distance to the other side of a river or other large obstruction, when the staff figures are beyond the limits of vision. In this case, the regulator is adjusted until either the whole staff, or a portion whose length is known, placed on the opposite side of the obstruction, is enclosed between the cross webs.

Without further touching the regulator, a staff is now held on the same side of the obstruction as that on which the instrument is, but 100 feet from it, and the figures enclosed within the horizontal webs then noted. A proportion sum then gives the width of the obstruction. For instance, if the regulator be adjusted to include the whole of a 14-foot staff between the horizontal webs, when the staff is placed on the opposite bank of a river and the reading of the distance between the horizontal webs (the regulator not having been moved) on a staff 100 feet distant from the instrument on the same side of the river is 1·45 foot; the distance across to the opposite side is as 1·45 : 14·00 : : 100 : 965·52 feet.

If in addition to the regulator for adjusting the space between the two horizontal webs, a scale of equal parts be fixed outside the telescope, and a point conveniently attached to the adjusting arrangement be made to move along the scale, the amount of the space may be shown on the scale. When the two webs coincide and there is no space between them, the pointer on the scale would indicate zero. Any other position of the second cross web would be read on the scale of equal parts. With this arrangement no staff would be required on the other side of the river. The telescope may be directed to any suitable object such as a tree or a house, and the space between the horizontal webs carefully adjusted until the object selected is exactly enclosed therein, and the reading on the scale of equal parts noted. The instrument is then moved backwards to a suitable distance, and from this new position the space between the horizontal webs is again made to enclose the same object within it. The reading on the scale of equal parts is noted, and as the distance is increased the reading will be less than that first taken. The distance of the first position to the object is found by the following proportion sum. As the number of equal parts of the scale first read minus the number of equal parts of the scale next read is to the number of equal parts of scale at first observation so is the distance that the telescope was moved to the distance required. For instance, if at the first point of observation 8 equal parts were noted as the space between the horizontal webs enclosing the object, and at the second point of observation 120 feet back from the first point of observation 7 parts were noted, then as 8 — 7 : 8 :: 120 : 960 feet, the distance required.

This last arrangement of the regulator with scale is more suitable for range finding than ordinary surveying operations.

Mr. Wells has suggested to Messrs. Elliott Brothers, of St. Martin's Lane, London, opticians, a contrivance which may be inserted in any levelling telescope by boring a small hole in the tube just behind the diaphragm, which will give a vertical space between the horizontal webs, and adjustable by the surveyor in the diaphragm of the telescope without interfering with existing cross webs and a scale of equal parts attached thereto. The surveyor can obtain this at a small cost, and it will be so constructed that he can himself attach it to any make of levelling telescope.

Levels have now frequently two extra webs placed one on each side of the central horizontal web, fixed at such a distance apart that the image of 10 feet of the staff when placed at 10 chains distance may exactly cut the inner space between the lines. These webs may be used as a means of measuring distances, and so form a check upon chain measurement. A diaphragm webbed in the manner described is shown in Fig. 452.

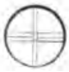

Fig. 452.—Diaphragm of Dumpy Level with Stadia Webs.

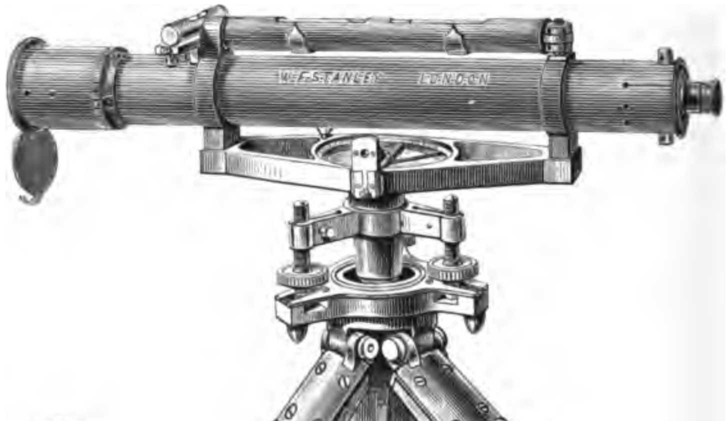

Fig. 453.—Stanley's Model 14-inch Dumpy Level.

In Stanley's improved dumpy-level, shown in Fig. 453, the mounting of the level tube, instead of being placed in a stiff joint at one end, or between rigid clamping-nuts at both ends, has a barrel-fitting at one end which is ground into a parallel hole. This plan admits of circular self-adjustment to the bubble-tube, which the clamping of the nuts can never twist or strain during vertical displacement; and the joint can be made perfectly sound with certainty. This considerably saves the risk of accident to the bubble from expansion by heat and some other conditions. The telescope straps are fitted at their stumps solidly down upon the limb, as shown in Fig. 432. Adjusting-screws are placed under this, as in the dumpy-level described, but the pressure-screw is not employed except in case of accident far away from an optician, when it is there simply ready for use. The limb is framed out into two-edge bars; this gives greater vertical sectional strength and resistance to torsion without increase of weight in the instrument. Where a compass is used, this is included in the frame of the limb, as shown in the engraving. The compass is read with a prism, this being much more convenient and exact than looking down upon the divided circle, the instrument being necessarily placed at nearly the height of the eye. The compass-ring is made of aluminium.

A great improvement in this instrument consists in the vertical axis being fixed directly and firmly upon the limb, and not through a loose screw fitting for separation at this point, as in the ordinary dumpy.

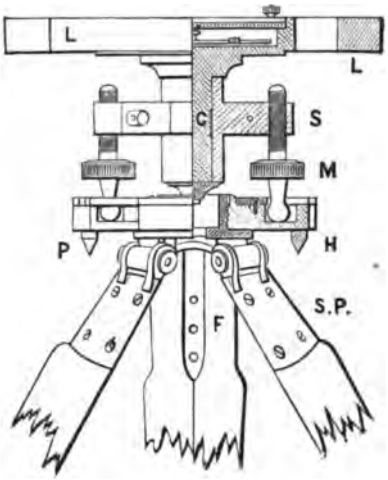

Fig. 454.—Details of Stanley's Dumpy Level. Half Elevation to Left; Half Section to Right.

The setting-up adjustment of the instrument is upon tribrach limbs, with three screws only. These screws can never strain the vertical axis, which in this instrument is somewhat deeper and more firmly made than that of the dumpy. Mr. Stanley uses his own plan of mounting the *tribrach*, the screws being formed with ball ends, which rock upon a flat supporting surface. The necks of the balls are held down by a stiff spring-plate fitted upon them. The general construction of the lower part of this level may be seen from inspection of Fig. 454: L, limb fitted with compass; C, axis in one casting with the limb; S, sprang carrying the socket and supporting the instrument. P H shows the ball-head arrangement to the screws. A central screw in this part detaches the tripod. One point is shown at P, of which there are three to support the level upon a wall or rock in cases where the tripod cannot be used. The tripod head is made more firmly than that of ordinary construction, by extending two wing fittings, from the top of the shovel-plates, SP, as wide apart as possible. The shovel-plates are screwed to the staff by means of a stout nut-plate inside the tripod, F. Those who have experienced how much defective levelling is due to a shaky tripod head will appreciate this precaution.

The ray-shade to the telescope of Stanley's improved dumpy-level has two narrow slits opposite each other at 180°. A zero-line is carried from one slit to a line on the ray-shade fitting when the slits are quite horizontal—Fig. 455. Sight through the slits at zero enables an approximate cross-level to be taken. The edge of the tube of the ray-shade is divided 30° on each side of the zero-line to 2°, so as to take approximate lateral inclines of the surface of the land in levelling. This plan of cross-sighting was originally proposed by Gravatt.

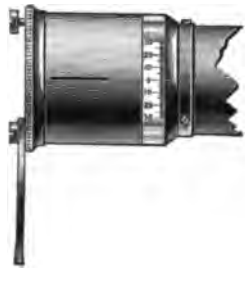

Fig. 455.—Ray Shade.

The most important variation from the telescope of the dumpy-level described is in the diaphragm, where webs or wires of any kind are entirely done away with, and are replaced by a special form of index, shown in Fig. 456. The movable part carrying the opening of the diaphragm is placed in a sliding fitting, as previously described for the dumpy-level. The index which replaces the web is a finely-pointed needle formed of platino-iridium (platinum, ·75; iridium, ·25). This alloy has about the hardness of spring-tempered steel, and is, as far as known, perfectly non-corrosive in air or moisture. A pair of vertical points indicates the position for holding the staff. It will be found by experiment that the point-reading is much more exact than with the web, as iridation due to edge-reading of the web is entirely avoided, as is also the covering of the object intersected by the web due to the angle its thickness subtends upon the staff, which is very palpable at 10 chains distance. The iridium point is sufficiently strong to be kept perfectly clean by touching it occasionally with the point of a camel-hair brush if it appears dusty. By care this point will last in adjustment for as long a period as the level itself remains in use. Upon first impression the point may not appear so fine as a web, but practically it is more exact, as the following exaggerated images will show. Fig. 457 is the image of a division of the staff partially covered by a web, WW'; Fig. 458, that of the magnified image of a point P, brought towards a division for reading. It will be readily observed that the fractional part of the block which the point P cuts is much more easily estimated than that of a similar block covered by a web.

Mr. Stanley usually makes VV', Fig. 456, stadia points, by making the distance of the extreme ends of these subtend an angle equal to 10 feet of the levelling-staff at 10 chains distance, + a constant, by which measurements of the distance of the staff can be taken or checked.

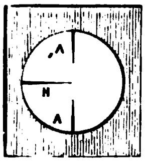

Fig. 456.—Stanley's Platino-Iridium Point Level Stop.

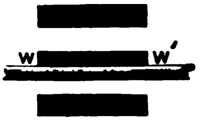

Fig. 457.

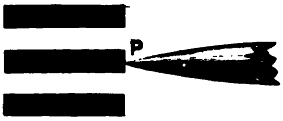

Fig. 458.

Difference of Reading with a Web and a Point, Shown much Magnified.

The setting-up adjustment of a level supported on the tribrach system differs from that already described for the parallel-plate system, particularly in that there are no opposite screws. For the setting-up adjustment the level is at first set lineally with any two of the screws, and the long bubble is adjusted by these screws to about the centre of its run. The cross bubble is then adjusted without moving the level, by the third screw only, which adjusts at right-angles to the first adjustment. When the cross bubble is correct, the long bubble may have been slightly disturbed by the last process; therefore this may be returned to for final adjustment, and the instrument may be rotated to see if it is in good adjustment all round. If not, slight adjustment may be made alternately with the telescope lineal with any two screws, and then with the telescope turned over the remaining screw, so that all adjustments are in this manner finally made by the long level-tube. With this tribrach system the level tube may be much more sensitive than with the parallel-plate system; therefore the adjustment, although not taking longer time, is much more exact. In a large level, the divisions of the tube may represent a displacement of five seconds of arc only, representing less than 1½ inches in the mile displacement of the direction of the telescope.

The foot-screws of the tribrach are adjustable for tightness if they become worn by the cross-screws at any time. The grip should be just sufficient to prevent sensible shake on the screw, which yet should turn without such force as would disturb the instrument in the slightest degree.

In mountainous countries the mountain barometer is used to measure altitudes.

The number of feet of elevation is obtained by subtracting the upper reading from the lower and then multiplying by 900; or the barometrical tables in engineers' pocket books may be used to ascertain the elevation. The surveying aneroid barometer used for the same purpose is furnished with an altitude scale near the periphery of the dial.

For use in the mine the mountain barometer is unsuitable. The surveying aneroid is occasionally used there, but levels cannot be taken by it with the accuracy of the dumpy- or Y-level.

The most common instrument used for measuring angles underground is the *Miner's Dial*, also called the Circumferentor or Compass. It is fixed on three strong legs, which should have joints to unscrew in the middle or at other divisions to suit the height of the seam. It has a magnetic needle turning on an agate centre, and is hung on a very fine hard-pointed pin, so that it may swing freely, and the north end is usually distinguished by a short pin placed through it. It has one or two graduated circles showing degrees and half degrees, and a pair of sights fitted with hairs in an exact line with the centre of the instrument. The best kind of dials have also verniers attached to them which read to minutes (the $\frac{1}{60}$th of a degree), a tightening screw for clamping the lower plate by means of a collar, and a slow-motion-screw working a rack and pinion for giving a slow motion to the sights, by means of which angles are read without the use of the needle, or by what are sometimes called "fast-needle" angles. Some dials have a ball-and-socket joint at the neck, others have parallel plates, and more recently the Hoffman joint has been invented and applied to others. Two spirit levels are placed in the compass box at right angles to each other, to ensure the needle being level when taking an observation. All modern instruments have a vertical arc, or an equivalent arrangement, placed on them for reading the angles of elevation and depression, but in the old form of dial this was wanting.

The original form of miner's dial is shown in Fig. 459. It consists of a compass, divided to single degrees, read by a finely-pointed edge-bar needle mounted on a jewelled cap. The needle has a sliding rider placed upon it, so that it may be carefully balanced to a horizontal position in the locality in which it is used. The divided compass is raised on a step, and the upper surface of the needle is quite level with the division when the compass is horizontal. In erecting the instrument with the needle correctly balanced, the compass may therefore be brought to a horizontal position by the coincidence of the upper surface of the needle with the plane of the divisions, without the necessity of having spirit-levels.

The compass-box is extended in one meridian, north to south, by strong arms that carry a pair of sights which are hinged to turn down to the surface of the cover for portability. The compass-box and arms together are termed the *limb*. The limb of the instrument is mounted upon a *ball-and-socket joint*, to be described. The socket is slotted down on one side to permit the limb to be turned to a vertical position. In this position the level shown on the front of the instrument is used for levelling by means of the sights; this level is not, however, put on all plain dials.

The cover of the compass-box, Fig. 460, is fixed on the box to a given position by a stud and slot. It has an arc divided upon its outer surface, which is centred from a small hole placed near the outer edge. A line from the centre of the hole to the zero of the arc is made perpendicular to the central indices of the sights. A piece of silk or a horse-hair carrying a small plummet is fixed to hang from the hole. By this means, when the limb is turned down in the slot of the socket and the silk or hair stretched by the plummet to allow it to hang in front of the arc, it will then cut the divisions and thus form a reading-index to the arc, giving thereby the vertical angle at which the sights are set.

The instrument is mounted on a simple jointed tripod, to be described. It will be seen by the above description that this instrument is cheaply made, and is not designed for very exact work. It is now giving way to more exact instruments; but it forms the groundwork on which mining survey instruments are most generally constructed. The height of this dial with sights erect is 11 inches; weight, 6 lbs.

Each sight has a slit and a window, above which are also a hole and cross-sight. One sight is shown separately in Fig. 461. The slit A is usually a narrow opening cut through the metal by machinery, and terminating above and below in a hole, for the better sighting an object. The window B is an oblong opening, across which a hair, wire, or a thin plate placed edgewise is fixed. The slit in one sight is placed opposite the window in the other, so that the slit is above the window on one sight and below on the other, and this allows each sight to be used in either direction. The hair or wire is deeply sunk in a groove, and is in the same plane as the centre of the slit. Small holes are pierced through the metal above and below the oblong opening to receive the ends of the hair, which on being passed through are pressed tightly by means of dry, conical pinewood plugs. If a thin plate is used it is soldered into the top and bottom of the window. In sighting, the eye

Fig. 460.

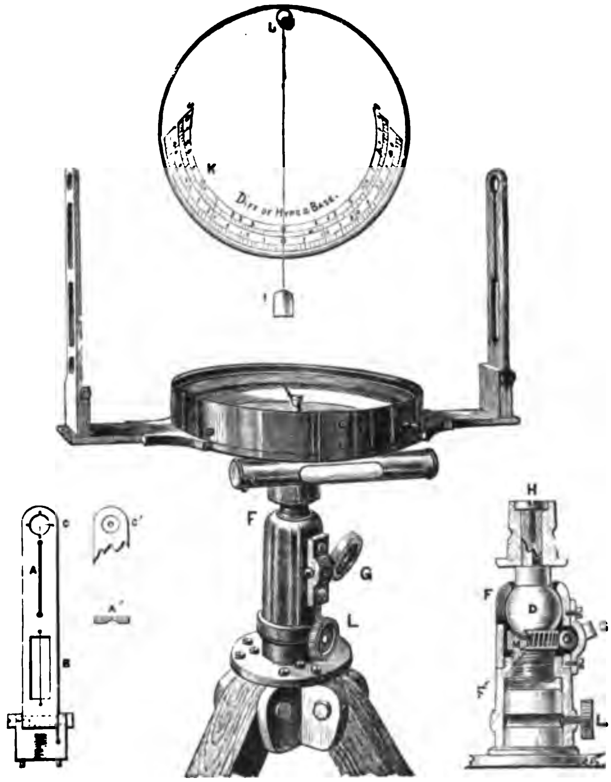

Fig. 461. Fig. 459. Fig. 462.

Fig. 459.—Mining Dial. Fig. 460.—Cover to the Same. Fig. 461.—Sight.
Fig. 462.—Section of Ball and Socket Joint.

is placed near the slit and the instrument turned until the object to be observed appears to be bisected by the hair in the window. The hole C′ and cross-sight C, Fig. 461, allow of more definitely sighting the angles of inclination. For this purpose the eye is placed in front of the hole opposite the central circular window, the sights being then moved vertically until the cross-hair appears to cut the object of observation. The sights are turned down upon the compass-box when not in use. On the completion of a survey, the brass cap is first placed over the compass-box to protect the glass.

An ordinary ball-and-socket joint is shown in section at F, D, in Fig. 462. It enables the observer to adjust the head of the dial in a level position after the legs have been opened to support it. When the clamping-screw G is released the ball D is free in its socket F to move about its centre in any direction. The extent of the motion, however, is limited by the opening at the top of the socket. The lower half of the socket is a plug E, which screws into the part F′ in the socket-piece. The plug is turned upwards by its screw, so as to press on the ball by means of a tangent screw, G, which works in a rack-thread cut in a part of the circumference of the plug, thus forming a screw and cross-screw which clamps the ball firmly.

The upper part of the tripod stand of an ordinary miner's dial is shown in Fig. 463. The legs are made about 1¼ inches in diameter, and at the top are shaped into heads which are fitted without brasswork between the book-plates A, to which they are secured by cross-screws or bolts which form the joint on which the legs move for shifting. A plate carries the book-pieces and also a male-plug centre, to which the dial is fixed by a milled-headed screw. This tripod head remains permanently fixed to the legs. The usual height of the full tripod legs is 5 feet, but each leg is jointed to part in its centre. The usual form of joint is shown in Fig. 464, C being the male screw which is fitted to the woodwork by a socket. When unscrewed the point of the screw is pressed into the surface of the ground to hold the short stand securely. D is a socket-piece screwed over the point to lengthen the leg when the full tripod is required. At the bottom of the lower piece there is a conical metal point to bite the ground when it is set up. Other arrangements of legs are made to suit the various heights of roadways in which they may have to be fixed. Sometimes three joints are used.

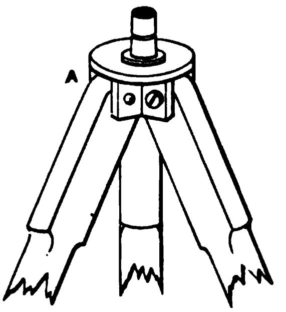

Fig. 463

Fig. 464.

Jointed Tripod Legs of a Miner's Dial.

Fig. 465 shows Davis's improved Hedley Dial. The improvement in this dial consists of an arrangement by which the bearings are taken simultaneously with loose needle and vernier, the latter *automatically* checking the former, thus any error arising from incorrect reading or from any local attraction is detected.

A telescope or upright sights may be used, being interchangeable on the dial, as shown at A B in the drawing. The vernier is placed on the outside circumference of the instrument, in which position it is easier read than when placed inside the compass box. When the telescope is substituted for the upright sights, it more nearly approaches the form of a theodolite, and in consequence gives a result to the surveyor using it closely approximating to the superior instrument, whilst its weight and size give it an advantage over that instrument for most of the surveys required underground. The figure arrangement on the dial is designed to inspire the surveyor with confidence in the work of surveying as it progresses. Those marking the divisions in the compass box circle are formed so that the readings correspond with those of the vernier circle. Both circles are marked from left to right for this purpose.

In place of a side arc for taking vertical angles, as with the old form of Hedley, the improved instrument has a circular box $1\frac{3}{4}$ inches in diameter, the pointer on which traverses a dial-plate divided into 90°. It is very neat and compact, and presents no obstacle to the surveyor when manipulating the screws.

In place of the rigid sights of the plain miner's dial, Hedley's has a rocking-ring which surrounds the compass-box. This rocking-ring moves on its pivots with the sights after the catch is withdrawn. The extent of the movement is indicated in degrees on the vertical arc, which is carried on a prolongation of one of the pivots carrying the rocking-ring. The extreme movement of the rocking-ring does not allow of the sights being tilted sufficiently to take high angles of depression or elevation; if above about 45° they must be taken with another instrument.

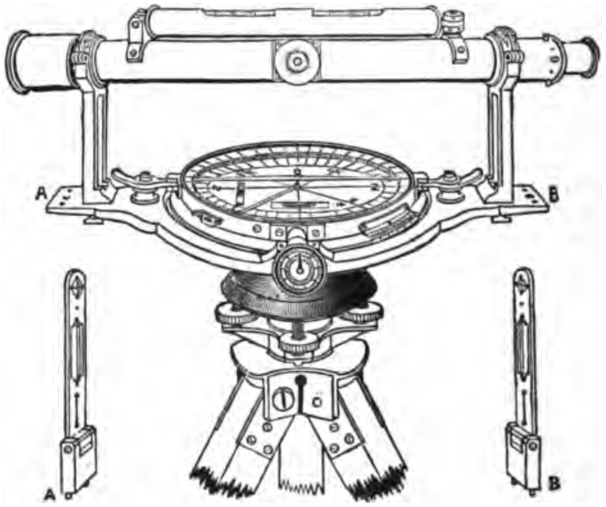

Fig. 465.—Davis's Improved Hedley Dial with Hoffman Joint.

Two spirit-levels are placed inside the compass-box at right-angles to each other, by means of which the instrument is set up horizontally when in use.

Fig. 465 shows the improved Hedley Dial having a Hoffman patent joint. This joint, as specially designed for Miner's Dials without telescopes, is shown in Fig. 466, and the following advantages over the ball and socket are claimed for it.

(1) The plumb line is suspended from D, the actual centre of the dial, which cannot be achieved with the ball and socket.

(2) The rubbing surface is some ten times greater, and therefore the joint is far more rigid.

(3) The joint is manipulated with greater ease and despatch, a slight turn of the milled flange, A, A, from right to left liberates the two concentric balls, B and D, the dial is then levelled up, and a slight turn of the flange from left to right secures the joint.

(4) Only one hand is required to manipulate the joint.

(5) The total height of the improved joint is 3 in., the ordinary ball and socket $3\frac{5}{8}$ in.

(6) The length of the centre is 2⅛ in., that of the ball and socket is barely 1½ in., thus the dial reverses with greater accuracy.

While possessing many and important advantages, the joint is no heavier than the ball and socket.

Fig. 467 shows the Hoffman patent joint for levels or dials having telescopes with four screws for fine adjustment and for clamping joint.

Fig. 468 shows an improved miner's dial as made by Mr. W. F. Stanley, Great Turnstile, Holborn, London, partly from his own design. The telescope is carried in Y supports, and may be removed for sights fitted to the same instrument. The sights are cranked; the angle-piece formed on them increases their distance apart to about 12 inches. These are not shown in the illustration. The horizontal circle is placed on the exterior rim of the box, and has two verniers fitted to it 180° apart for convenience in reading in different positions. The compass is divided upon the upper surface of the step to degrees, and in the same manner on the interior cylindrical surface of the step. The compass is adjusted by clamp and tangent motion. The axis of the instrument is supported upon a ball-and-socket arrangement for roughly bringing the compass to level, and a parallel-plate adjustment for final setting. The ball is fixed by clamping a pair of plates together by thumb-screws. Each plate is hollowed in the centre to hold nearly half the ball. When properly fixed, the instrument is very rigid.

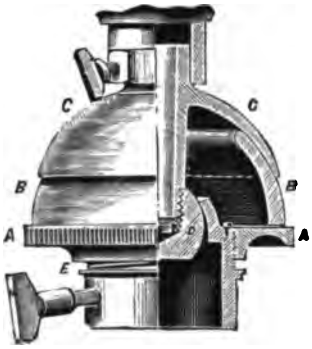

Fig. 466.—Hoffman Joint for Dials with Upright Sights.

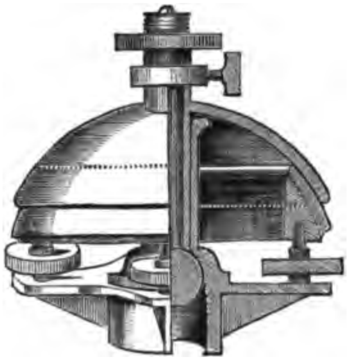

Fig. 467.—Hoffman Joint for Levels or Dials with Telescopes.

Another of Mr. Stanley's plans of clamping, of greater neatness and equal rigidity to that above described, is shown in elevation, Fig. 469. Here the upper half of the socket is screwed down outside the lower half socket by means of four projecting pins. The height of the instrument is about 6½ inches from the tripod, the weight being 11 lbs.

Mr. Stanley's improved form of tripod for dials is shown in Fig. 469. It is adjustable to all heights from 30 to 57 inches. Each leg is formed of two stiff bars of ash, G, shown in detail, Fig. 470, of section about 1¼ inches by ⅝, and a third bar or leg, G′, about 1¼ inches square, which slides between the other two. The sliding surfaces are grooved and tongued together in the solid. Two brass straps S, S′, are fixed near the ends of the bars, the one S′ being firmly soldered to a boss-piece that takes a thumb-screw of sufficient power to hold the leg G′ firmly at any position of extension. It is a rigid stand, which may be adjusted to leave the tripod head nearly vertical upon any inclination of the floor-surface.

In order that the telescope on the miner's dial may be used to range a line at the bottom of a shaft in azimuth with one taken above or to take nearly vertical angles, it is provided with a reflecting cup, Fig. 471, to place over the end of the telescope. The cup is formed of a tube which fits the outer surface of the object-end of the telescope. This is prolonged sufficiently to lock the telescope against revolution by a dowel when the points that are used for index in the diaphragm of the telescope are vertical. The tube is cut in two and hinged to turn up, as shown in two positions, H and H′. When turned up it leaves the tube open for direct vision. A reflector, R, is placed in the cup, and there is an opening below it equal to the full aperture of the telescope. By this means a pair of lights or a line may be sighted up or down a shaft, and the azimuth of its direction be reflected to follow a line by slightly rocking the telescope upon its pivots. This may be done, however, with more refinement if there is a clamp and tangent motion to the vertical arc, which is placed only on first-class instruments.

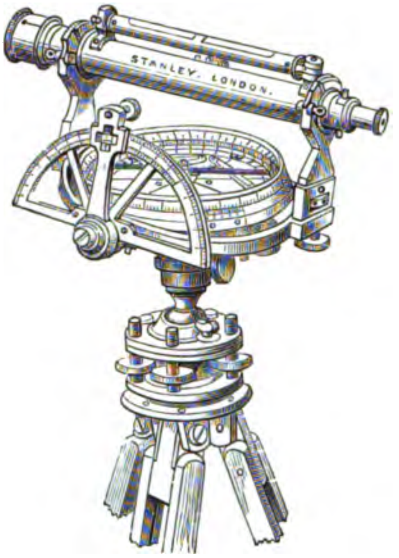
STANLEY. LONDON.

Fig. 468.—Improved Miner's Dial.

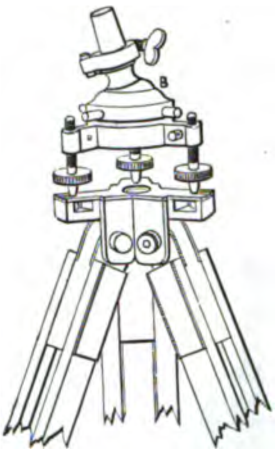

Fig. 469.

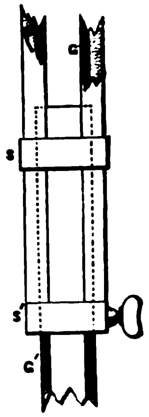

Fig. 470.

Fig. 469.—Stanley's Adjustable Ball Joint and Socket Tribrach Stand. Fig. 470.—Adjustment to Leg of Tripod.

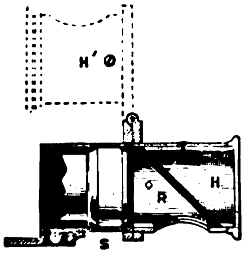

Fig. 471.—Reflecting Cup to Miner's Dial.

The method of using the needle for measuring angles underground consists in placing the instrument in the roadway leading from the shaft or other starting point, having first removed to a distance of 5 yards on either side all iron, which is known to deflect the needle, and having levelled the instrument, the sights of the compass are turned to a light at the starting point, the needle having first been liberated by a spring placed for the purpose. When settled, the needle will point to the magnetic north and the observer has to read and note the number of degrees the sights are lying from magnetic north, and in which quadrant those degrees are. The number of the sight beginning with 1 is entered in the survey book, the magnetic bearing and also the distance in links as measured from the shaft or other starting point to the dial, as well as the angle of inclination (if any) are also noted. The sights are then directed to a forward light, and the reading of horizontal and vertical angles together with the chain distance between the dial and the forward station booked. The dial is then lifted, carried forward, plumbed under the forward station, and a light being held at the station from which the instrument has just been removed, it is sighted through the dial, when the angle reading observed should correspond with the reading last booked; if not, some error has crept in which must be eliminated before proceeding further. If it correspond, the sights of the dial are directed to the forward light and the reading of the needle and any angle of inclination booked, the measurement from the dial to the last sighted station is then made, and noted, the instrument being again moved to the forward station. The process is here repeated and so throughout the length of road it is desired to survey. If a branch road leading from that which has now been surveyed, has also to be included in the survey, the surveyor, after completing the main road survey, must come back to where it branches off, and having made a note in the survey book at the time of advancing along the main road, such as " mark left opposite branch road," he now makes another note to identify the new starting point, thus say, " from 350 in 11," which means that the next sight following this remark was taken from 350 links in the eleventh set or sight going in. The survey may now be continued along this branch road, and afterwards in the same way along any number of other roads leading out of this, or other branches from the main road.

Now supposing it is required to survey from the shaft into the workings with a Hedley dial of ordinary construction, by angles not measured with the needle, but by the vernier, so as to avoid much labour in rail lifting.

First take out the screw which tightly holds the vernier plate to the bottom plate of the dial and prevents the possibility of the upper and lower plates having a separate motion during an ordinary needle survey. This screw will not be replaced till the completion of the survey. Set up the dial at a distance from the shaft, the farther from it the better, so long as a light at the pit bottom can be seen. Great care should be taken that at this first point of setting up the dial, which call A, the rails are taken up, and everything likely to attract the needle of the compass removed to a safe distance.

The instrument having been levelled, the needle is freed, and allowed to steady; after which the sights are turned so that the north end of the needle exactly coincides with the *zero* point of the lower plate of the dial, while at the same time the *zero* point of the vernier plate corresponds with that of the lower plate. The bottom plate is then clamped, by means of the collar tightening screw, the position of which is just above the ball-and-socket joint. The clamping screw of the vernier plate (the position of which is immediately underneath the vernier) is then slackened, and by means of the slow-motion screw (the position of which is underneath the dial plates, and opposite the clamping screw of the vernier plate) the sights and vernier plate are together moved until a light held by an assistant at the centre of the shaft is bisected *through the bottom slit* of the sights. The vernier plate clamping screw is then tightened, the angle read by means of the vernier to 3 minutes, or to 1 minute according to the accuracy of the instrument used, and the reading booked. The vernier plate clamping screw is then slackened, and the slow-motion screw used for bisecting *through the upper slit* of the sights a light which has been taken forward by an assistant along the road to the next station B. The vernier plate clamping screw having been tightened, the angle is read by means of the vernier and booked. Whilst these angles have been taken in the way indicated, the needle has remained on the zero point of the lower plate. The dial is now plumbed, a mark left to look back to, and the needle "locked," after which the instrument is removed to station B, plumbed underneath the mark previously sighted there and set up level. The lower clamping screw must now be slackened, but that of the vernier plate remains tightened, and a light held at A, is bisected *through the bottom slit* of the sights; the lower clamping screw is then tightened. Having proceeded thus far, as a matter of prudence, the angle indicated by the vernier should be again read, and if it does not agree with the angle last booked some slip must have occurred in carrying the instrument or otherwise, and it will be necessary to repeat the operations at A. Suppose the angle indicated by the vernier to correspond with that last booked; if the needle be now freed it will settle over the *zero* point of the lower plate, unless it is subject to some attraction. The vernier plate clamping screw is now slackened, and by means of the slow-motion screw a light held by an assistant at a forward station, C, is bisected *through the upper slit* of the sights; the vernier plate clamping screw tightened and the angle read and booked. The instrument is then removed to C, and the same process gone through as at B, and so on throughout the survey. By continually sighting the back station through the lower slit, and the forward one through the upper, the vernier is kept at the leading end of the dial throughout the survey.

It is desirable that the survey should be constantly checked as it proceeds, and for this purpose, after every few "sets" the rails should be taken up, and as much care exercised to remove all iron, as there was at the first station of the survey. If at any point where this has been done, after the back-sight has been taken, the needle, on being freed, instead of lying on the zero point of the lower plate, should settle a degree or two from it, it is a proof that either a mistake has been made in the survey or that some attraction exists, which, perhaps, may not be easily detected. If the survey be gone over again up to this point, and with the same result, one would be justified in deciding that it was owing to attraction, and not to incorrect surveying, that the needle did not settle on the zero point. It is necessary to say this, because, with all the care that can be taken, at times there is some influence over the needle which cannot be discovered, and the amount of that influence is indicated by the position of the needle with regard to the zero point of the lower plate.

It may be wiser in some instances, instead of surveying in from the shaft to the face to work from the face outwards. In others, a disused road may be taken advantage of as a good starting point. It would be wise to do so, for instance, where the first set from the shaft is a short one, or where iron pipes are laid, or the shaft itself contains a great deal of iron—pumps, &c.

In this method of surveying, besides the advantage of the constant check kept on the survey as it proceeds (by means of the needle), there is a further advantage in all the angles taken being magnetic bearings. They are as readily plotted as are the "sets" of an ordinary survey with the needle, and are as easily converted into northings, southings, eastings and westings. In adopting it, the angles should be written in the book continuously from 0° to 360°; thus an angle which if read by the needle would be south would be booked 180° by "fast needle" as it is called, and N. 50° W. by the needle would be booked 310°. Some surveyors adopt this method of booking their bearings when taken by the needle.

It is a very good practice to take the angles of a survey from true north in preference to magnetic north, as the latter varies continually. Another good practice is to have three sets of legs with lamps to fit on them and to be interchangeable with the dial.

It is the practice of some surveyors when working with "fast needle," instead of taking the angles continuously from magnetic or true north, to take the angle that one part of the road forms with another, irrespective of the magnetic or other bearing of the first part. But this plan is not desirable because by taking the first sight as a base line, all the other angles have to be submitted to calculation, in order to render their plotting easier. Or, as an alternative to calculation, a great waste of time arises in the plotting from having to shift the protractor for every set. In old makes of dials, however, no other course can be pursued, owing to the incomplete appliances on the dials themselves. The method of reducing all the angles of a survey to the original base line is as follows:—

If the angle is less than 180° the difference must be subtracted from the previous angle. If it be more than 180° the excess must be added to the preceding angle. It should be stated, that if in following out this rule the angle as thus reduced should exceed 360°, the excess should be taken as the reduced angle.

Supposing the following angles to have been taken in a survey. (1) base line, (2) 89° 20′, (3) 110° 52′, (4) 192° 35′, (5) 211° 27′, the reduced angles would be (2) 89° 20′ (this of course, being an angle from the base line, remains unaltered), (3) 89° 20′ − (180° − 110° 52′) = 20° 12′, (4) 20° 12′ + (192° 35′ − 180°) = 32° 47′, (5) 32° 47′ + (211° 27′ − 180°) = 64° 14′.

Too much reliance should not be placed on dial surveys, the horizontal angles of which have been taken with the needle. The best form of Miner's Dial approaches to that of a theodolite, and the angle readings obtained are read by means of a vernier. Being more portable than the theodolite, it may be very usefully employed for many purposes underground, but in cases where extreme accuracy is required surveyors use the more perfect instrument in the mine, and rely almost altogether on it for surface work.

Fig. 472 shows Stanley's hanging-dial, which is designed for surveying in tortuous mineral veins where sighting is difficult. It consists of a circle of brass about 6 inches in diameter, $\frac{1}{2}$ inch wide, and $\frac{1}{8}$ inch thick. Two arms are formed on the upper part of this circle, the extension being 12 inches long. At the end of each arm a hook is formed for hanging the instrument upon a cord or chain. Upon the lower part of the circle a fork-piece, with a bearing clipping the circle, is attached with two screws. The fork-piece supports two axes concentric to the vertical circle in which the compass-box is suspended much above its centre of gravity, so that it moves by its own weight in use to a level position. The compass-box carries an index which reads into half-degree graduations upon the interior surface of the vertical circle. By this means the inclination can be taken simultaneously with a bearing.

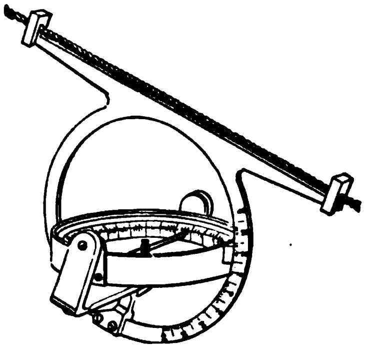

Fig. 472.—Stanley's Hanging Dial.

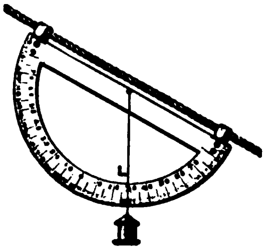

Fig. 473.—Hanging Clinometer.

In Germany two instruments are used simultaneously—the hanging-compass for taking the bearing, and a light hanging clinometer, Fig. 473. In this a thin brass semi-circle, 5 inches in diameter, is graduated to degrees and packed in the cases with the hanging dial. The ends of the semi-circle are formed into hooks for hanging on the line. The plummet has a horse-hair line which cuts the degrees. The clinometer is used only when the hanging dial, Fig. 472, cannot be suspended near the centre of the line, in which case this light semi-circle will cause less deflection of the line and give the inclination approximately.

Where three tripods are used for the underground survey, two lamps are also provided, the flame of either of which will take the position of the vertical axis of the instrument when the lamp is placed upon the tripod previously occupied by it. Stanley's mining survey lamp, Fig. 474, differs from the ordinary form inasmuch as its accuracy does not depend upon the regularity of the flame. A vertical axis is formed under the lamp, which may be inserted in the same fitting on which the mining survey instrument is placed. The lamp is entirely eccentric to the vertical axis in such a manner that a vertical line formed by a wire upon its face may stand central and lineal with the axis. A cross-line is also placed at the same height above the tripod head as the centre of the axis of the telescope or cross-sight. By this means, although the lamp throws its light broadly in one direction only, the cross is a perfectly-defined object, easily picked up and brought to exact bearing to the instrument when placed upon another tripod. The lamp is changed from a fore to a back sight by a turn half round on its axis, which is effected without any displacement of the relative position of the cross in vertical or horizontal directions. For use in fiery mines the lamp is made on the safety-lamp principle.

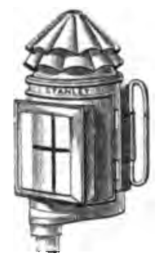
Fig. 474.—Mining Survey Lamp.

An instrument that may be very usefully applied to survey workings after the main roads of the mine have been laid down on the plans from theodolite surveys, more especially if the workings are dry, is Henderson's rapid traverser, Fig. 475.

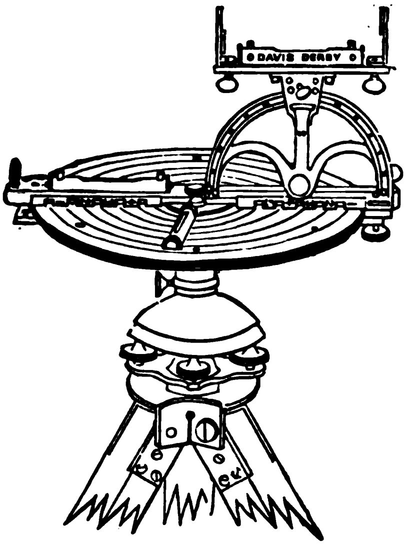

Fig. 475.—Henderson's Rapid Traverser.

The instrument may be described as a circular brass plane-table, having a brass alidade revolving round it, and being mounted on an ordinary dial-stand or tripod. The circular table can be clamped to the stand by the usual screws and the alidade to the table. Upon the face of the table a disc of celluloid, Willesden waterproof paper, or other suitable material, is fixed by pins constructed of any metal other than iron, the alidade being grooved so as to pass easily over them. The disc of celluloid or paper is divided by concentric rings, while the fiducial edge of the alidade is notched so as to afford to each annulus of the disc a certain length of fiducial edge, each length being distinguished by a number. By this arrangement more than one short survey may be completed on one disc, and overcrowding of direction-lines in any one spot is avoided in a long one.

The instrument is set on its stand in the usual way, and both table and alidade are firmly clamped. A back-sight to the starting-point of the survey is then taken. Two additional stands are provided, one of which is fixed at the forward station. Whilst it is being planted in the roadway the bearing of the first line of the survey is marked by ruling two short lines with a finely-pointed pencil on the selected annulus, and numbered or lettered in the space afforded by the notches in the alidade. The alidade is then unclamped and sighted to the forward stand or tripod and again clamped; the direction and number or letter of the second line of the survey are marked on the annulus as before. The traverser is then removed from the stand and fixed on the forward stand, and adjusted so as to sight back to the stand it formerly occupied, and clamped. The alidade is next unclamped, sighted to another forward stand, clamped again, and the direction and number or letter of the third line of the survey duly marked on the annulus, and so on for the remainder of the survey. In an enclosed traverse, if marks be left at the ends of the first line and in returning it be made the last survey line, then the first and last lines as marked on the annulus of the disc should exactly correspond, and a sure check as to the accuracy of the work is immediately afforded. The magnetic meridian is taken at any convenient spot in the survey by means of a trough compass, which is placed against the back edge of the alidade. The alidade and compass are then revolved together until the needle points to the north, when the line of the magnetic meridian is marked on the disc. The lengths of the several lines, offsets, &c., are taken and entered in the surveyor's book in the usual manner. The sights of the alidade are graduated to give angles of depression or elevation up to 25°, and thus the traverser with plain sights can be used for all ordinary surveys. For gradients of more than 25°, or for more accurate reading of dips, a quadrant with sights, or with reversible telescope, is supplied. These accessories are in addition to the graduated sights, and can be applied at pleasure.

In plotting, or laying down the work, the disc of celluloid is removed from its circular table, weighted down on the drawing-paper, and the direction of the survey-lines transferred in due order by means of the ordinary rolling parallel ruler. For future reference the disc itself may be kept, the name and date of the survey being recorded thereon, or the magnetic bearings of the lines may be readily read off, and the same duly entered in the survey book. A convenient method of reading the bearings is to place the disc in the centre of an ordinary cardboard protractor (which has been cut out for the purpose); the protractor is fastened down on a board, and the disc pinned down in its proper position. A central metal spill projects from the board, and by means of an alidade, constructed so as to work on this spill, the bearings are read off with ease and rapidity.

The instrument, which is supplied by Messrs. John Davis & Son, Derby, is not suitable for use in wet mines where it is impossible to fix it clear of water-droppings, which obliterate or render indistinct all the pencil-markings and numbers on the disc.

It is well-known that the magnetic north differs from true astronomical or geographical north. The angle formed between them is called the declination of the needle, and is continually changing. The declination may form an angle either east or west of true north; at the beginning of 1893 the magnetic needle had a mean westerly declination of 17°12′ at Greenwich, but is diminishing about 8′ annually. The rate of diminution varies locally and from year to year. In 1663 there was no declination, and the time will probably come again when there will be none. The declination is not the same in all parts of the world. In travelling west of Greenwich it increases. At present the declination is west in Europe and Africa; east in Asia and the greater part of North and South America. The following table gives an idea of the variation in declination for London approximately for a few dates:—

Year	1580,	Dec.,	11° 36′ E.	Year	1860,	Dec.,	20° 40′ W.
„	1663	„	0°	„	1870	„	20° 19′ W.
„	1700	„	8° 20′ W.	„	1880	„	18° 58′ W.
„	1818	„	25° 41′ W.	„	1888	„	17° 33′ W.
„	1850	„	19° 31′ W.	„	1890	„	17° 9′ W.

It will be seen by the above table that the needle pointed due north in 1663, and that it attained its greatest western declination in 1818, and that it is now losing westerly declination.

Two magnetic needles have seldom exactly the same variation, so that if a needle survey were made of the underground workings, and the surface were surveyed with another needle, an error would arise in connecting the two if proper allowance were not made for the difference of variation in the needles. The needle is subject to periodical variations, the most important being a diurnal fluctuation from east to west. The north end of the needle reaches its most westerly position at about 2 p.m., and its most easterly position at night or early morning, the difference between its extreme positions amounting to 10 or 12 minutes or more in summer, but to much less in winter, and is irregular at times between the extreme seasons. The average position of the needle in making its diurnal movement is attained about 10 a.m., and again about 6 p.m. The fluctuation is at its minimum near the equator and increases in advancing northwards from it. The needle is acted on by magnetic storms, which may seriously deflect it from its ordinary position.

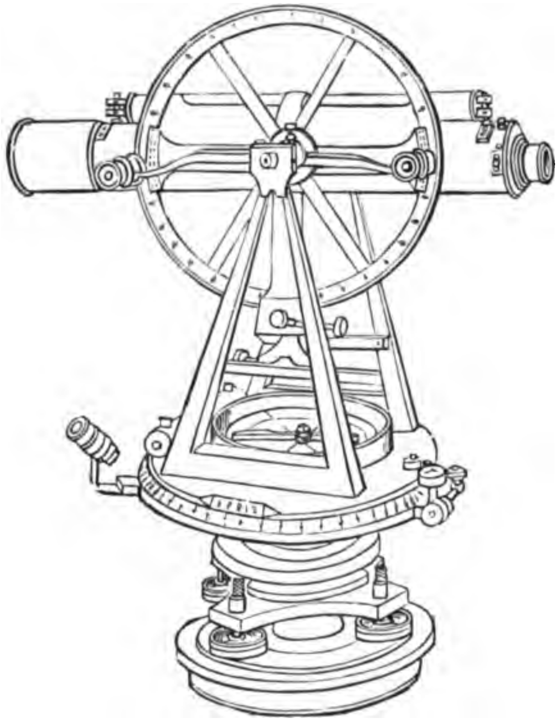

Fig. 476.—Davis's Transit Theodolite.

What is called the dip of the needle is its direction as compared with a vertical line. In order that a needle may assume a horizontal position after being magnetised, it is first carefully balanced before magnetization and is then slightly weighted at its southern extremity. The mean magnetic dip at Greenwich in the year 1886 was 67° 27′ and was diminishing annually at the rate of 1′ 24″. It is uniformly nearly *nil* at the equator, and increases until over one of the magnetic poles, where it becomes vertical. There are two magnetic poles in the northern hemisphere active in directing the needle, one in Siberia, but the most active is about Melville Island; also two in the southern hemisphere, which are supposed to be nearly together, of which the exact position is not ascertained. The plan adopted to balance the needle in opposition to the direction of the dip is to place a rider over it. This clips the needle sufficiently to hold firmly to its place, and yet is sufficiently loose to be moved by the fingers to balance. The rider has to be shifted when the instrument is taken into a country where the dip is different. Where a needle is taken abroad without any rider, it may be balanced by means of a little sealing-wax placed upon its uptending end.

The modern form of theodolite has a telescope which can be moved round the entire circle in the vertical plane, and is called a transit theodolite.

Fig. 476 shows Messrs. Davis & Son's Transit Theodolite, and Fig. 477 that of Hoskold's Miner's Transit Theodolite, as supplied by Messrs. John Davis & Son, All Saints Works, Derby.

In Fig. 476, the telescope, with a spirit level fixed on it, rests on upright supports, which are of such a height above the horizontal circles as to admit of the telescope turning right round on its axis. It is provided with a vertical circle, and by means of verniers and microscopes the angles of elevation or depression are read. The horizontal limb is composed of two circular plates, which fit accurately one upon the other. The lower one is chamfered and graduated at every half or every third of a degree. The upper is called the vernier plate, and has portions of its edge chamfered off, so as to form, with the chamfered edge of the lower plate, continued portions of the same surface, and these chamfered portions of the upper plate are graduated to form the verniers by which the limb is subdivided to single minutes. Usually there are two such verniers placed 180° apart. By means of clamping screws the upper plate may have a motion independent of the lower plate or it may move with it.

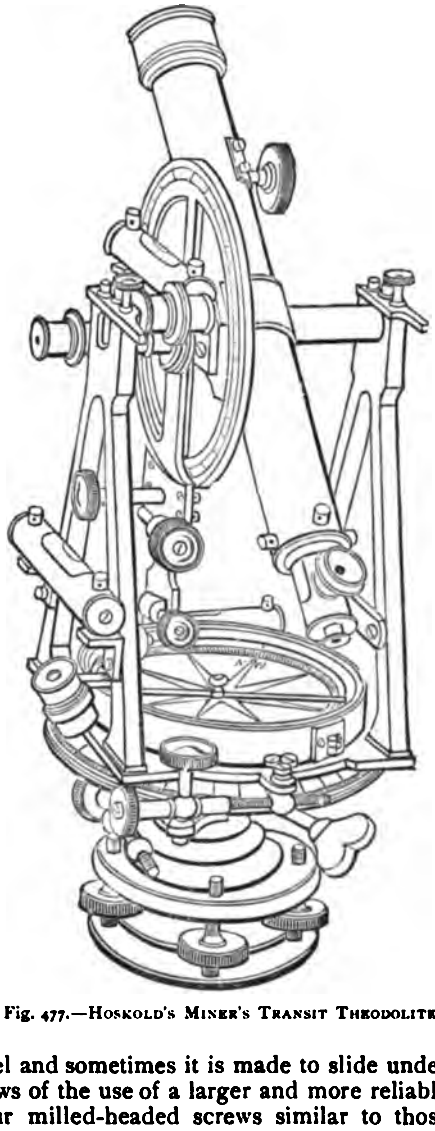
Fig. 477.—Hoskold's Miner's Transit Theodolite.

There is a clamping screw fixed to the vernier plate for the purpose of keeping the two plates together when tightened, or of allowing the upper plate to move whilst the bottom one is fixed. A tangent or slow-motion screw gives the upper plate a slow motion after the clamping screw is fastened. Similarly, a clamping screw tightens the collar below the bottom plate, and a slow-motion screw is placed for moving the whole limb through a small space so as to adjust it more perfectly after tightening the collar. Two spirit levels are placed upon the horizontal limb at right angles to each other and a compass is sometimes placed upon it in the centre and between the supports for the vertical limb. As, however, this only allows of a small compass, the more modern plan is to fix a magnetic needle in an oblong shaped box, and this trough compass is sometimes fixed on the telescope on the side opposite the main spirit level and sometimes it is made to slide under the lower plate. This arrangement allows of the use of a larger and more reliable needle. Two parallel plates with four milled-headed screws similar to those on the spirit level are placed below the lower plate as a means of adjusting the levels accurately before making an observation with the instrument. The vertical limb is divided upon one side every 30 or 20 minutes and two verniers are placed so as to read the vertical angles to single minutes. The vertical limb has a clamping screw and a slow-motion screw; the former on being tightened holds the telescope at the desired inclination, and for more carefully adjusting it in taking an observation, the slow-motion screw moves it through a small space. The instrument is screwed on to the top of a tripod in a manner similar to the spirit level.

The theodolite, like the miner's dial, and the spirit level, may have Hoffman's patent joint attached to it for the more speedy setting up and plumbing.

A good arrangement for underground work is to have three sets of legs for the theodolite and to make these with joints that are all interchangeable and will accommodate themselves to any height. A lamp should rest on these when taking the sights, to be afterwards lifted off and by the substitution of bayonet catches for the ordinary screw the theodolite instead of being screwed on to the tripod head may be dropped into position, to be as easily taken off again after the necessary observation has been made and carried to the forward set of legs, whilst a lamp is placed on the set from which the theodolite has just been lifted.

The method of using the theodolite underground is similar to that described for fast needle surveying.

The instrument after being levelled, and the needle liberated, is clamped by the lower clamping screw while the needle rests on the zero point of the compass box, and the upper limb is securely held to the lower by its clamping screw with the verniers reading respectively 360° and 180°. The vernier plate clamping screw is then slackened, which allows the upper plate, with, of course, the telescope and vertical limb, to move freely round in a horizontal direction whilst the lower limb remains securely clamped. To allow of the telescope being tilted to suit any inclination in the roadway, the clamping screw of the vertical limb must now be liberated. The telescope is then directed to the back sight, and when the light is bisected by the cross webs in the diaphragm of the telescope or nearly so, the vernier plate and vertical limb clamping screws are fastened, and if the object has not been exactly bisected, the slow-motion screw of the vernier plate and that of the vertical limb bring the intersection of the cross webs in the telescope accurately on the object. By means of a microscope placed for the purpose both the horizontal and vertical angles may be read and booked; the latter read from 0° to 90° and the former from 0° to 360°, and may be any intermediate figure. Having booked these readings the vernier plate and vertical limb clamping screws are slackened, the lower one still remaining fast and the telescope directed to the forward station when the vernier plate and vertical limb clamping screws are again fastened, and the horizontal and vertical angles booked.

Having first "locked" the needle the instrument should be plumbed so as to leave a mark under it and removed to the forward station, but much plumbing is saved by the use of three sets of legs. It is here fixed accurately over the mark previously sighted, and levelled, with the vernier plate clamping screw remaining fast in the position it occupied at the previous station, but the lower clamping screw is liberated. When the instrument has been set up, levelled, and plumbed over the forward station, the telescope is directed to the station previously occupied by the instrument and the lower clamping screw fastened. During these operations the vernier plate clamping screw has remained tight, and it is well now to check the reading of the vernier, which at the commencement was set at 360°, to see that it corresponds with the angle last booked. Having done this, the vernier plate clamping screw should be liberated, the telescope turned over on its axis so as to look the opposite way to the back station, and while the lower clamping screw remains firm the telescope is directed to the next forward station, the vernier plate clamping screw is fastened and the angles read and booked. The instrument is then removed to the next forward station and the same process repeated there and so throughout the survey.

If at any stage, after the back sight has been taken, the needle is freed, it should on steadying point accurately to the zero point of the compass box.

The method of traversing on the surface is similar to that underground, but for extensive surveys, the principal points should be determined by a system of triangles proceeding from an accurately measured base of considerable length. The details are afterwards filled in, but it would occupy too much space to fully describe surface surveying here. It may be said in passing that in filling in the details, offsets are taken with a rod or tape from the traverse lines, usually at right angles to them, to all the points required to be shown in the survey, such as houses, hedges, fences, the edges of streams, &c.

Fig. 478 shows a plain theodolite, which, however, is not much used now, it having been superseded by the transit. It differs in construction from the transit by having only a half-vertical circle, with a single vernier, and in the arrangement of the fittings connected with the telescope. There are two verniers on the horizontal circle, but a single microscope is used to read them by passing it in a groove from one vernier to another.

The standards or A-frames in the plain theodolite are firmly attached to the vernier-plate clear of the compass-box. The pivots of the transverse axis rest on coupled bearings on the tops of the standards. The vertical arc is fitted over the transverse axis, which has a turned flange, to which the arc is firmly screwed. The arc is divided to 30′, and reads with a vernier to minutes. The vernier is sometimes read with a microscope jointed on the transverse axis, and sometimes with a loose magnifier, for economy. Difference of hypothenuse and base is generally divided on the back of the arc. The vertical arc has a clamp and tangent placed at the back, not seen in the engraving. Along the bar of the vertical arc a stout plate is attached by screws. From this a pair of Y's with clips and eye-pins support the telescope, as in the Y-level. The diaphragm of the telescope is cross-webbed, and a simple cap is used to cover the object-glass end instead of a ray-shade. The principal level is fixed to collars round the telescope, and rests under it for compactness: it can be adjusted by capstan-headed nuts.

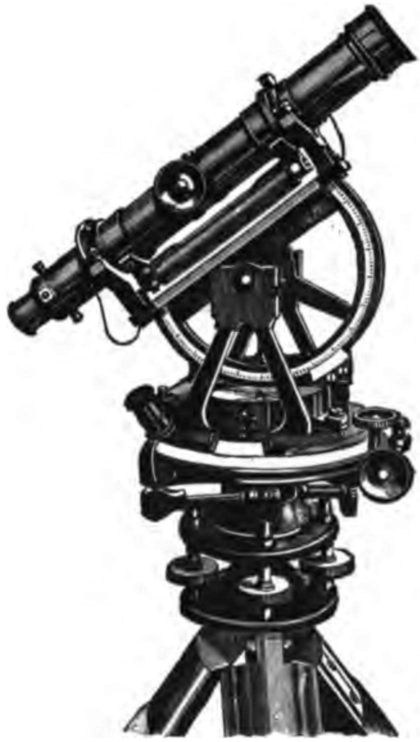

Fig. 478.—5-inch Plain Theodolite.

The vernier-plate carries only one level at right-angles to that of the telescope. The telescope is therefore set to zero by the vertical arc, and the two levels are then used as the pair upon the vernier-plate of the transit.

The telescope of the plain theodolite cannot be moved round an entire circle in a vertical plane, but it may be reversed in the clips for ranging lines.

In Stanley's new model transit theodolite, Fig. 479, the principles of construction are the same as in the ordinary transit theodolite, but the distribution of materials and details is very different. The illustration is of a 5-inch instrument.

It is not built of so many separate castings as the ordinary transit, and every possible casting is shaped out of the solid to the finished form. The vertical axis is made in one casting with the upper framework, the central axis being twice as strong as that of the ordinary transit. The vernier-plate is formed of thin, hard, hammered metal, which is screwed upon the axis. This plate has not to support the superstructure, as in an ordinary theodolite, but has only to hold the two axis bubbles, which are thereby brought distinctly in view, and the clamp and tangent motion, which is also placed conveniently for use upon this plate, in a position where there is less risk of accident than when it is placed upon the outer edge of the limb.

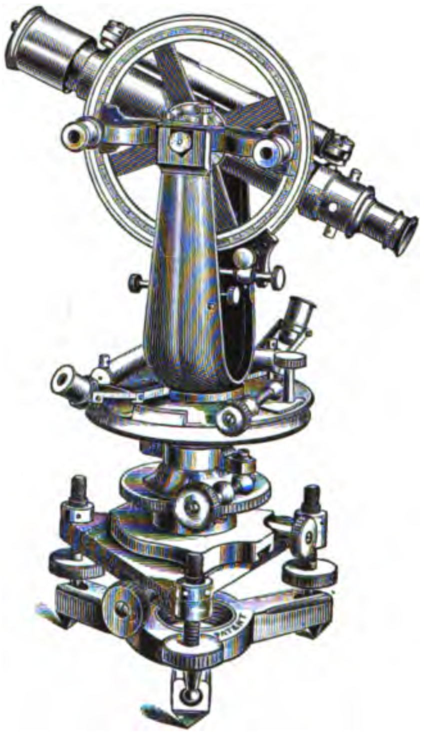

Fig. 479.—STANLEY'S PATENT NEW MODEL THEODOLITE.

The readers to the horizontal limb are double-jointed to turn up against the standards and adjust for reflection. In this manner the readers do not need detachment to place the instrument in its case.

The central axis and the standards are made in one casting in hard gun-metal. They are of light cylindrical section. This construction, although of only about one-half the weight of the A-frame arrangement with its attachments, was found upon testing to have more than double the rigidity in resisting deflection, with certainty of avoiding the accidental occurrence of imperfect fitting of parts.

Instead of carrying the clips on the clipping-arm, in this instrument double clips are placed on the standards: this enables the instrument to be set down in its bearings with more certainty.

The transverse axis is adjusted by one direct-acting vertical screw, and two lateral screws instead of a Y adjustment. This axis, for compactness of the instrument, is made shorter than is usual—4 inches only.

The tubular compass-box is attached under the limb, the magnetic north being set to its zero. This is not shown attached in the drawing.

This theodolite has screw adjustment to the eye-piece, and stadia-points in the telescope. The stadia-points allow of direct subtense measurement of distances by the instrument.

In the place of ordinary circular parallel-plates adjusted by four screws, this instrument has a *tribrach* of new form which is adjustable for centre, see Fig. 481. The upper plate is triangular with rounded corners. The nuts of the foot-screws, Fig. 480, are sawn down on one side so that they spring tightly upon the screws. These nuts are turned conical upon the exterior surface at the lower part, and are cut with a male screw at the upper part. Each nut is fitted into a conical hole B in the upper plate, so that it may be drawn up by a capstan-headed nut D, which closes the slit upon the screw at the same time and thereby tightens it. This plan gives the screws about $\frac{3}{4}$ inch of thread, and permits adjustment for comfortable movement and for wear without any risk of shakiness. The screw has a cap

E to exclude dust. The foot of the screw A has a ball joint which rests on a flat surface, upon which it is sprung tightly by a spring-plate above the ball.

The upper plate of the tribrach is fitted with a stage, Fig. 481. A dovetail-slide B B is fitted upon the base of the stage, adjustable for wear by a slip-piece with two screws at the narrow part of it. The slide is adjusted to position in the direction of its dovetail fitting by the screw B′, so as to move the whole instrument above it for centring in this direction. A slide acting in the same manner, with dovetail fitting-pieces at each end only, A A, moves the slide for an equal distance for centring transverse to the slide B by the milled head A′. The motion given the screws A′ and B′ permits the perfect adjustment of the theodolite over a point on the ground by means of a suspended plummet, after the instrument is set up to nearly its true position by movement of the tripod legs. The range of motion is about ¾ inch—an amount quite sufficient for final adjustment, but which does not materially affect the equilibrium of the instrument upon its rigid tripod, as it has in this case a broad, solid base even in the extreme positions of the slide.

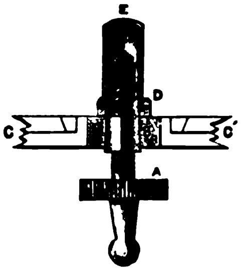

Fig. 480.—Foot-screw Stanley Transit.

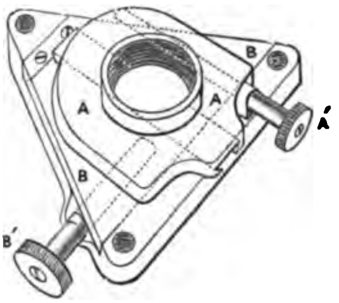

Fig. 481.—Stanley's Tribrach Mechanical Stage.

For examination or adjustment of the theodolite the tripod stand should be at first firmly fixed with its legs extended to an angle of about 70° to the ground, which should be solid and hard. As the telescope has to be brought to the height of the observer's eye, it is well to mention his stature in ordering an instrument. The tripods are made for tall men, and are often awkward and unsteady if the legs are extended to bring the telescope down to the height of a person of short stature. They may always be cut down and re-made. When the tripod is set up, the toes should be each separately pressed down, so that future slips are impossible. This being done, the instrument is taken from its case and grasped firmly by the body part under the horizontal circle, and is placed on the tripod at once, and screwed firmly but not too tightly down upon its bearing-surface. With a transit theodolite the upper part is generally detached and packed separately in its case. Where this is so, after the body part is fixed on the tripod, the cleats on the top of the standards must be opened out, and the upper part of the instrument, lifted by its telescope, be slowly lowered into its bearings, being particular at the same time that the clips under the telescope embrace their stay-piece on the standard. The cleats must then be closed over the pivots. The instrument being set up to position, all levels may be adjusted to the centres of their runs, and every part clamped sufficiently to make the instrument firm, but in no case using violence to produce a strain in any part. The clamps or other fittings are afterwards separately released as they are required for examination or for the adjustment of the separate parts to which they relate.

Examination for Coincidence of Exterior and Interior Vertical Axis.—The theodolite being set up solidly, and all clamps fixed as above described, unclamp the lower or exterior axis clamp and set the vernier-plate levels parallel with opposite pairs of parallel plate-screws if the instrument adjusts on the parallel-plate system, or one level parallel with one pair of foot-screws if it is made on the tribrach system. Now adjust both levels. Turn the instrument half round (180°) and observe if the levels keep the centre of their runs. If they do so they are in adjustment to the exterior axis. If found imperfect, the adjustment by the capstan-heads of the levels are set, by means of the *tommy* or pin which is provided in the instrument-case, for half the error as it appears by the bubble, the other half being given by readjustment of the parallel-plate or tribrach-screws. In these adjustments it is necessary to be particular ***always*** to observe the bubbles after the hands have left the instrument, ***not during the adjustment***, which produces strain upon the instrument. Now tighten the lower clamp and note if the action has disturbed the levels. If these are very sensitive it will do so in a slight degree, but the disturbance should be very small if the clamp is perfect. Now unclamp the vernier-plate and note again if this clamp disturbs the levels: this should also disturb them very little. Now observe the levels if they stand exactly as they did when the exterior axis was unclamped at their present position, and also at right-angles to this. If they remain as before the axes are truly concentric. If they are not so, there is no remedy except at the hands of the maker. The vertical axis to which the above examination applies is considered the most important part of the instrument, and the work should be thoroughly well done; nevertheless, if the levels are very sensitive, which they seldom are, such minute faults may be detected, that a small allowance may be made for imperfection of work, and the instrument still be considered a sound one. In the use of the instrument it is always well, after the circle is set either by the magnetic compass or by sighting a distant point for direction, to clamp the lower clamp and readjust the levels to the vernier-plate. In this way the axis that will afterwards be used for triangulation will be vertical, and small errors due to want of coincidence of axis be eliminated.

Examination of the Azimuthal Level.—This level, which is placed over the telescope, being made of superior sensitiveness to the vernier-plate levels, is much more accurate for adjusting the vertical axis, but much slower in operation for testing. The verniers of the vertical circle should be accurately set to zero, in which position the run of the bubble should exactly agree with those on the vernier-plate when placed parallel with them in any direction. But this level may also be considered by itself. Assuming the circle and verniers correct, or otherwise, it may be reversed over the axis by half-turns in all positions over the parallel-plate or tribrach-screws, and adjusted by the capstan-heads half the error, as before described, for the vernier plate levels.

Examination of the Divisions and Centring.—The vernier-plate being unclamped, the verniers, if two, should be brought approximately to 0° (360°) and 180°, and then the plate be softly clamped. The miscroscopes or readers are then to be set truly radial with the zero reading of the verniers, and the tangent screw adjusted to make one of the readings, say the 360°, exact. The opposite reading, 180°, is then carefully examined, and the error, if any discovered, is due to the imperfection of centring, assuming the division perfect. At this point it is well to record the amount of difference. The same examination is then repeated with the 90° and 270°. In a properly-centred and accurately-divided 5-inch or 6-inch theodolite this difference will not amount to more than 1′ error; in larger instruments proportionally less. From the difficulties at all times of reading the circle correctly from difference of direction, of light, and what is termed personal error, it is well to entirely repeat the examination, turning the instrument half round. It is also well to repeat this examination at what are termed the *half-points*, 45°, 225°, and 135°, 315°. This will sometimes detect the error of centring, if there is one, in its true direction. The purpose of the two verniers is to discover this error. In practice the two readings are always taken, and the mean is considered as the true reading. Where there are a greater number of verniers exactly the same principle is followed, but the mean of three or more readings is taken, which of course assures greater accuracy.

Testing an Instrument for its Stability.—The stability of an instrument will depend principally upon the quality of the workmanship; but the same test will also indicate whether the instrument has been submitted to sufficient wear to need the repair of the optician at any time. For this examination the eye-piece of the telescope requires to be focussed against a piece of white paper held obliquely in front of the object-glass so as to throw a soft, white light into the telescope. After the eye-piece is focussed, any distant point may be taken for a sighting-object upon which to direct the telescope. This point should be focussed by the telescope so that its image falls centrally upon the intersection of the webs. The eye should then be shifted up and down or sideways within the range of clear vision of the webs in the eye-piece to ascertain that there is no parallax, that is, that the adjustments of the eye-piece and the telescope are in true focus upon the webs. This preliminary arrangement being made, which will serve for future examination for other adjustments, all parts of the instrument should be examined to see that the clamps are firmly clamped. The object to be used as an index or sighting-point should be brought by the clamp and tangent motions exactly upon the intersection of the webs as they appear in the telescope, when the following examinations are to be made.

Tripod Head Examination.—The telescope being sighted upon an index point, and all clamps screwed down and the tripod firmly fixed on the ground, take the tripod head of the theodolite in both hands and give it a twist of about a pound pull in one direction; then examine the telescope to see if the index-point is displaced in the telescope. If it still stands correct give a like twist in the opposite direction and again examine the telescope. If it stands these opposite firm twists retaining its position, the stand is good and in good order. If it does not, assuming good construction of stand, the remedy may be found in tightening up all its screws; but if its construction is bad it will not, even after tightening, keep in order. There is no doubt that more defective triangulation is caused by imperfect tripods than from any other cause whatever. A perfect instrument is useless on a bad tripod.

General Examination of Fixed Parts.—The stand being found good by the above process, the general fittings of the instrument may be examined, after clamping all its parts and directing the telescope upon a distant point, by taking a quill pen by its root or pipe and pressing its feathered end upon one side of the eye-piece of the telescope sufficiently to bend the quill, and afterwards examining the telescope to see that the webs are not displaced from the index-point. This may be done first to the right hand and then to the left. If the webs still cut the same object it is clear that the whole of the centres, fittings, clamps, and tangent-screws of the horizontal circle are correct. If there be displacement discovered, the amount of difference between the right and left-handed twists will be the total error due to imperfection of work or wear, as the case may be. In exactly the same manner, but by pressing the eye-piece upwards and downwards, the transit axis and its fittings may be examined. If the instrument is not generally sound enough to bear the above tests, other critical adjustments become necessary.

For the correction of faults that may be included in the above operations, the parts of the instrument must be separately examined.

Examination of the Transit Axis.—The best means of adjusting this axis in a theodolite is by a *striding-level.* Where this is not provided with the instrument—and it is often left out for economy—the axis is generally better left as it is adjusted, in this particular, by the maker. To adjust the transit axis the vernier-plate bubbles are set exactly true by reversing angles of observation. The cleats are opened and the striding-level is mounted above the instrument resting upon the pivots. The telescope is placed exactly over an opposite pair of parallel plate-screws, or parallel with two screws if the base adjustment is on the tribrach principle. The striding-level is then carefully observed and reversed on the pivots. If there is any difference in the run of the level-bubble the transit axis is adjusted by raising or lowering the movable V on which one pivot rests by turning the capstan-nuts until it is quite correct. This adjustment is almost superfluous, as the axis is generally set right at first and is not subject to change.

For larger theodolites of 12 inches and over, the transit axis is much better adjusted by means of an artificial horizon. By the use of this instrument in the northern hemisphere the pole star is first observed directly by the telescope, and then by its reflection from the horizontal surface of clean mercury placed on the ground at 12 feet or so from the instrument. If the star and its reflection cut the webs equally in directing the telescope by movement of its transit axis only from the one to the other, this axis must be truly horizontal. If the vernier-plate be then turned a quarter of a revolution, and the exterior axis a quarter of a revolution, the telescope transited and observation be repeated, the verticality of the principal axis may be adjusted with perfect certainty. The principle axis should be moved one-eighth revolution all round, and the bubble examined at every position to assure perfect adjustment. With the plain theodolite, Everest's, and some others, the transverse axis is fixed to position by the maker.

Examination and Adjustment of Webs.—The ordinary manner of webbing the diaphragm of a theodolite is shown in Fig. 428. Horizontal angles are taken by the upper intersection of the nearest to vertical webs. A single web is placed horizontally for taking vertical angles: it is necessary that this should be nearly true. When the theodolite has its axis vertical, as shown by the vernier-plate bubbles being in the centre of their runs, if one end of the horizontal web be set to cut a small distant object by sight in the telescope, the same object should keep on the web while the tangent-screw of horizontal circle is moved a distance sufficient to traverse it, the hand being always taken from the screw while the observation is made. If it does not, the collimating-screws should be lightly tapped with the back of a penknife in the direction to set it right. These screws have a slot in the body of the telescope, under the loose covering plate, sufficient to permit of this small adjustment.

Adjustment of the Telescope to Vertical Collimation.—The eye-piece is first focussed as before against a piece of white paper held obliquely in front of the object-glass until the webs are sharply seen. The axis of the telescope is then examined for vertical collimation error. The method of doing this has been already described for a telescope placed in Y's, as it is in the Y-level, and the plain theodolite. The only difference with the transit theodolite is, that instead of turning the instrument in its Y's, the telescope is *transited,* as it is termed, over on the transverse axis exactly half a revolution, or 180°, as seen by the vernier reading; and the horizontal circle is moved also half a revolution, so that the telescope points again on the same distant point which is used for an object. If the webs still cut the same point or small object, the webs are in vertical collimation, or truly in the optical axis of the telescope, as regards the vertical direction which this adjustment is intended to secure, presuming the circle has been correctly divided and centred, and the verniers accurately set. If the webs do not cut the same point, half the error is corrected by the top and bottom collimating-screws situate near the eye-piece. The process is repeated until it is exact, being particular to observe, as before mentioned, that there is no parallax. This adjustment cannot be made with the plain theodolite; but the zero of altitude may be examined on both sides of the arc.

For the transit theodolite, adjustment by means of a collimator is much more convenient and exact, as lateral and vertical errors in the position of the webs can be detected in one operation. When a Y-level is at hand this may be used as a collimator if set to solar focus.

Examination for Perpendicularity of Transit Axis and Telescope.—The whole of the lower part of the instrument retaining its position with all clamps firm, open the cleats upon the top of the standards so as to release the transit axis. Now release one of the clip-screws and gently lift the upper part of the instrument out of its bearings. Turn the telescope the reverse way upwards, which will be in this case bubble downwards. Release the clamp and turn the clips to the reverse position of the telescope, and reverse the position of the pivots in their bearings. If the telescope be now directed to the same point as before, if the webs still fall upon it, the telescope adjustment is at right angles to the transit axis.*

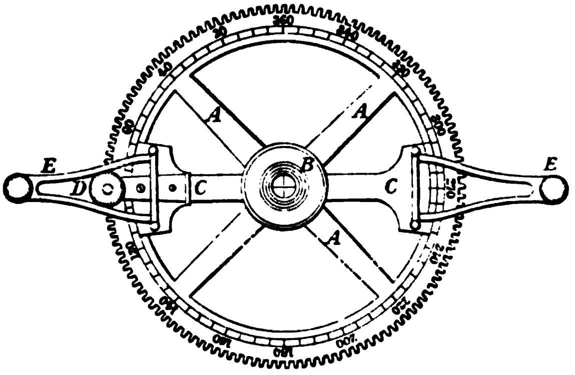

Fig. 482.—Davis's Circular Protractor.

Of the instruments used for plotting, the protractor is one of the chief. A very useful form for working out geometrical problems is the small 6-inch ivory or box-wood protractor usually supplied with a box of mathematical instruments. Besides the divisions of degrees round three of its edges, it has a scale of chords and other scales on it.

For plotting surveys the protractor is usually a complete circle of brass, the centre being marked in some definite manner and the circle divided into 360 degrees, there being in those of large size divisions to show 15 minutes or $\frac{1}{4}$ degrees on them. Another form of protractor is the semi-circular. This is divided into 180° and has a folding arm with a vernier reading to single minutes.

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* We are indebted to Mr. W. F. Stanley for his courtesy in allowing us to make use of the foregoing description of some instruments, taken from his admirable work, ***Surveying and Levelling Instruments***, which deals in a very clear and practical manner with all ordinary surveying and levelling instruments, &c.

The method of using it is to place its straight edge against a T-square, the paper for plotting on having first been secured to a drawing board, and the T-square being moved along the board in parallel lines, forms a meridian to work from. Any desired angle may then be marked from that meridian by the protractor whose straight edge is kept close against the T-square. The best form of protractor is the circular, having folding arms with prickers, and two verniers placed 180° apart, which read to single minutes. Fig. 482 shows this kind of instrument as made in brass, or electrum, by Messrs. John Davis & Son, Derby. A clamping screw secures the verniers at any desired reading, and a slow-motion screw is fixed on it for the more accurate adjustment to odd minutes.

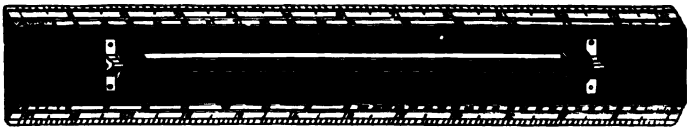

Fig. 483.—Rolling Parallel Rule.

A magnifying glass should be held in the hand whilst adjusting the verniers. On their being placed in any required position each arm is lightly pressed on the paper, a puncture being left by the pricker under each arm, and these punctures should be numbered in accordance with the survey-book. A good plan is to mark only the punctures from one of the arms on first taking off the "sets," and then as a check to mark all the "sets" a second time, using the other arm to form a fresh set of punctures, each of which would be 180° from the first. The parallel rule (Fig. 483), on being placed so that its edge is in a line with the two punctures of any one set, should also be in a line with the centre mark over which the centre of the protractor was first placed, and if not, this indicates an error, to correct which the protractor must be placed on the paper again.

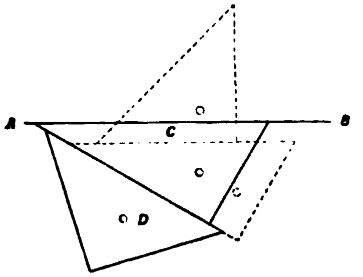

Fig. 484. Drawing Parallel Lines with the Set-squares.

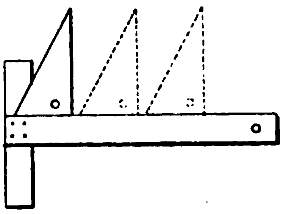

Fig. 485.—Drawing Parallel Lines with a T-square and Set-square.

The best form of parallel rule is made of brass or electrum, and runs on milled wheels. It should be 2 or 3 feet long and sufficiently heavy to roll truly over the paper. Unless a parallel rule is of the very best design and construction it is much better not to use it, but to rely more on a couple of large-sized set-squares which must be very true and accurate for getting parallel lines. The method of using these to obtain parallel lines will be understood from Fig. 484. Let it be required to draw a straight line parallel to A B. Place the edge of one set-square, C, against the line, and place the other set square, D, against the first: hold D firmly down and move C along the edge of D, and a parallel line will be obtained as shown in the dotted line; if lines are required at right angles to that parallel, it is only necessary to hold C and place D on it as shown in the dotted portion of the figure. Parallel lines may also for some purposes be obtained, as shown in Fig. 485, by sliding a set-square along the edge of a T-square.

Whether set-squares or parallel rules are used for the plotting, care should be taken after the line is drawn to run the square or the parallel rule back

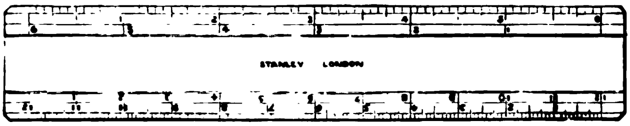

Fig. 486.—FULLY DIVIDED CHAIN SCALE.

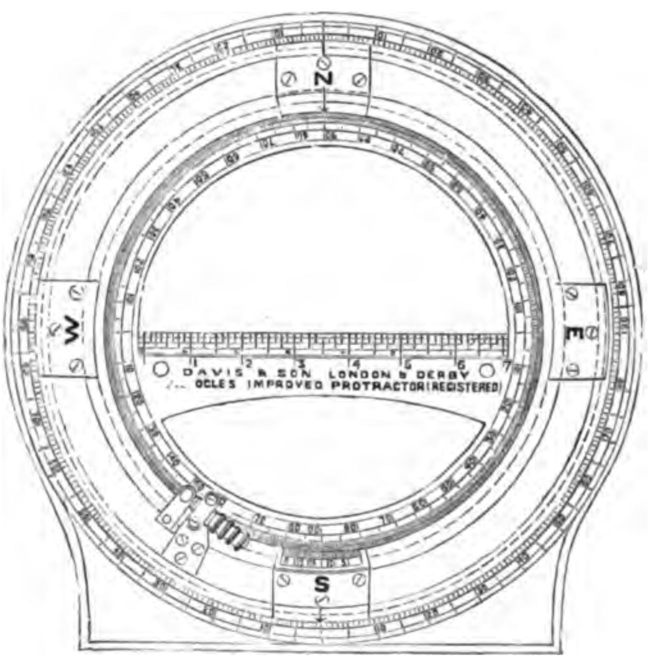
N
S
E
W
DAVIS & SON LONDON & DERBY
OGLES IMPROVED PROTRACTOR (REGISTERED)

Fig. 487.—OGLE'S PROTRACTOR.

to the punctures indicating the set, as a check that the line has been rightly placed.

Scales of equal parts are used for measuring straight lines and laying down distances. A very common scale for colliery plans is two chains to one inch, and the Mines Act, 1887, states that these plans shall not be kept on a smaller scale than that of the Ordnance Survey of 25 inches to the mile, or $\frac{1}{2500}$. A good form is to have a scale 2 feet long, of box-wood or ivory, and marked in links on one side of the scale and correspondingly in feet on the other. Shorter scales may be more useful if long lengths have not to be measured. A small offset scale about 2 inches long, similarly marked, is useful for plotting the offsets of surface surveys.

Mr. Stanley's scales are divided by an engine of his own invention. All scales are readily made by it to any standard, English or foreign, with equal accuracy. The boxwood used has been seasoned from 7 to 15 years, the ivory 5 to 8 years.

Fig. 486 shows a chain scale, fully divided flat section, containing any two ordinary scales. The divisions are marked along the bevelled edges to bring them nearer the drawing-paper and so render easier the reading of measurements.

Messrs. John Davis & Son, Derby, make a new form of scale, consisting of a strip of well-seasoned mahogany, the two bevelled edges of which are faced with perfectly white celluloid, upon which the graduations of the scale are engraved. The divisions are exceedingly clear and readily distinguished. It is stated that the celluloid facings retain their colour, whereas ivory becomes discoloured with age, and boxwood, if not kept clean in use, is difficult to read owing to ink or colour marks obliterating the divisions. The celluloid scales have the further advantage of being inexpensive.

Ogle's protractor, shown in Fig. 487, is a very convenient instrument for plotting the surveys on plans which are not too large to allow a T-square sliding over their surface. It has an outer frame with a true edge for working on the T-square; an intermediate ring free to rotate within this frame; and an inner ring or protractor proper, which is also free to rotate within the intermediate ring. Its cross-bar is divided as a universal scale, so that with it the use of a parallel rule and scale are not necessary. In using this protractor the N and S points on the intermediate ring are placed parallel with the meridian line of the plan after allowance has been made for such alteration as is due to change in the declination of the needle, and clamped in position whilst the working edge is in close contact with the T-square, where it remains until the completion of the plotting. The inner ring or protractor is rotated until the particular reading in regard to the N and S points on the intermediate ring is obtained and the line of the correct length is drawn by means of the scale on the cross-bar. The T-square is moved along the edge of the drawing-board, carrying the protractor to the desired position on the plan. The length of line which can be drawn for any bearing is of course limited to the diameter of the protractor, but the line may be lengthened by changing its position.

The least expensive circular protractor is made of paper, without arms, and with the divisions engine-divided. This, as well as that made of celluloid, may be placed in a proper position on the drawing-paper, so as to allow a rolling parallel rule to pass over them. When the paper protractor is used in this way it is not necessary to prick off and number the set lines of a survey, as the bearings may be transferred direct from the protractor to a different position on the plan-sheet, when required. If, however, weights have to be placed on the protractor to prevent it from shifting, they must be constantly moved to allow the parallel rule to be moved from place to place across the protractor. If drawing-pins are used to hold the paper protractor to the plan-sheet, the trouble of moving weights about during the plotting is removed; but the plan-sheet is left punctured by the pin-holes. This is objectionable in any plan-sheet, and certainly is not permissible in any of importance.

The booking of surveys should be clear, so that they can be re-plotted, if necessary, years afterwards. Careless booking is much to be deprecated, as great importance may hereafter attach to a survey. Instead of first writing at the beginning of the book and working down the first page, and then passing on to the top of the second page, the surveyor begins to fill in a survey at the end of the book and on the last line of the last page, writing upwards to the top and then passing to the bottom of the last page but one and writing up it. There are different methods of ruling a book, but the simplest appears to have one or two lines through the middle of each page, which represent the chain line as the survey proceeds. The following is given as an example of an underground survey, showing the method of keeping the book and of plotting the survey:—

Face
2 20
198° 22'
(9)

5 6
197° 15'
(8)
From 1 26 in 5

Face
6 0
345° 58'
(7)

7 8
303° 18'
(6)

2 22
Road 1 26
297° 10'
(5)

5 0
265° 47'
(4)

4 26 End of Engine Plane
257° 19'
Fall (3) ∠ 10°

7 39
255° 49'
Fall (2) ∠ 9°

5 08
256° 33'
Fall (1) ∠ 10°

transit theodolite.
19° 30' W. Commenced at No. 1 shaft, the survey being made with a 5-inch
1886. All horizontal angles are from true north, the magnetic declination being
plane and into the face of the workings on the Four feet seam. February 12th,
Survey made at the —— Colliery from the No. 1 shaft along the engine-

The number indicating the set is written on the central line and enclosed with a ring, whilst the vertical angle is written on a line with it, in the example given, showing at a glance that the engine plane dips from the shaft about 6 inches per yard. The horizontal angle is then booked, and this shows that the first part of the road makes an angle with the true north of 256° 33', and the length of the part of the road which makes this angle is 508 links, and so on with the other parts of the road. Had a Miner's Dial been used and needle bearings taken, if its magnetic variation agreed with the theodolite needle, the horizontal angles would have read as follows:—(1) $360^\circ - (256^\circ\,33' + 19^\circ\,30') = 83^\circ\,57'$, or N $83^\circ\,57'$ W; (2) $360^\circ - (255^\circ\,49' + 19^\circ\,30') =$ N $84^\circ\,41'$ W; (3) $360^\circ - (257^\circ\,19' + 19^\circ\,30') =$ N $83^\circ\,11'$ W; (4) $360^\circ - (265^\circ\,47' + 19^\circ\,30') =$ N $74^\circ\,43'$ W; (5) $360^\circ - (297^\circ\,10' + 19^\circ\cdot 30') =$ N $43^\circ\,20'$ W; (6) $360^\circ - (303^\circ\,18' + 19^\circ\,30') =$ N $37^\circ\,12'$ W; (7) $(345^\circ\,58' + 19^\circ\,30') - 360^\circ =$ N $5^\circ\,28'$ E; (8) $(197^\circ\,15' + 19^\circ\,30') - 180^\circ =$ S $36^\circ\,45'$ W; (9) $(198^\circ\,22' + 19^\circ\,30') - 180^\circ =$ S $37^\circ\,52'$ W.

The paper used for permanent colliery plans is worthy of more attention than it receives; for after all possible care has been taken with it, it is subject to a certain amount of shrinkage. The paper should be well mounted and be properly seasoned before it is used. It is much better to show a true north line on these plans rather than a magnetic north, for the true north will not change, and either the instrument may be adjusted from time to time to read angles from the true north when making a survey, or the protractor may be placed by the true north point in such a way as to make allowance for the declination at the time of survey. A good plan is to have a number of faint lines dividing the plan into squares of, say, 10 acres area each; that would be 10 chains to each side of the square. One of the series of lines would represent a true north and south line, the other series east and west lines. These lines should be carefully put on with a long steel straight-edge, divided into inches and tenths. The squares will assist in any computation of quantities that may afterwards be required, and also any plotting that may be done either from a computation of the northings, southings, eastings, and westings of a survey, or in the ordinary way.

Having a plan-sheet of suitable size divided by lines as indicated, the surface surveys should be plotted on it, the boundary lines and pits inked in, and a distinctive marginal colour run round the different royalties which form the taking. The underground surveys of one seam should be afterwards plotted on this same sheet. If more than one seam is being worked it is better to have a separate plan-sheet for each seam rather than attempt to show the workings of two or three different seams on one plan as is sometimes done. To save plotting the surface survey again, a careful tracing may be made of the surface lines and transferred to one or more sheets. In this way there may be as many plan-sheets as there are seams of coal working, all having the same boundary lines and relative position of pits, but with the underground workings shown clearly for each seam.

There is not always sufficient care taken in making or extending colliery plans. The most complete not only show the underground workings and roads, but also, by distinctly coloured lines, the position of all faults, with the direction and amount of their throw; air-crossings, inclines, engine-planes, main doors, the position of engines and boilers; the direction of main and minor currents of air, and the flow along water-courses. Old workings and drowned wastes should have special colours on them, and in districts much disturbed by faults the plans should be supplemented by sections, the lines of which are indicated on the plans. Few plans, however, have all possible information on them. This may be excusable in mines in which the arrangements are in an experimental stage and subject to constant alteration; in a large number, however, which have arrangements developed and are not subject to much disturbance, these particulars are neglected. Again, contour lines should be shown, or the levels of the roadways marked in some clear manner, as these are always useful and may be most valuable hereafter. Contour lines are simply lines joining places of the same altitude. Thus it may be decided to show a contour line at a level of 50 feet above the pit bottom, another at 100 feet, another at 150 feet, and every other 50 feet, or contour lines may be marked at every change of 10 feet or other number according to circumstances. By levelling underground, these points of equal altitude in the different roads are ascertained, and afterwards connected on the plan by faint dotted lines. Or, if contour lines are not marked on the plan, the levels may be written neatly and clearly about the plan on all main roads. They show at a glance the probabilities of water taking certain directions; whether the coal has to be hauled up or downhill to the shaft, and direct the manager's attention to parts of the road which probably may be improved. A record of this sort on one seam becomes a very valuable guide in directing the operations on another, the workings of which may follow over the same ground, and in the case of abandoned collieries, plans containing such information are of the utmost assistance to a neighbouring venture.

The Mines Act, 1887, renders it not only necessary to survey the workings and extend the plans in accordance therewith at least every 3 months, but also to show the general direction and rate of dip of the strata, together with a section of the strata sunk through, if the last be reasonably practicable.

To plot the survey by the protractor and parallel rule, place the protractor as stated on one of the lines representing the true north and south meridian of the plan; if the angles of the survey were taken from the true north make use of the parallel meridian line which is most convenient. The protractor would be placed with its zero point on the north end of the line and with its centre on a continuation of the same line where it crosses one of the parallel lines at right angles to the meridian. A continuation of the same meridian line southwards will coincide with 180° on the protractor. The numbers of the sets should be then pricked off, each being twice marked, supposing a good protractor with folding arms to be used. A magnifying glass is held in one hand (the survey-book lying open on the plan), whilst with the other the vernier of the protractor is set at the angles, corresponding with those of the survey-book. The best plan is to mark all these off with only one of the prickers the first time, and then to go through them either backwards or forwards a second time and use the pricker on the opposite folding arm as before remarked. Should any error arise it will be discovered from the fact that a straight line between the two prick marks of the same set ought to cross through the centre point of the protractor, which, in laying on the plan as stated, should be at the intersection of two of the lines referred to. The parallel rule is then used and placed in a line with the two punctures representing the first set, and taken in a truly parallel course by means of its rollers across the paper to the point from which on the plan the first set was taken. A pencil line is carefully ruled of a reasonable length; by looking at the survey-book some judgment of this may be formed, but it is better to draw it long enough as the extra length not required may be rubbed out afterwards. The parallel rule is now carefully moved back to the punctures at which it was first set, and if it corresponds with these the line will be truly drawn. If not, take the line out with india-rubber and repeat the operation. The operator must be careful not only to place the lines truly parallel with the set-marks but also not to reverse any of them and rule them on the plan in the opposite direction to which they should be. The scale may be now laid down to the line drawn and the length taken from the survey-book having first been subjected to the proper deduction for the vertical angle (if any) the horizontal length is marked on the line by a puncture, a pencil ring made round it, the number of the set written in pencil near the ring, and the excess of the line beyond the ring is taken out with india-rubber. The parallel rule is then placed truly by the punctures representing the next set and brought in a carefully parallel line to the ring at the end of the line forming the first set. A pencil line is again ruled from this ring in the proper direction and the method of procedure corresponds with that described for the first set, and so on throughout the survey, taking care that if a remark "from —— in —— set" appears in the survey-book that the plotting proceed from the point indicated. The pencil used should be finely pointed and fairly hard—an H.B. is very suitable for the purpose. As these pencil lines represent the centre of the road (it being impossible to fix a surveying instrument right in the side of the road) it is customary, instead of inking in a single line on the plan representing the centre line, to rule two lines representing the sides of road, and these should be inked in at equal distances from the centre line which is shown in the plotting. Indian ink and a proper drawing pen are used for the purpose of inking in. Fig. 488 shows the plan of the survey before alluded to.

It is highly desirable that, wherever practicable, surveys should have proof of their accuracy by what is called "ties." A traverse survey commencing at some well-defined point on the surface may always have a "tie," that is, it may be closed at the starting point, and although it entails a little more labour where a doubling back and returning along a different route is resorted to, it may be well worth the labour. If such a survey be accurately made, on plotting it, the end of the last "set" line will exactly reach to, and coincide with, the beginning of number one "set" line. Similarly there may, where convenient, be intermediate checks or ties, taken from any one point of the main lines to others, and these checks on plotting an accurately made survey strengthen the proof of accuracy in the whole. If a slip has occurred in conducting the survey, on plotting, it will be discovered by the "set" lines not truly fitting where they should, and the error now known to exist must be discovered or a fresh survey made.

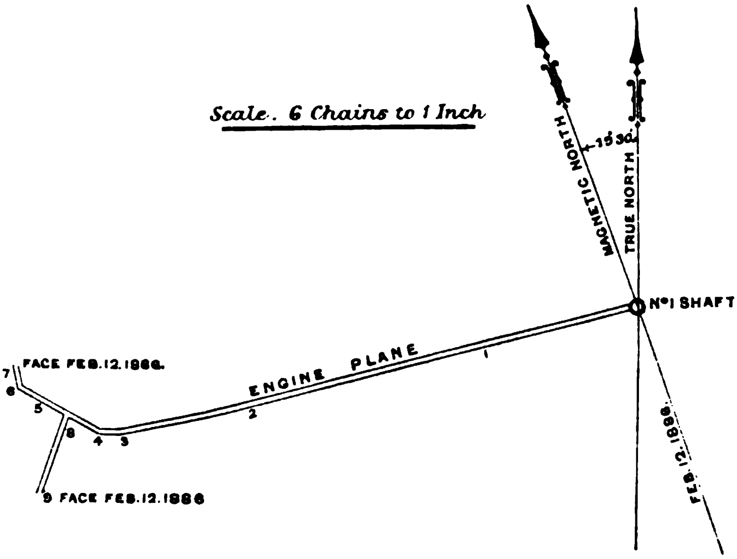

Fig. 488.—The Plotting of a Survey.

Again, a "tied in" survey may be tested by Euclid, for, "The sum of all the interior angles of any rectilinear figure, together with 4 right angles, are equal to twice as many right angles as the figure has sides." This is not so thorough a test as the plotting, because it checks only the angles taken and not the chainage, still it may sometimes be useful. For instance, it may be applied where the plotting proved an inaccurate survey, and if the angles are found to be correct as surveyed, the error or errors must be in the chainage.

Underground workings do not afford the same facilities to "tie" surveys as are obtained on the surface, for however tortuous their course, the roadways must be traversed. Frequently, however, a "tie" can be obtained by following a roadway from its junction with another road until by means of branches the first-mentioned junction is returned to.

Advantage should be taken of these checks wherever they can be obtained.

Fig. 489 shows an underground survey "tied in" in the manner described.

	166 to 0 in 1	
22	19° 5′	(166)
21	21°	(96)
20	14° 45′	(60)
19	5° 40′	(105½)
18	342°	(61)
17	332° 55′	(217)
16	350° 55′	(56½)
15	359° 15′	(239½)
14	20° 25′	(41)
13	31° 35′	(78)
12	249°	(221)
11	247° 47′	(157½)
10	240° 20′	(103)
9	214° 33′	(29½)
8	176° 55′	(243)
7	163° 12′	(102½)
6	178° 58′	(127)
5	170° 11′	(261)
4	134° 27′	(35)
3	113° 26′	(67)
2	99° 51′	(84½)
1	109° 25′	(169)

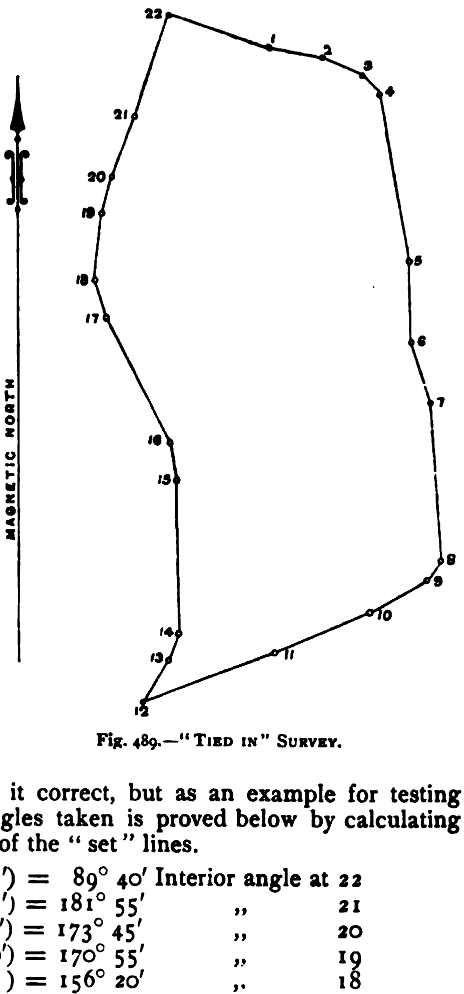

Fig. 489.—"Tied in" Survey.

The plotting of this survey proves it correct, but as an example for testing other surveys, the accuracy of the angles taken is proved below by calculating all the interior angles at the numbers of the "set" lines.

$180^\circ - (109^\circ\ 25' - 19^\circ\ 5') = 89^\circ\ 40'$ Interior angle at 22
$180^\circ + (21^\circ - 19^\circ\ 5') = 181^\circ\ 55'$,, 21
$180^\circ - (21^\circ - 14^\circ\ 45') = 173^\circ\ 45'$,, 20
$180^\circ - (14^\circ\ 45' - 5^\circ\ 40') = 170^\circ\ 55'$,, 19
$180^\circ - (365^\circ\ 40' - 342^\circ) = 156^\circ\ 20'$,, 18
$180^\circ - (342^\circ - 332^\circ\ 55') = 170^\circ\ 55'$,, 17
$180^\circ + (350^\circ\ 55' - 332^\circ\ 55') = 198^\circ$,, 16
$180^\circ + (359^\circ\ 15' - 350^\circ\ 55') = 188^\circ\ 20'$,, 15
$180^\circ + (380^\circ\ 25' - 359^\circ\ 15') = 201^\circ\ 10'$,, 14
$180^\circ + (31^\circ\ 35' - 20^\circ\ 25') = 191^\circ\ 10'$,, 13
$249^\circ - (180^\circ + 31^\circ\ 35') = 37^\circ\ 25'$,, 12
$180^\circ - (249^\circ - 247^\circ\ 47') = 178^\circ\ 47'$,, 11
$180^\circ - (247^\circ\ 47' - 240^\circ\ 20') = 172^\circ\ 33'$,, 10
$180^\circ - (240^\circ\ 20' - 214^\circ\ 33') = 154^\circ\ 13'$,, 9

$180^\circ - (214^\circ\,33' - 176^\circ\,55') = 142^\circ\,22'$ Interior angle at 8
$180^\circ - (176^\circ\,55' - 163^\circ\,12') = 166^\circ\,17'$,, 7
$180^\circ + (178^\circ\,58' - 163^\circ\,12') = 195^\circ\,46'$,, 6
$180^\circ - (178^\circ\,58' - 170^\circ\,11') = 171^\circ\,13'$,, 5
$180^\circ - (170^\circ\,11' - 134^\circ\,27') = 144^\circ\,16'$,, 4
$180^\circ - (134^\circ\,27' - 113^\circ\,26') = 158^\circ\,59'$,, 3
$180^\circ - (113^\circ\,26' - 99^\circ\,51') = 166^\circ\,25'$,, 2
$180^\circ + (109^\circ\,25' - 99^\circ\,51') = 189^\circ\,34'$,, 1

Total Interior angles $3{,}600^\circ$
Add 4 Right angles $90^\circ \times 4 = 360^\circ$

Total $3{,}960^\circ$

No. of sides.
$22 \times (90 \times 2) = 3{,}960^\circ$

In plotting sections of considerable length the horizontal distances are not laid down on as large a scale as is necessary for the vertical heights above datum. A horizontal scale of 4 inches to the mile with one of 100 feet to 1 inch for the vertical scale is frequently used. In the following plotting of the levelling

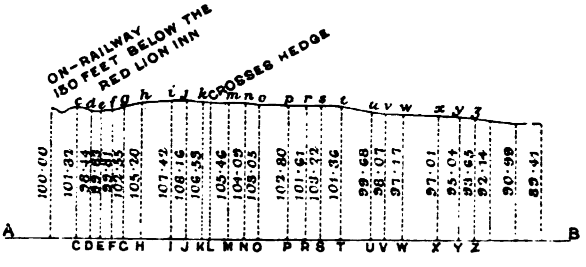

Fig. 490.—Plotting of a Levelling.

previously given to show the form of levelling book, a horizontal scale of $7\frac{1}{2}$ chains to 1 inch and a vertical scale of 150 feet to 1 inch have been adopted, but of course for a levelling of this length a much larger scale may be used.

It will be observed that A B, Fig. 490, is ruled for the datum line; on it are set off from A, the horizontal distances at the points, C, D, E, &c., according to the horizontal scale and through the points, C, D, E, &c., are drawn lines C*c*, D*d*, E*e*, &c., perpendicular to A B: on these lines are set off the vertical distances to the points, *c*, *d*, *e*, &c., according to the vertical scale of 150 feet to 1 inch, and the line, *c*, *z*, passing through all the points, *d*, *e*, *f*, *g*, *h*, *i*, &c., will represent the surface of the ground levelled over.

Different practices prevail in different parts with reference to the colouring of colliery working plans. In nearly all, however, the different boundaries have a thin edging of distinctive colour run round them in order to distinguish which particular taking the workings are advancing under, and to facilitate a calculation of the areas worked from time to time under that particular territory, as well as to draw attention to the near approach of workings to its extremities. The method of calculating these areas consists in dividing them into a series of triangles by ruling pencil lines on the plan, or on a tracing of the areas taken from the plan. The area of each triangle is obtained separately by multiplying the base by half the perpendicular height as measured by the scale on which the plan is made, and an addition of these separate areas of triangles gives the total acreage. It is usual to measure these triangles in links, and as there are 100,000 square links in an acre, by pointing off 5 places of decimals, the result is obtained in acres and decimals. The decimal portion is then multiplied by 4, and five places of decimals marked off to the right again, any figure there may be to the left of the decimal point denoting roods. Again, multiplying by 40, and pointing off five decimal places, the figure or figures on the left of the decimal point will be perches. For example, suppose the acreage of some workings on the plan consisting of three triangles of the following measurements is required:—

	Base.	Perpendicular.
(1)	422 links	126 links
(2)	562 ,,	238 ,,
(3)	321 ,,	92 ,,

$$\begin{array}{rcr}
\text{For (1) } 422 \times \frac{126}{2} & = & \cdot 26586 \\
\text{(2) } 562 \times \frac{238}{2} & = & \cdot 66878 \\
\text{(3) } 321 \times \frac{92}{2} & = & \underline{\cdot 14766} \\
 & & 1\cdot 08230 \\
 & & \underline{4} \\
 & & \cdot 32920 \\
 & & 40 \\
 & & \underline{\underline{13\cdot 16800}}
\end{array}$$

The area is thus ascertained to be 1 acre, 0 rood, 13·168 perches.

If the amount of coal contained in that area on a 4-foot seam is required, proceed thus:—

The difference in the specific gravity of coals obtained from different seams causes authorities to give different methods of calculating the yield. Newcastle coal is taken as weighing ·936 of a ton per cubic yard, or at 1,510 tons per foot thick per acre. An easily remembered rule is to take 120 tons per inch thick per acre (in cases where the specific gravity is not stated), but this is not quite so much as 1,510 tons per foot thick per acre. A liberal allowance is usually made in any calculations of this sort for loss in working and faults, and the amount of deduction will depend much upon the prevalence of faults or otherwise over the area to be calculated, from ⅕th to ⅓rd being usual. Adopting 1,510 tons per foot thick per acre, then 1,510 × 4 × 1·0823 = 6,537 tons; from which deduct ⅓rd, and there are 4,358 tons as the probable yield under the area calculated. If the seams lie at a high inclination, allowance must be made for it in calculating the quantities.

In some plans the workings have a different colour for each year's work put on them, others have only one colour, which is extended from time to time; the date of the survey is neatly printed on these, indicating the face at that time, and the colour is added on the workings as they advance from time to time; others again are not coloured at all. As a general rule, the less colouring the better, because moistening tends to shrink the paper.

In setting out the curves on the surface for the colliery railway much care must be exercised. If the sidings are to be cleared by the railway company's locomotives, the radius of the curves on which these are allowed to work by the Company's regulations must be ascertained, and the curves given a sufficiently long radius to meet such requirements. Much thought and some preliminary surveys may be wisely undertaken before constructing the railway, so that the best route be selected and adopted. This route when decided on should be shown on the plan prepared for the purpose by marking on it, with a distinctive colour, the centre line of the proposed railway. This centre line must then be staked out on the ground and the levels taken along its course. From the levels taken a section is plotted, the various gradients decided on, and the amount of cutting and embankment ascertained and marked on the section.

If the survey of the land has been accurately made, no difficulty will arise in setting out the centre line of the railway on the ground. Measurements from the intersection of fences and other points may be used to fix the termination of straight portions of the railway, but the curves will require very careful treatment in pegging out.

One method of setting out curves on the ground is shown in Fig. 491. A dozen ranging rods, shod with iron, a couple of plumb-bobs, a surveyor's measuring rod, a Gunter's chain or steel measuring band, and the marking pegs will be required. The curve extends from the point B to K in Fig. 491, A B being the straight portion of the railway at one end of the curve, and K D that at the other end.

By examining the plan in the office before going on the ground, the radius of the curve and the offset required in the setting out may be ascertained.

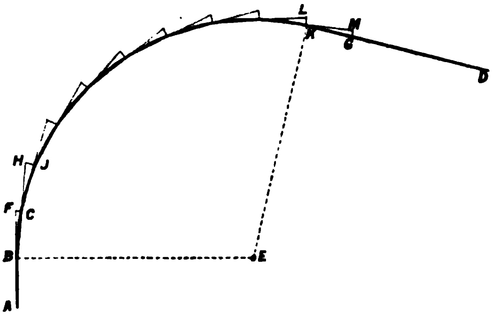

Fig. 491.—Setting out a Curve.

From B on the plan, draw the line B E at right angles to A B on the inside of the curve for any length. From K also draw K E at right angles to D K, till it intersects B E at E. E then is the centre of the curve B K, and the radius may be measured off by means of the scale. Divide the arc B K, into equal distances, B C, C J, &c., of a chain, or thereabouts. Continue the straight line A B to F, and make B F = B C. Then the offset from the tangent $F\,C = \frac{B\,C^2}{2\text{ radius}}$. Join B C, and continue the straight line to H. Make C H equal to C J. Then the offset $H\ J = \frac{C\,J^2}{\text{radius}}$. All the other offsets will be the same as H J throughout the curve to its end at K, but an additional one beyond the curve must be made to fix the position of the straight line K D. This offset M G is the same as F C, and is just one half the others.

Assuming the radius E B to be 20 chains, and the equal divisions along the arc to be one chain long, then $F\,C = \frac{100^2}{4{,}000} = 2{\cdot}5$ links, or expressed in feet $\frac{66^2}{2{,}640} =$ 1 foot 7·8 inches. $H\ J = \frac{100}{2{,}000} = 5$ links, or expressed in feet $\frac{66^2}{1{,}320} = 3$ feet 3·6 inches. The offset to be used throughout, then, is 3 feet 3·6 inches, except F C and M G, which must be half this, or 1 foot 7·8 inches.

Proceeding to the ground on which the curve is to be set out, first fix a ranging rod at A, another at B, and then in exact line with A B fix another rod at F, one chain from B, or such other distance as was decided on in dividing out the curve. Mark off the offset, 1 foot 7·8 inches, F C, and carefully test that the length of B C = B F. C then is the first point in the curve.

To determine the next, a rod must be set up at H in exact line with B C, making C H equal in length to B F. Mark off the full offset of 3 feet 3·6 inches at H J, making C J = C H. A similar mode of procedure is carried on until coming to K, the remaining offset after this being M G, and is only 1 foot 7·8 inches. The continuation of K G to D gives the straight piece at that end of the curve.

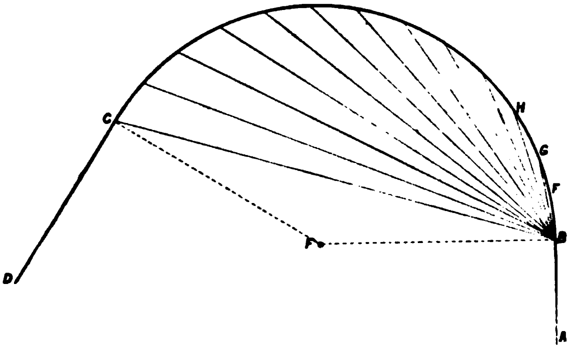

Fig. 492.—Setting out a Curve.

Another plan of setting out curves is shown in Fig. 492. The country must be open to admit of its adoption. A B and D C are two straight portions of the proposed railway communicating with each other by a curve B C, of which E is the centre. A theodolite is set up at B and directed towards F, which is one

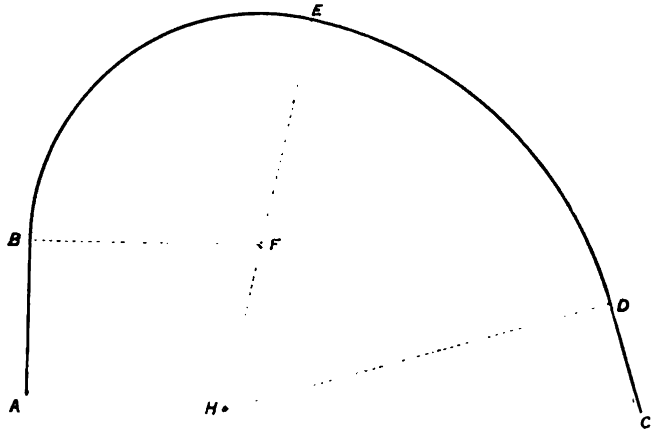

Fig. 493. Setting out Curves.

of the even divisions into which the curve is divided. The angles F B G, G B H, &c., being equal when B F, F G, G H, &c., are all equal, the surveyor stands at the theodolite and sets off these angles, and directs two assistants with the chain where to set up the rods G, H, &c.

There are other methods of setting out curves, but that first described is the most commonly adopted, and it is applicable to the underground roadways when these require to be driven with a curve.

The curve separating two sections of straight railway may not proceed from a single centre. Fig. 493 is an example of this, where the curve B E is described from the centre F. The curve E D is continued with an increased radius, and to determine its centre before describing it join E F and continue the line to H, making H E the desired new radius. From the centre H, describe the curve E D. The continuation of the railway D C must be at right angles to H D.

If an S curve is required as shown in Fig. 494, draw D G at right-angles to C D, and make D G equal the radius of the curve. From the centre G, describe the curve D F. Join G F, and continue the line to E, making E F equal the radius of the second curve. From the centre E, with the radius E F, describe the curve F B. The straight continuation of the railway, A B, will be at right angles to E B.

If an underground roadway has to be driven with, say, a 50-yard radius, the necessary offset must be short, as the roadways are narrow. The equal divisions of the curve must be short; in this instance take a yard. The first offset then, $\frac{1^2}{2 \times 50} = \cdot 01$ of a yard, or ·36 of an inch, and all others, except the last, will be double that. Bearings may be placed in the roof in a similar way to ranging the rods on the surface, but the bearings will have to be frequently changed, as the driving cannot proceed far with one instalment of them.

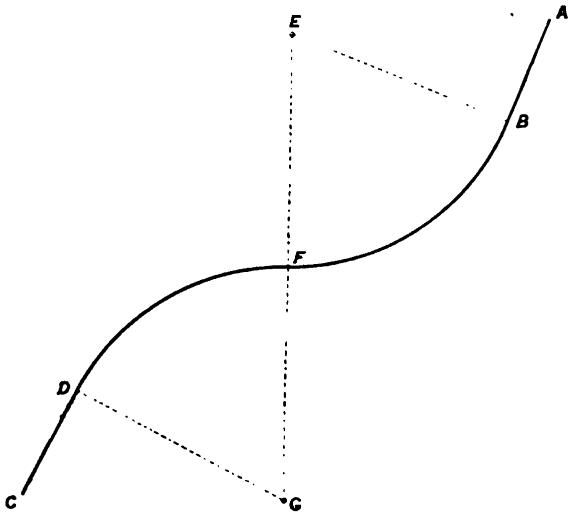

Fig. 494.—Setting out Curves.

With regard to making geological sections, the mode of procedure is first to select the line of section, unless this is already some particular line which must not be deviated from. The line chosen will be influenced to a certain extent by the information afforded by different routes, keeping the line, however, as straight as practicable.

Perhaps by slightly deflecting from the straight line useful information may be made available by including the results of pit sinkings or workings. If, however, it is necessary to make a section from one given point to another in a direct line, which is unalterably fixed, the best must be made of the circumstances, and such information as can be obtained, used. A surface levelling between the two points may then be proceeded with, and plotted on a suitable scale, determined by the length of country over which the section extends, and the purpose for which it is required. On this plotting would be shown the surface levels of the different pits crossed, and all other prominent objects. A careful scrutiny of the ground, quarries, cliffs, railway cuttings, and an examination of the fossils obtained from the rocks, &c., may then follow; the classification of the strata noted, with the amount of dip or rise in the direction of the line of section, and the direction of dip. By this means it is possible to calculate the thicknesses of the different rocks, and to plot them below the surface line previously shown on the drawing paper. In colliery districts, the pit sinkings and workings enable the surveyor to show the position of the seams of coal, the different strata, and also the faults met with, all of which require to be drawn.

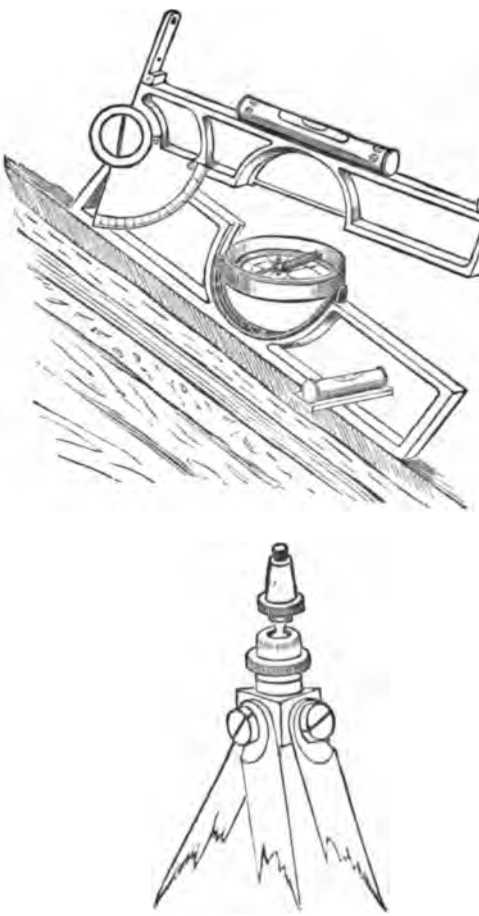

Fig. 495.—Louis's Improved Davis's Clinometer.

The section is outlined in pencil, and afterwards inked in with Indian ink, the main lines being shown in black, the faults with some other striking colour. Coal-seams are shown with bold, thick black lines, and the different strata are distinctly coloured. When the colours are all dry, the names of the coal-seams may be neatly printed over them, and either a reference printed in a corner of the paper identifying the strata represented by the different colours, or the names of the strata may be printed on them without using a reference. A title or heading is then printed along the top of the paper explaining the nature of the section and its scale or scales drawn or given.

A very convenient instrument for ascertaining the necessary particulars is Louis's improved Davis's clinometer, made by Messrs. John Davis & Son, of Derby, and shown in Fig. 495. In it a compass is placed in the lower portion of the clinometer frame, where it is carried by means of pivots on a brass arc, which may be revolved in the frame. This allows of the instrument being laid on strata of any dip, the rate of which will be ascertained by raising the upper portion of the clinometer frame until the air-bubble of the spirit-level placed on top is in the centre of its run, as shown in the drawing. The angle of dip may now be read by means of the graduated arc, whilst its direction is obtained by giving motion to the compass-box till it assumes a horizontal position in the under portion of the frame, and then reading the needle-bearing. The strike of strata is usually determined by first ascertaining their maximum dip, and the compactness of this instrument allows it to be readily turned whilst readings are taken to fix the exact direction of dip in any one place. The bubble of the lower limb is mounted on a swivel, and this enables the instrument to be levelled both ways without being reversed. The compass-box is reversible in the under portion of the frame. The size of the clinometer is 6¾ inches long × ½ inch wide × 3 inches deep, and weighs 1 lb. 2 ozs. It is provided with a tripod with ball-and-socket joint, 3 feet 10 inches long, and weighing 1 lb. 8 ozs. When attached to the tripod it may be used as a Dumpy level where great accuracy is not required, or for ascertaining the inclination of a sloping adit, or undulations of the ground.

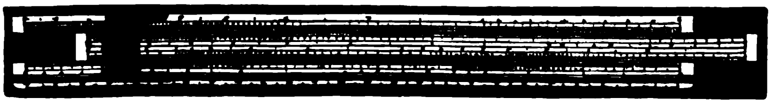

Fig. 496.—Celluloid Slide-Rule with Ordinary Cursor.

The number of calculations required in large colliery offices makes it highly desirable to use a calculating slide-rule in order to save labour. Although slide-rules are somewhat complex, there is no difficulty in using them if the instructions supplied by the inventors are carefully followed. A great number are made for special purposes only, some being more useful in one branch of engineering than others.

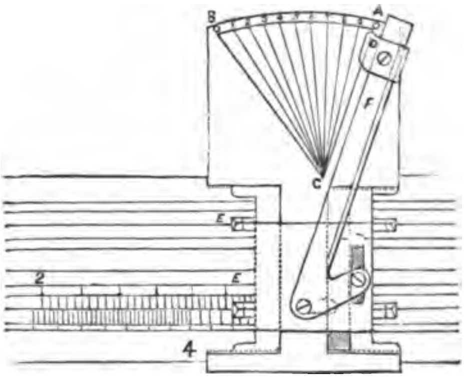

Fig. 497.—Goulding's Improved Cursor.

Fig. 496 shows a celluloid slide-rule with ordinary cursor, introduced by Messrs. J. Davis & Son, Derby, which enables a variety of technical calculations to be expeditiously made in addition to ordinary multiplication, division, evolution, and involution. The body of the rule is of wood, in which strips of hard white celluloid are inlaid, and on these celluloid strips the scales are engraved, being thus in black lines on a white ground. This rule is of the Gravet type, 10 inches in length, and the markings remain without relative change from variation of temperature. The upper part of the rule is divided into two parts, containing scales perfectly alike. The lower scale is exactly like the upper ones, but twice the length. The slide is divided the same as the rule, so that when the 1 of the slide is at 1 of the rule all other divisions will coincide. The cursor is the small metal frame fitting in two small grooves in the edges of the rule, and is used for pointing and subdividing the spaces on the rule.

With this slide-rule the engineer may compute the strength of beams, moments of inertia, horse-power of engines, hydraulic formulæ, areas of circles and ellipses, cubic contents, weights, conversion of British into metric or other systems of measurement, without putting down a single intermediate quantity. The electrical engineer may solve all problems involving the application of Ohm's law of the relation between current, electromotive force, and resistance. The surveyor may plot surveys by rectangular co-ordinates and set out curves. The chemist is enabled by its aid to calculate percentage compositions rapidly.

The celluloid slide-rule is specially useful for computing the *costs per ton*, which are usually calculated at collieries every week or fortnight. If used with Goulding's improved cursor (Fig. 497) instead of the ordinary one in order to divide a certain number of tons into any number of amounts of money with a view to arrive at the cost per ton in pence, set the dividing figure on B opposite 240 (pence in a £) on A, and opposite each of the amounts on B read on A the cost per ton in pence. Thus, if, in a colliery producing 6,000 tons in one week, £130 is spent on timber, the cost of that item per ton of output will be found by placing 6,000 on B opposite 240 on A, and by reading the result (5*s.* 2*d.*) on A above 130 on B. The results may be expressed as percentages by placing the total cost per ton on B opposite 100 on A, when opposite each of the items on B the relative percentages may be read on A.

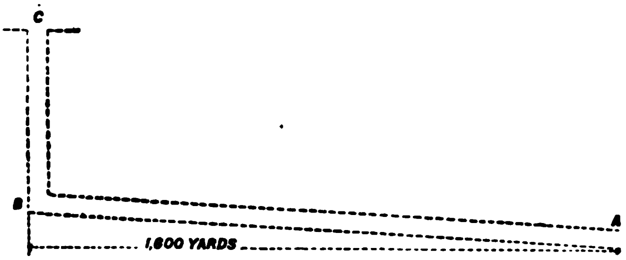

Fig. 498.—Driving to Air-shaft.

The following questions and answers, with which this chapter will be concluded, are such as are asked at examinations:—

> *Question* 128.—It is desired to drive from A to B, a distance of 1,600 yards (horizontal measurement), the driving to rise an inch in every 60 feet, and to put down an air-shaft, C B; what will be the depth of the shaft if its mouth is 350 feet above the level of A?

The driving would rise $\frac{1600}{20} = 80$ inches = 6 feet 8 inches between A and B (see Fig. 498), and 350 feet − 6 feet 8 inches = 343 feet 4 inches, which is the depth the air-shaft C B must be sunk.

> *Question* 129.—What would be the variation of a plan's meridian when the bearing of two objects thereon with its meridian is N. 13° E., and the bearing of the same two objects on the surface is found by an instrument whose needle has 19° of West variation to be N. 8° E.?

The objects on the surface will form a bearing with each other of N. 11° W. by the true meridian, therefore $13^\circ + 11^\circ = 24^\circ$, and the plan's meridian has 24° of West variation.

Question 130.—State in what way errors should be prevented from arising on plans owing to the variation, if the meridian shown on that plan be a magnetic one.

A surveyor should have some fixed points on the surface at long distances apart, one of which may be some prominent object, such as a church spire, and the other a stake driven into the ground in a convenient position to himself, free from all influence likely to deflect the needle. Periodically he should fix his surveying instrument, by plumbing it over the stake, and take the magnetic bearing to the other object. By keeping a record of these at different dates he can make allowance when plotting his surveys for the alteration in the variation. It is a good plan to make an observation of the sun from the same stake, by doing which he can fix the true north point and find the angle made between it, the position in which his instrument is, and the other prominent object referred to. Having once established this, he may at any time afterwards find the difference between true and magnetic north. If he is in the habit of making surveys at places far apart, allowance must be made for the difference in longitude of those places.

Question 131.—Describe any method for finding the true meridian.

In Lintern's *Mineral Surveyor and Valuer's Guide*,* the following method for finding the true meridian at any place whose latitude is known is described. The method is by equal altitudes of the sun or a fixed star. "The theodolite being fixed in a position selected as one of the points in the desired line, measure the horizontal angle from the said line to the sun or fixed star (whichever is used), when it is not near the highest or lowest point of its apparent daily course, and take also the altitude, adjusting the intersection of the webs of the telescope to it, in some marked and definite manner. Leave the limb clamped, and let the instrument remain quite undisturbed until the sun or star is approaching the same altitude at the other side of its apparent circular course. Then, without moving the body of the instrument, unclamp the limb, and again direct the telescope towards the same object, and follow it with the aid of the tangent screw until the cross-webs correspond precisely as before with the same relative position upon the disc of the object, as in the first observation; this being done, read off the horizontal measure of the angle between the assumed line and the new direction of the observed object; the mean between the two horizontal angles will define the position of the true meridian, *if a star has been observed.*

"If the *sun* has been observed, a correction is required in consequence of the sun's change of declination. When that declination changes towards the north, the approximate direction of the meridian, as found by the foregoing method, is too far to the right; if declination changes towards the south, it is too far to the left.

"The required correction is found by the following formula:

$$\frac{\text{Change of sun's apparent declination}}{2} \times \text{sec. latitude} \times \text{co-sec. } \tfrac{1}{2} \text{ angular}$$

motion of the sun between the observations."

An example will now be given showing an application of the method. Date of observation, July 20th, 1886. By consulting the Nautical Almanack for the date given, the sun's apparent declination is stated to be changing towards the south at the rate of 11′ 28″ per day. Time between the observations, which were taken in Springfield garden, Pontypridd, 6 hours. Change of apparent declination for six hours $= 11' \, 28'' \times \frac{6}{24} =$ say 3′. Theodolite fixed over a stake in the garden, from which Pontypridd church spire was observed. The upper plate being securely clamped to the lower, with the vernier at zero, the instrument was directed towards the church, and the lower clamping-screw tightened with the webs of the telescope correctly bisecting the top of the church spire. This being done, the upper plate was loosened, and an angle from the church to the sun obtained at that time, 9.15 A.M. Reading, $269^\circ\ 52'$. At 3.15 the upper clamping-screw was slackened, and the telescope, with the vertical limb undisturbed, directed to the sun; on its edge appearing in contact with the webs of the instrument, the reading of the angle was recorded as $41^\circ\ 45'$. The angle contained between the two readings therefore was $(360 - 269^\circ\ 52') + 41^\circ\ 45' = 131^\circ\ 53'$.

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* London: Crosby Lockwood & Son.

For the correction we have—

Log. of $\frac{3}{2}' = 90''$ . . . . = 1·954243
,, sec. $51^\circ\ 36'\ 15''$ (N. latitude of Pontypridd) = 10·206845
,, co-sec. $\frac{1}{2}$ of $131^\circ\ 53'$ . . . . . = 10·039467

2·200555 = Log. of 158·7, or say $2'\ 30''$. Therefore the approximate direction as found by the observation is too far to the *left* by $2'\ 30''$.

∠ from church to 1st observation $360 - 269^\circ\ 52' = 90^\circ\ 8'$
,, ,, 2nd ,, $= 41^\circ\ 45'$

Difference $2)\ 48^\circ\ 23'$

Difference halved $24^\circ\ 11'\ 30''$

but being $2\frac{1}{2}$ minutes too much to the left, the correct reading should be $24^\circ\ 9'$. The magnetic bearing from the same point of observation was taken to Pontypridd church by the same instrument at the same date as S. $43^\circ\ 12'$ W. from which deduct $24^\circ\ 9'$

Making the magnetic declination $19^\circ\ 3'$

Question 132.—Describe the underground and surface levelling staves, and state how you would proceed to level underground with the level and staff.

The staff used in underground operations is usually made about 9 feet long, and is in three pieces, which are connected together like the joints of a telescope, and these close down to 3 feet 6 inches. It is graduated into feet, tenths, and hundredths. The figures appear in an inverted position as seen through the telescope, but the surveyor soon gets accustomed to this. To avoid this some surveyors prefer having an additional lens placed in the telescope, others have the staff figures arranged so as to appear the right way up when seen through the telescope. The staff used on the surface is of the same description as that ordinarily used underground, being made in three pieces, the two upper sliding into the lower like the joints of a telescope, and being graduated into feet, tenths, and hundredths. The total length when out is 14 or 16 feet, and it closes to 5 or 6 feet. Staves of 18 and 20 feet are occasionally used. The method of proceeding with an underground levelling is as follows:—

An assistant holds the staff in an upright position at the shaft, and the levelling instrument having been set up truly level in the road which it is desired to level, a sight of the levelling staff is taken and the result entered in the book. The levelling staff is then moved along the road to where its inclination changes, or to where, owing to a change in the direction of the road, it is impossible to see farther, and a sight from the instrument gives the reading there. The fore-sight as this reading is called, is entered in the levelling-book, an example of which has been given. The difference of level between the two points is thus obtained, and this is either added to or subtracted from the level of the starting-point, known to be a certain number of feet above the datum line, which may be the Ordnance datum or have some clear and well-defined relation to it. At most collieries it is necessary to adopt a datum line which is 1,000 or 2,000 feet below that of the Ordnance. A number of sights is sometimes taken with the instrument in one position, but after the fore-sight is booked, the instrument is lifted and carried to a position inside the fore-sight, set up level there, and the staff, which has remained on the fore-sight last observed, is now turned round to face the instrument, but rests on the same part of the road as before, or what is preferable, in a shoe used for the purpose. This reading being entered as a back-sight, any intermediate sights may be taken, and the last will be a fore-sight. This method continues throughout the levelling. Some inconvenience and liability to error arises from the fact that owing to the varying heights of underground roads portions of the staff have occasionally to be pushed in, and these may be fractions of a foot, for which allowance must be made when entering the readings. This has led to one or two improvements in form for an underground levelling staff.

Question 133.—Should the workings be too low to admit the use of the level and staff, how would you proceed?

The usual method is by means of a straight-edge, which may be 6 or 12 feet long, according to the steepness of the measures, and by an ordinary mason's level. The straight-edge is laid along the road (with the mason's level placed on it), and then one end is lifted off the ground till it is in a truly level position as shown by the spirit level resting on it; a two-foot rule or other measuring strip is then used for taking the height the one end of the straight-edge has been raised, and the amount is entered in the book as a rise or fall. If the leading end of the straight-edge, or the inside end going "in bye," has been raised, the road falls, but if the outside end is raised, the road rises. The straight-edge is then moved forward another length, and the same process repeated, and so on throughout the levelling. This is only an approximate method of levelling, and too much reliance must not be placed on it.

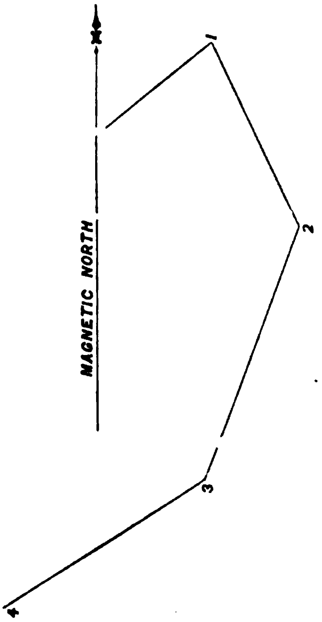


Fig. 499.—Approximate Plotting of Bearings.

Question 134.—Sketch on paper as near as you can, without the aid of scale or protractor, the following bearings of a survey: (1) N. 50 E., 46 links; (2) S. 26 E., 63 links; (3) S. 20 W., 85 links; (4) S. 56 W., 76 links.

It is not expected that the candidate without a scale or plotting instruments will get a very correct answer to this, but he may get an approximate result. Thus, consider the top of the paper on which the answers are being written as the north end and the ruled lines will be east and west. A line half way between these will represent an angle of 45°, and half way again $22\frac{1}{2}°$, and for the rest he may guess as nearly as possible. With regard to the distances, he must only consider them relatively, unless a rule or scale be used. No. 1 being 46 links, whatever length is chosen to put this on paper he must make No. 2 $\frac{63}{46}$ = nearly 1½ times as long, No. 3 $\frac{85}{46}$ nearly twice as long, and No. 4 would be not quite so long as No. 3. Bearing this in mind the result obtained would be in some such form as shown in Fig. 499.

Question 135.—State the different methods of connecting the underground workings with the surface boundaries.

First, by the magnetic meridian. Secondly, by suspending two copper wires from a balk at the top of the pit, and which have heavy weights attached to them in the sump. When the wires are steady, the balk on the surface should be turned round until the wires are in a line with the underground road, as determined by a light seen from the shaft along the road. Marks may then be made in the roof in a line with the wires, and rods may be ranged on the surface in a line with the same wires, and pegs afterwards put in so that the surveyor may have a base line on the surface and underground to refer to at any time. A third method is that of connecting them by means of a transit instrument, but this is not frequently done by ordinary surveyors. A fourth, and undoubtedly the best, method where circumstances admit of its being adopted, consists in connecting them by means of two or more shafts at a considerable distance apart, and, of course, the more direct the underground connection between the shafts the better. The bearing which the one shaft makes with the other on the surface, and their relative position with regard to the boundaries having been accurately determined, proceed to survey between the shafts and elsewhere underground. Plot the surface and underground survey on separate sheets of paper, trace the plotting of the underground survey and place the tracing on the surface plan, with the one pit of the tracing correctly adjusted over the same pit of the surface survey, then the other pit or pits of the tracing ought, when it is turned in proper position, to fall on the pit or pits of the surface survey. If they do not, some error exists, and the surveys must be again made. If they do, the underground roads may be pricked off from the tracing to the plan, while the pits of the tracing and plan exactly coincide, and the workings will thus be placed relatively right to the surface boundaries.

Question 136.—Explain what is meant by a scale of ½ inch to the chain.

It means that on any straight line of ½ inch long of the plan the actual measurement is a chain, or 66 feet, and therefore on 1 inch, 2 chains, or 132 feet, of actual measurement is represented.

Question 137.—Suppose a driving is going towards an old waste which is shown on a plan 30 years old, explain the precautions to be taken as regards meridian.

Reference may be made to the magnetic declination for the particular year in which the plan was made if it bears date, which it would appear from the question it does. Knowing the declination then and the declination now, bearings may be put on any underground road to direct its course clear of any wastes it is desirable to avoid or towards any particular point it is desired the driving to hole into. If the plan have no date fixed to it, the variation of its meridian may be determined by observing the bearing which any two objects make on the plan by its meridian, and also the bearing on the surface which the same two objects make with each other, by a needle whose declination is known.

Question 138.—What was the magnetic variation in 1893?

The mean magnetic variation at Greenwich for the beginning of the year 1893 was 17° 12′ W., but it is constantly changing, and this will not apply to other years nor to all other places for the same year.

Question 139.—Find the quantity of coal in one acre of a seam of the following section:—

Top coal . . .	2 feet	3 inches.
Band . . .		10 „
Bottom coal . .	1 „	9 „
Total	4 „	10 „

taking the specific gravity of coal at 1·25.

A cubic foot of water weighs 62·5 lbs. and therefore a cubic foot of coal whose specific gravity is 1·25 is 62·5 × 1·25 = 78·125 lbs. In one acre there are 4,840 square yards, or 4840 × 9 = 43,560 square feet. The actual thickness of the coal in the seam, disregarding the band, is 2 feet 3 inches + 1 foot 9 inches = 4 feet, and 43,560 × 4 = 174,240 cubic feet of coal in an acre, and as one such cubic foot weighs 78·125 lbs., the weight of the whole is $\frac{174{,}240 \times 78{\cdot}125 \text{ lbs.}}{2{,}240}$ = 6,077 tons, the total weight per acre.

Question 140.—If pillars are left 30 yards by 18 yards, the width of the headways being 2 yards and bords 4 yards, what percentage is got in the first working?

By sketch Fig. 500, the block of coal before the headways and bords are out will be 32 yards × 22 yards = 704 square yards. The pillar remaining is 30 × 18 = 540 square yards and the coal taken out is 704 − 540 = 164 square yards. Therefore as 704 : 164 :: 100 : 23·3, which is the percentage obtained in first working.

Question 141.—If 20 per cent. is allowed for loss of working and 31 per cent. screened out at bank, how many chaldrons of round coal will a foot thickness of a seam yield per acre?

20 + 31 = 51 per cent. of small and lost, therefore there would be 49 per cent. of large coal yielded. A seam of coal 1 foot thick contains 1,510 tons per acre, and since a chaldron weighs 53 cwt., then $\frac{1{,}510 \times 20}{53}$ = nearly 570 chaldrons. As 100 : 49 :: 570 : 279·3 chaldrons of large coal per acre.

Question 142.—What are traverse tables?

These are tables giving the northing, southing, easting, or westing each bearing of a survey makes. Suppose, for instance, a bearing of N. 30 E. to be taken and its distance is 200 links. Let AB in the accompanying sketch (Fig. 501) represent such bearing plotted. Then AC is the north meridian. If now from B a line is drawn at right angles to AC to meet AC in the point C, the northing of the bearing AB for the distance indicated is measured by the length of AC, and similarly its easting is measured by the length of CB. So with southings and westings when the bearings of lines are in that quadrant, or with southings and eastings, or northings and westings. Refer to any traverse tables now under the proper number of degrees and for 200 links will be found for the example given, AC = 173·2 links and CB = 100.

The results taken from these tables of a number of survey sets are added together so that it is possible to tell at any point of the survey the northing, southing, easting and westing from the starting to that point or from any other intermediate point passed in the survey.

Question 143.—In a heading going S. 45° E., what must be the course of a cross heading going at right angles on the north side?

N. 45° E. must be the desired bearing.

Question 144.—What is the use of the vernier? Describe the method of using it to find the 120th part of a degree.

The vernier is a contrivance for measuring the fractional parts of a graduated scale, and is connected with various scientific instruments, as the theodolite, sextant, barometer, &c. It is so constructed as to slide evenly along the graduated limb of an instrument, and enables the observer to read off the subdivisions of a scale with remarkable nicety. The divisions in the vernier are usually shorter than those upon the limb to which it is attached, the length of the graduated scale of the vernier being exactly equal to the length of a certain number ($n-1$) of the divisions upon the limb, and the number (n) of divisions upon the vernier being one more than the number upon the same length of the limb.

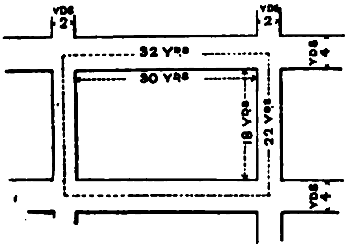


Fig. 500.—Sketch of a Pillar with Dimensions marked.

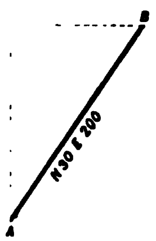
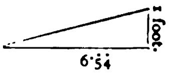


Fig. 501.—Plotting of a Bearing.

To find the 120th part of a degree, or 30″, the vernier should be divided into half divisions, and after sliding it along the limb to the position desired, the degrees and minutes may be read, and also the nearest half minute, by observing the divisional line of the limb that exactly coincides with a divisional line of the vernier.

Question 145.—If a heading A to B rises $5\frac{1}{2}$ inches per yard for 60 yards, from B to C, $2\frac{1}{2}$ inches per yard for 40 yards, what is the inclination from A to C, and the depth of cutting at B? Assuming the cutting to be 6 feet wide and every 14 cubic feet of the cutting weighed one ton, how many tons would such cutting yield?

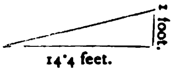
A rise of $5\frac{1}{2}$ inches per yard $= \dfrac{36}{5\frac{1}{2}} = 1$ in $6{\cdot}5\dot{4}$. Therefore for every $6{\cdot}54$ feet of horizontal measurement there would be a rise of 1 foot, thus,



The hypothenuse of such triangle will measure $\sqrt{6{\cdot}\dot{5}\dot{4}^2 + 1^2} = \sqrt{42{\cdot}843 + 1}$ $= 6{\cdot}6214$ feet measured along the slope, and as the full length of that slope is said to be 60 yards = 180 feet, its horizontal measurement is

As $6{\cdot}6214 : 6{\cdot}\dot{5}\dot{4} :: 180 : 177{\cdot}93$ feet.

Similarly in the next section of the heading, where there is a rise of $2\frac{1}{2}$ inches per yard $= \frac{36}{2\frac{1}{2}} = 1$ in 14·4. Therefore, for every 14·4 feet of horizontal measurement there would be a rise of 1 foot thus,



$\therefore \sqrt{14{\cdot}4^2 + 1^2} = \sqrt{207{\cdot}36 + 1} = 14{\cdot}4346$ feet, measured along the slope for each 14·4 feet of horizontal measurement, and as the full length of that slope is stated to be 40 yards = 120 feet, its horizontal measurement is As 14·4346 : 14·4 :: 120 : 119·71 feet.

Fig. 502 will further assist the answering of this question.

The point C is horizontally distant from A 177·93 + 119·71 = 297·64 feet = 99·21 yards.

B will be above the point A $\frac{177{\cdot}93}{3 \times 12} \times 5{\cdot}5 = 27{\cdot}184$ feet.
C " " " B $\frac{119{\cdot}71}{3 \times 12} \times 2{\cdot}5 = 8{\cdot}3$ "
C " " " A $\underline{\underline{35{\cdot}484}}$ "

A cutting then having a uniform gradient between A and C would rise $\frac{35{\cdot}484}{99{\cdot}21} = {\cdot}357$ of a foot per yard $= 4{\cdot}284$ inches per yard.

To find the depth of cutting at B.

$\frac{177{\cdot}93}{3} = 59{\cdot}31$ yards, and $\frac{59{\cdot}31 \times 4{\cdot}284}{12}$ or $59{\cdot}31 \times {\cdot}357 = 21{\cdot}174$ feet. $27{\cdot}184 - 21{\cdot}174 = 6{\cdot}01$ feet. Therefore the floor of cutting would be 6·01 feet below the heading floor at B.

To find the cubical contents of the rubbish which would come from such cutting, beginning at 0 at A, increasing to 6·01 feet at B, and then decreasing to 0 at C, we have $180 + 120 = 300 \times \frac{6{\cdot}01}{2} \times 6$ the width $= 5{,}409$ cubic feet, and since 14 cubic feet weigh 1 ton $\frac{5{,}409}{14} = 386\frac{5}{14}$ tons, which is the total weight obtained from the cutting.

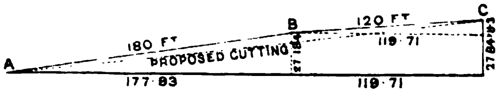
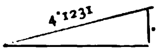


Fig. 502.—SKETCH SHOWING GRADIENTS OF HEADING.

Question 146.—What is the horizontal base and vertical height of an inclined plane 30·75 chains long and rising 1 in 4?

Here $\sqrt{4^2 + 1^2} = \sqrt{16 + 1} = 4{\cdot}1231$ measured along the slope for every 4 measured horizontally, thus



. As the whole length of slope is 30·75 chains, then as 4·1231 : 4 :: 30·75 : 29·832 chains for the horizontal base, and the vertical height is $\frac{29{\cdot}832}{4} = 7{\cdot}458$ chains.

Question 147.—Two shafts are separated by natural strata, and the workings are approaching the boundary; describe the method of making a check survey.

In the case of two shafts separated by natural strata and having an underground communication between them on the same level as the road leading from one of them to the workings approaching a part of the boundary, a surveyor would take advantage of the connection between the two shafts so as to strengthen the proof of accuracy in the survey, and to ensure that the barrier left (if any) should be of the width desired.

The method would be as follows. First, a very careful survey of the surface should be made, with a true and carefully adjusted instrument (preferably a transit theodolite), such survey to include the two pits, which call No. 1 and No. 2, and the portion of the boundary which an inspection of the working plan would show to be necessary, and taking care to include rather more of the boundary lines than actually required. The method of carrying out the details of the surface survey would be decided by the particular circumstances, and the lines chosen would be those offering the fewest obstacles to a successful survey, and therefore calculated to promote accuracy. These lines would pass near the portion of the boundary required, so as to reduce the offsets as much as practicable, and when completed they would contain proof of their accuracy by the "tie in," so that the surface survey would be complete in itself, and independent of the underground survey. If the pits were in a mountainous district, and it commended itself to the surveyor's judgment, he might in preference to running traverse lines as indicated, measure accurately a long base line in a valley, and fix the position of the pits and two extreme points of the portion of the boundary required, by means of angles taken off the base line; and a few traverse lines could be run along the boundary between the extreme points afterwards.

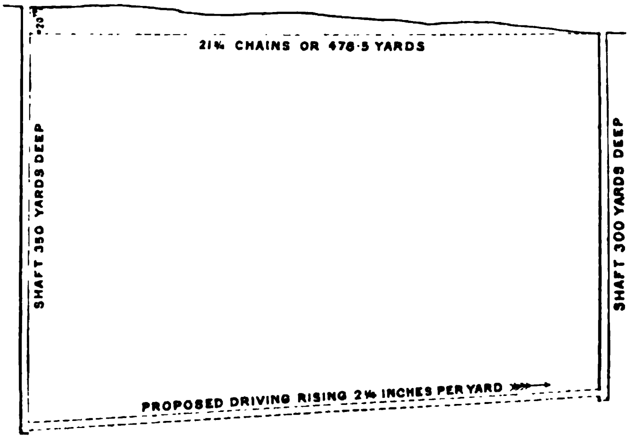

20
21¾ CHAINS OR 478·5 YARDS
SHAFT 350 YARDS DEEP
SHAFT 300 YARDS DEEP
PROPOSED DRIVING RISING 2¼ INCHES PER YARD

Fig. 503.—Sketch showing a Driving between two Shafts.

The method of procedure in the underground workings would be by traversing, taking care to use the same instrument as when making the surface survey, and if, say, No. 2 shaft is that from which the underground road leads into the workings approaching the portion of boundary already surveyed, the surveyor should commence the survey at No. 1 pit (supposing there to be no pumps in it, and the road leading therefrom to be clear of iron pipes, rapper wires, engine plane ropes and other substances of magnetic attraction), traverse the road to No. 2 pit, and continue on to the workings in question. If, from the presence of iron near No. 1 pit, the surveyor deemed it advisable, he would commence the underground survey at the face (which is usually free or easily freed from iron) of the boundary workings, or in some disused road, traverse "out bye" to No. 2 shaft and continue the survey to No. 1 pit; and if this were done, it would be convenient, but by no means necessary, to plot the underground before the surface survey. Should circumstances admit of a "tie" of any sort in the underground workings, full advantage should be taken of it, so as to strengthen the proof of the whole survey, but the position of the two shafts as ascertained on the surface, and also as ascertained by the underground survey, ought of course to agree, and if they are at some distance apart, a carefully made survey as now indicated, and with all its ties when plotted fitting in, would leave little more to be desired.

Question 148.—At the bottom of a pair of shafts 300 and 350 yards deep, it is required to drive a drift from one to the other; the distance between the shafts is 21¾ chains and the difference of surface level is 20 yards, what is the inclination of the heading?

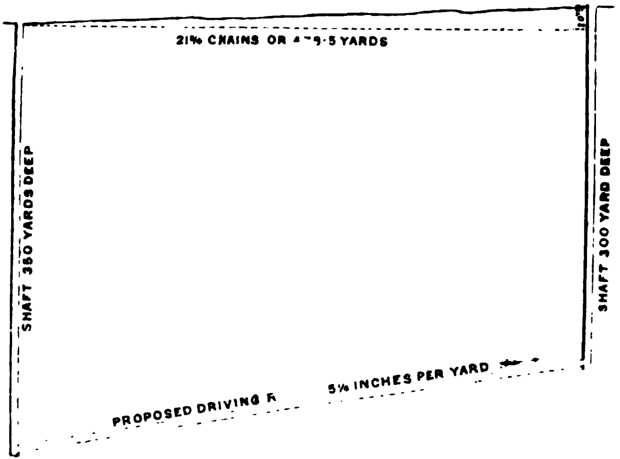


Fig. 504.—Skrtch showing a Driving between two Shafts.

This question is capable of two answers, the question not being complete in itself. It is not enough to state that there is a difference of surface level of 20 yards; it is necessary also to know whether the shallower or deeper pit top is at the higher level. First, supposing the deeper pit top to be 20 yards above the 300-yard pit top, then Fig. 503 will assist the answer now given.

350 — 320 = 30 yards, bottom of 300-yard pit above bottom of 350-yard pit. Distance apart 21¾ chains × 22 = 478·5 yards. $\frac{30 \times 3 \times 12}{478{\cdot}5} = 2{\cdot}25$ or 2¼ inches rise per yard from the 350-yard shaft bottom to the 300-yard shaft bottom.

Assuming, however, that the 300-yard shaft is at the higher surface level, it would affect the answer very materially, as the sketch (Fig. 504) and following working will show.

In this case 350 + 20 = 370 and 370 − 300 = 70 yards the bottom of the 300-yard pit is above the bottom of the 350-yard pit. Then $\frac{70 \times 3 \times 12}{478{\cdot}5} =$

5·26, or $5\frac{1}{4}$ inches rise per yard from the bottom of the 350-yard pit to that of the 300-yard pit.

Question 149.—The chord of a segment of circle is 60 feet, and the versed sine 78 inches. What is the radius of the circle?

The following formulæ are applicable to segments of circles:

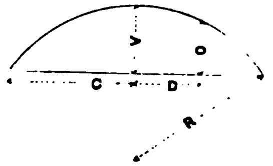


Fig. 505.—SEGMENT OF CIRCLE.

V = Versed sine.
C = Semi-chord.
R = Radius.
O = Any ordinate.
D = Distance of ordinate from centre.

$$O = \sqrt{R^2 - D^2} - (R - V).$$

$$R = \frac{V^2 + C^2}{2V}, \text{ or diameter } = \frac{V^2 + C^2}{V}.$$

$$V = R - \sqrt{R^2 - C^2}.$$

$$\text{Area of segment} = \frac{4V}{3}\sqrt{(0{\cdot}626\ V)^2 + C^2}.$$

Therefore, in the question given

$$R = \frac{6{\cdot}5^2 + 30^2}{2 \times 6{\cdot}5} = \frac{6{\cdot}5}{2} + \frac{30^2}{2 \times 6{\cdot}5} = 3{\cdot}25 + 69{\cdot}23 = 72{\cdot}48 \text{ feet.}$$

Question 150.—The radius of a circle is 72·48 feet, and the length of chord 60 feet, what is the versed sine?

$$\begin{aligned} V &= 72{\cdot}48 - \sqrt{72{\cdot}48^2 - 30^2} \\ &= 72{\cdot}48 - \sqrt{5{,}253 - 900} \\ &= 72{\cdot}48 - \sqrt{4{,}353} \\ &= 72{\cdot}48 - 65{\cdot}98 = 6{\cdot}5 \text{ feet or 78 inches.} \end{aligned}$$

Question 151.—If, where the workings of a coalfield proceeding in the direction of the full rise which is 1 in 10 and due north, the main levels are "set away" 1 in 130, what are the bearings of the levels?

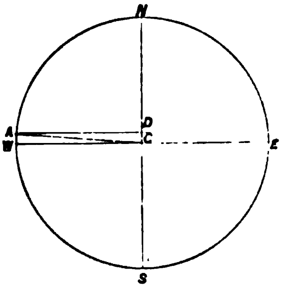


Fig. 506.—SKETCH SHOWING DIRECTION OF MAIN LEVELS.

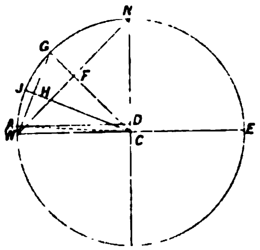


Fig. 507.—SKETCH SHOWING DIRECTION OF MAIN LEVELS.

Take a small ivory or boxwood protractor which has a scale of chords and other scales marked on it, and after setting the two legs of a pair of compasses at 60 degrees by the scale of chords, describe the circle NWSE from the centre C in Fig. 506. From a point W in the circumference draw a line through the centre C to the circumference E, and draw the lines CN and CS at right angles to WE, joining the circumference at N and S.

Let N represent north and CN will represent the full rise of the seam, which consider as being 130 long. By means of a scale take the point D, making CD equal to 10, or $\frac{1}{13}$th of CN. Draw DA parallel to CW, cutting the circumference at the point A. Join CA, and the line CA will represent the direction of one of the levels, for the point A will be 1 above C, and CA is 130 long. Place one leg of the compasses on W, and extend the other to A, and then measure that distance on the same scale of chords used before, and it will read about $4\frac{1}{2}^\circ$; or the lines CW and CA may be continued on the paper a suitable length in a direction from C and the protractor placed on the paper with its centre at C and the angle ACW may be thus read off, and it will be found to be about $4\frac{1}{2}^\circ$. The bearing of the level CA then is $90 - 4\frac{1}{2} = 85\frac{1}{2}^\circ$ N.W. approximately, because it cannot be read to exact minutes this way. In the same manner a level course going on the opposite side from C may be shown to have a bearing of N. $85\frac{1}{2}^\circ$ E.

If a protractor is not allowed, or is not at hand, a very close approximation to a correct answer may be obtained without it, thus:—

From the centre C, Fig. 507, with the radius 130, describe the circle NESW. Let NS represent the north and south, and E and W the east and west points respectively on the circle. Make CD equal to 10, and draw DA parallel to CW, cutting the circumference at the point A. Join CA and WN. Then CA will represent the direction of the level course and A will be 1 above C or W.

Bisect the chord WN in F and join CF. Continue the line CF to cut the circumference in G. Then the length of the chord WN is $\sqrt{130^2 + 130^2} = 183{\cdot}847$, and since the point N is 13 above C, the point F is 6·5 above C. The angle CWF is equal to the angle WCF, therefore the side CF is equal to the side WF, and WF is $\frac{183{\cdot}847}{2} = 91{\cdot}924$. As F is 6·5 above C, it will be found that the point G is as 91·924 : 130 : : 6·5 : 9·1924.

Now join GW, and its length will be $\sqrt{91{\cdot}924^2 + (130 - 91{\cdot}924)^2} = \sqrt{91{\cdot}924^2 + 38{\cdot}076^2} = 99{\cdot}17$. Bisect WG in H and WH $= \frac{99{\cdot}17}{2} = 49{\cdot}585$. Join CH, and continue CH to cut the circumference in J. The point H is $\frac{6{\cdot}1924}{2} = 4{\cdot}5962$ above C or W. Find the versed sine JH in the segment GW by the formula given in Answer 149, thus $130 - \sqrt{130^2 - 49{\cdot}585^2} = 9{\cdot}828$. Therefore CH $= 130 - 9{\cdot}828 = 120{\cdot}172$. As H is 4·5962 above C, the point J is as 120·172 : 130 : : 4·5962 : 4·972.

The angle GCW contains 45° and JCW $22\frac{1}{2}^\circ$, and approximately the proportion of degrees for the angle ACW will be found, thus as $4{\cdot}972 : 1 : : 22\frac{1}{2} : 4{\cdot}525$, or, say. $4\frac{1}{2}^\circ$, and therefore the bearing of the level CA is approximately $90^\circ - 4\frac{1}{2}^\circ = 85\frac{1}{2}^\circ$ N.W. The level proceeding on the opposite side will bear approximately N. $85\frac{1}{2}^\circ$ E.

The most satisfactory, and indeed the only strictly accurate, answer to a question of this sort is obtained from a solution by trigonometrical rules, involving the use of tables, &c.

CHAPTER XV.

LIGHTING: SAFETY-LAMPS: FIREDAMP DETECTORS: CARBONIC ACID GAS DETECTOR: NAKED LIGHTS.

Early Lighting—The Steel Mill—Humboldt's Lamp—The Davy Lamp—The Stephenson—The Clanny—Morgan's—Protector—Gray's—Hepplewhite-Gray—Marsaut—Mueseler—Ashworth's Mueseler—Evan Thomas—Clifford—McKinless—Howat Deflector—Thorneburry—Marshall's—Porch Lamp—Purdy's Lock—Craig & Bidder's Magnetic Lock—Cuvelier's Patent Lock—Extinguishing Locks—Lead Rivets—Wolstenholme Locking Machine—Lead Rivet Moulding Machine—Lighting and Re-Lighting Locked Lamps—Shields—Shut-Offs—Wick—Illuminants—Glasses—Cleaning Lamps—Lamp Room—Examining and Testing Safety-lamps before going into the Mine—Patterson's Testing Apparatus—Photometric Tests—Use of Lamps—Primary and Secondary Portable Electric Safety-lamps—Ansell's Diffusion Detector—Forbes' Damposcope—Hardy Detector—Aitken Indicator—Angus Smith's Air Compression Syringe—Liveing's Firedamp Detector—Maurice's Firedamp Indicator—Coquillon's Indicator—Le Châtelier's Eudiometer—The Shaw Gas-Tester—Garforth's Ball Detector—Safety Lamp Alarm Arrangements—Mallard & Le Châtelier's Safety Lamp—Pieler Lamp—Ashworth's Benzoline Lamp—Chesneau Lamp—Stokes' Alcohol Flame Testing Lamp—Hydrogen Gas Testing Lamp—The Test Chamber for Observing Flame Caps—Carbonic Acid Gas Detector—Candles and Small Oil Lamps—Large Oil Lamps—Sinclair's Comet Lamp.

EARLY LIGHTING.

In the early days of mining in this country torches are believed to have been used for lighting the outcrop workings. In the event of firedamp being found in troublesome quantities the simple workings were readily abandoned and fresh ones sought. The danger of explosion, however, was not so great as might be supposed, owing to the fact that marsh gas requires a certain proportion of atmosphere to be mixed with it in order to become explosive, or capable of ignition, whereas in primitive mines there was little, if any, circulation of air in the workings. Explosive mixtures, therefore, seldom existed, and even when an explosion occurred its effects must have been confined within comparatively narrow limits.

In other countries, on the Continent, relics in ancient mines give ground for the belief that earthen oil lamps, similar in shape to those now made, were used instead of torches. In later days the fear of danger in fiery mines led to the use of dried fish skins from the scales of which a faint phosphorescence was given. Such methods, however, must have been very inadequate, and in point of fact mining operations were often carried on without any light. The difficulties and dangers of working under such circumstances must have been very great, and although the danger of exploding the firedamp was thus obviated, yet the means of detecting noxious gases and other perils were equally absent. When serious difficulties were encountered which baffled the skill and courage or the appliances of the miner, he would readily desert the shallow shafts or adits in which the coal seams were inexpensively won, and would pursue his experiments elsewhere.

Tallow candles were used in the workings and in searching for gas. A search for gas consisted in "trying the candle." A small candle (30 or 40 to the lb.) was carried in a lump of wet clay, and the flame reduced in size by placing clay at its lower extremity near the exposed portion of wick. The candle was raised from the floor very gradually in an upright position with one hand of the observer held palm outwards towards the light to shield it entirely from view except the very tip of the flame. As it was raised an appearance of "top" or "cap" of blue flame above the candle flame indicated the presence of gas in the air. As soon as this appeared the candle would be quietly lowered, the searcher would withdraw with as little disturbance as possible, and some means would then be resorted to for dispersing the gaseous mixture. Where the quantity of gas was small attempts were sometimes made to get rid of it by beating it out with a coat or with a piece of brattice cloth. Before commencing to do this the man would retire to a distance, choosing a position away from the out-going current. His candle would be left here in a refuge of safety out of reach of the passing fire-damp. Another plan of disposing of gas believed to be present only in small quantity was to set fire to it. This was done at night after the withdrawal of the colliers from the working faces by a special operator with a good nerve, and sometimes a suitable dress. The dress was of wool or leather well damped, the face and head being further protected by a mask and a hood. Provided with a long stick, to the end of which his lighted candle was attached, he would crawl for the last few yards before reaching the explosive mixture previously discovered, and at the moment of firing it, would keep his candle the length of the stick in front of him, taking care to keep his body and head down on the floor that air might reach him during the passage of the explosion flame. When this had passed, in order that he might avoid the carbonic acid gas left by the explosion, he would immediately stand upright, or as nearly upright as the height of the road allowed. This officer of the colliery was called a "fireman" in this country, a name which is still retained to the present day for the man who searches for gas. In some countries he was called the ***penitent***, on account of the resemblance of his dress to that of certain religious orders in the Roman Catholic Church. In many instances, in spite of all precautions, the fireman fell a victim to the explosion he thus caused.

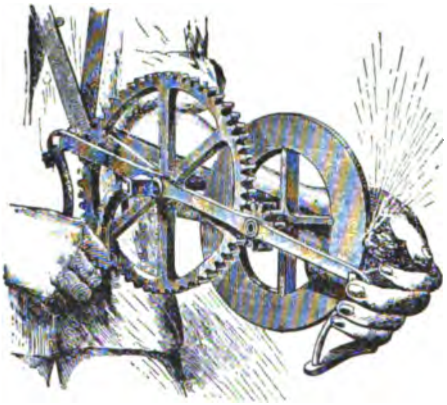


Fig. 508.—Spedding's Steel Mill.

The Steel Mill.—Frequent explosions of fire-damp rendered it highly desirable that a light should be introduced into mines which would be incapable of firing the surrounding atmosphere, and about the year 1760 the "Steel Mill" was invented by Spedding of Whitehaven, its object being to produce a shower of sparks which would give sufficient light to the collier, and yet would not fire an explosive mixture of gas and air.

Fig. 508 * shows the Steel Mill while in use throwing upwards a stream of sparks made by a sharp piece of flint in contact with the steel rim of a rapidly revolving wheel. A brass wheel about 5 inches in diameter having 52 teeth, works a pinion having 11 teeth, the latter being on the same axle as the thin steel-rimmed wheel against which the flint is pressed.

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* For this illustration we are indebted to the "Proceedings of the South Wales Institute of Mining Engineers," Vol. XVII., p. 125.

The three wheels are carried in a light iron frame, as shown in the illustration, and the whole, when in use, was supported by means of a detachable leather band passing through the back end of the frame and round the neck of the operator, who turned the handle attached to the axle of the driving wheel. Both hands were thus at liberty for use in working the mill, which required all the energies of a strong boy.

Frequently several steel mills had to be set going or "played," as it was called, in one place at the same time in order to get sufficient light, and it has been stated that at an ordinary North of England colliery their cost amounted to £80 per fortnight. This cost would be for the wages of those working them.

The sparks from a steel mill have a bright appearance in a pure atmosphere, but when discharged into explosive mixtures of gas and air their size and luminosity are increased according to the explosive proportions, and when the mixture of air and gas is in its most violently explosive state, if not before, these sparks are quite capable of themselves causing an explosion. The presence of carbonic acid gas or a preponderance of marsh gas beyond the explosive point is said to have been indicated by the sparks assuming a blood-red colour.

Notwithstanding the feeble glimmer of light obtained at great cost from the steel mill, and the fact that it caused explosions, it remained in use at any rate down to the introduction of safety-lamps in 1815. Whatever faith existed up to 1785 that the steel mill would not cause an explosion of fire-damp was then rudely shattered, as may be gathered from the following entry in Sykes' *Local Records of Northumberland and Durham, &c.* :—

" 1785 (June).—An explosion occurred in Wallsend Colliery, by which one man lost his life. This was the first explosion which was distinctly known to have taken place at the steel mill. Some doubt remained up to this time as to whether the fire-damp would explode at the spark of the steel mill or not; but the fact was clearly ascertained on this occasion, as the person, John Selkirk, who was 'playing' the mill at the time, survived the accident."

About this time Wallsend was the deepest colliery on the Tyne, being stated by Brand in 1787 to be 105 fathoms deep.

The same *Local Records* tell us that this explosion had been closely preceded by two others at the same colliery, one in November, 1784, by which three men lost their lives, and the other in December, 1784, resulting in two deaths. "These explosions were *supposed* to have taken place at the spark of the steel mill, by the light of which the people were working in the shaft. The bodies were not recovered for several months."

The remaining part of the entry for November and December, 1784, is interesting, as showing the date at which attempts were first made to use reflected sunlight in the shafts:—"In repairing the shaft after these explosions, the mode of throwing the rays of the sun down a shaft by a mirror, so as to light it, was accidentally discovered in the following manner:—While the people were working in the shaft, at about 80 fathoms from the surface, a carpenter was employed to do something at the head framing, immediately above the mouth of the shaft, and in using his saw, he turned the bright blade of it, accidentally, so as to throw a pencil of the sun's rays suddenly down the pit, to the great terror of the workmen below, who thought the pit had fired again. The cause of their alarm being, however, soon discovered, it suggested the idea of applying a mirror to throw the light of the sun down the shaft, which mode of lighting has since been frequently resorted to when other lights could not be used."

The light obtained from the reflection of the sun's rays must have been quite inadequate for workings, however limited in extent, for even on bright days such reflected rays could not have penetrated far from the shaft owing to obstructions in the galleries, while on sunless days not the faintest glimmer of light could have relieved the absolute darkness of the mine.

SAFETY LAMPS.

Humboldt's Lamp.—An ingenious lamp was designed by Humboldt in 1796, which burned without drawing the feed air from a surrounding atmosphere. It was only used for experimental purposes, as it very soon went out.

Clanny's First Lamp.—In 1813 a society for the prevention of accidents in coal mines was formed in Sunderland, and in the same year Dr. William Reid Clanny, of that town, devised and exhibited at the meetings of the society a lamp which was afterwards tried in an inflammable mixture of air and gas at the Herrington Mill pit, in the county of Durham, on October 16, and again on November 20, 1815. The flame of this lamp was insulated, and the air supply necessary for its combustion was blown through a stratum of water placed below by means of a pair of bellows. A second layer of water was arranged above the flame through which the products of combustion escaped. The water thus separated the air around the flame from that outside the lamp. In an explosive atmosphere it went out. This lamp was too unwieldy to be useful, and after being used experimentally was quickly superseded by later inventions.

The Davy Lamp.—In August, 1815, Sir Humphry Davy visited the collieries near Newcastle-upon-Tyne to investigate the nature of mine gases, and on November 9 of the same year read his celebrated paper on fire-damp before the Royal Society of London. This paper was published in the Philosophical Transactions of that Society, and contains a fascinating account of the experiments undertaken by Davy, which led to his safety-lamp, especially interesting in view of later inventions in the same line.

A flame is always a burning gas whether supplied directly as in the case of ordinary coal gas, or made from a liquid as in the case of oil lamps, or from solids as in the case of candles, or produced from the burning of wood or coal. In all cases the material, whether yielding one gas or more, takes a gaseous form before the appearance of flame. Before a gas can burn it must be first heated to what is called its temperature of ignition.

Davy found that fire-damp requires an admixture of a large quantity of atmospheric air to render it explosive. He experimented with small metallic tubes and sieves of brass and iron wire gauze, which he found, under certain conditions, prevented the passage of flame. Small metal rings have a remarkable power of reducing the size and illuminating power of the flame. Smaller rings were tried and altogether prevented the passage of flame. Metallic tubes one-fifth of an inch in diameter and 1⅛ inches long were found to be efficient. A spiral of wire at ordinary temperature, if quickly dropped into the flame of an alcohol lamp, will extinguish it, see Fig. 509. A spiral of thick copper wire is suitable for the experiment. The alcohol flame is not extinguished by the exclusion of air as in an ordinary extinguisher, but because it is cooled down by the wire spiral. If the latter be heated before being introduced into the flame, it loses its extinguishing power.

If gas be lighted at a Bunsen burner or at an alcohol lamp, the flame from either of which will not deposit soot on objects in contact with them, and if a piece of fine iron wire-gauze about 6 inches square be held at one corner in the hand, first above and then lowered on to the flame, there is no appearance of flame above the gauze any more than there would be above a solid iron plate.

Even if the gauze is brought down close to the burner there is no flame above the gauze, see Fig. 510. If, however, while retained in this position a lighted taper be applied to the upper part of the gauze, a flame is at once produced in continuation of that below, and it continues to burn as one flame. A similar result is obtained without the lighted taper if the flame be driven upwards by means of a blow-pipe.

Again, if the gauze be held about a couple of inches above the burner whilst the gas is turned off, a light applied above will, on turning the gas on, cause a flame above the gauze without passing below, see Fig. 511, or if the light be applied below, the gas-flame will appear between the burner and the gauze and not above it. In either case the flame may be produced above and below at the same time by applying a light to both sides of the gauze.

The explanation of these facts is that the wire gauze placed in the flame cools down that portion of the gas passing through the gauze below its temperature of ignition, the cool gauze being able to distribute by conduction the heat it receives and afterwards to emit that heat by radiation. A light applied above it heats the gas to the temperature of ignition, after which the ascending gas above the gauze continues to burn. A rapid current below directed upwards will force the flame through the gauze without other aid. The same explanation applies to the case in which the flame is first above the gauze. Until a light is applied there is not sufficient heat below the gauze to fire the gas immediately below.

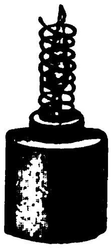

Fig. 509.—Experiment with Alcohol Lamp.

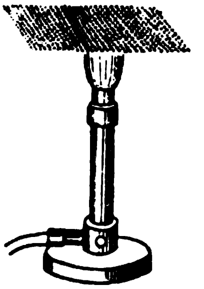

Fig. 510.—Experiment with Flame and Iron-wire Gauze.

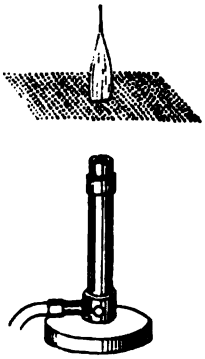

Fig. 511.—Experiment with Flame and Iron-wire Gauze.

If the gauze be made hot enough before it is lowered upon the flame, the flame appears above it afterwards without the application of a light. Or if it be cool at first and allowed to remain long enough in the flame to become sufficiently heated, the flame leaps through.

Davy's first practical safety-lamp was constructed of fine wire gauze, the wires being from $\frac{1}{40}$ to $\frac{1}{60}$ of an inch in diameter, and containing 784 apertures to the square inch. This gauze was made into a cylinder 10 inches long by 2 inches in diameter, and closed at one end with wire gauze. This then formed a cage into which the lighted end of a candle was passed at the open end of the cylinder. It was then sealed by soft clay pressed closely around the candle, and also around the bottom of the gauze cylinder so as to fill the space between the candle and the gauze and exclude the passage of air or gas, see Fig. 512. When this "lamp" or protected candle was used in an explosive mixture of marsh gas and air, the inflammable atmosphere passed freely through the gauze cylinder and was exploded within it, but the ignited gas did not pass back through the gauze, and therefore did not explode the external mixture. If the fire-damp in the explosive mixture was sufficient in quantity to absorb the whole of the oxygen of the confined air the candle went out. The mixture continued to burn after the explosion had extinguished the candle flame. But as the gas was being burned, the wire gauze was rapidly attaining a red heat, and Davy had discovered that when this was heated to a dull redness the flame would pass through the finest wire gauze meshes, as the temperature of the outside gas was then raised to the temperature of ignition. Davy knew therefore that his lamp did not afford absolute immunity from danger of explosion in the mine. He regarded it rather as a minimiser of such danger. The lamp as improved and left by him at the beginning of 1817 is shown in Fig. 513. To the top of the cylinder of wire gauze were added supplementary gauzes so as to increase the cooling surfaces at the point where the ignited gas always burned. These did not interfere with the passage of light through the lower gauze cylinder. An oil cup was introduced instead of the candle, and in order that it might be safely carried about three upright bars of the proper length were made, terminating at their lower extremity in a brass ring. This ring was provided with a thread, to which the oil chamber screwed and fitted nicely against the main gauze cylinder, previously inserted. The bars or pillars served also to protect the gauze cylinders from injury, and were bent at the top to form a loop to which a ring was attached as a handle. The oil chamber was provided with a suitable wick, and a wire pricker for trimming, raising, and lowering it. At the time this lamp was invented the air-currents probably had not a greater velocity than 4 or 5 feet per second, and it was very properly regarded as giving stronger protection from explosions in the mine. The first Davy lamp was used at Hebburn Colliery on the Tyne on January 1, 1816. It is worthy of record that when urged to patent his valuable invention so as to secure remuneration for his services, Sir Humphry Davy refused on the ground that he would take no payment for saving miners' lives. As a tribute of gratitude to the illustrious chemist the committee of the coal trade of the rivers Tyne and Wear, on October 11, 1817, presented him with a service of plate, valued at 2,000*l.*, for his invaluable discovery and disinterested labours.

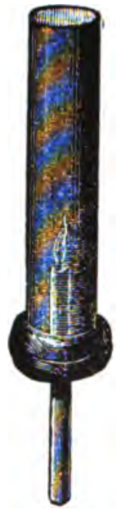

Fig. 512.—Davy's First Practical Safety-lamp.

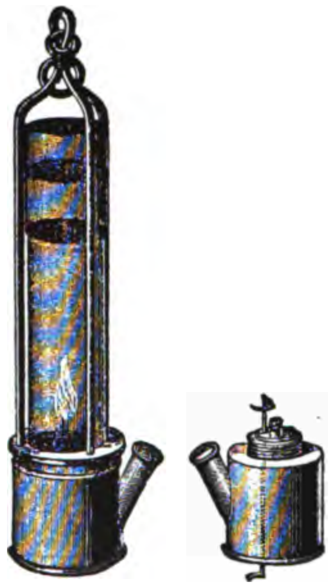

Fig. 513.—Early Form of Davy Safety-lamp.

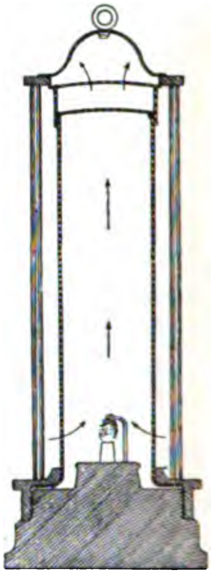

Fig. 514.—The Davy Safety-lamp.

The Davy lamp in use at the present time differs but slightly in construction from that made in 1817 (see Fig. 514). The feed air passes freely through the gauze cylinder and the products of combustion pass upward and escape through the gauze cap above the cylinder. The bars instead of being bent over at the top terminate in a brass annular ring to which is attached a metal roof or hood with a loop for the handle. It still depends on Sir Humphry Davy's discovery, that through a fine wire gauze, having 784 apertures to the square inch, formed by the crossing of 28 parallel wires, the flame of ignited gas would not pass unless acted upon by a strong current of wind. The gauze of the Davy lamp is constructed of iron wire about $\frac{1}{50}$th of an inch in diameter. The gauze for a surveying safety-lamp is made of copper, and if shielded, brass is used in its construction, as the lamp must be free from all iron. A glass lamp is, however, more often used for this purpose. Wire gauze of copper is about three times as costly as iron. No better material than iron has as yet been discovered for the construction of safety-lamp gauzes. In the Transactions of the North of England Institute of Mining Engineers, vol. xi., p. 178, will be found a description of the trial of a safety-lamp in which the wire gauze was made of aluminium. It was thought that this might have superior cooling qualities to the ordinary wire gauze, but it was found that when the lamp was subjected to explosive heat the wire melted. Aluminium is a very valuable metal for some purposes, and is used in the construction of some parts of expensive safety-lamps but never in the wire gauzes.

Notwithstanding the number of lamps in use, the Davy, protected from currents by a proper covering, is still found to be, under certain circumstances, an excellent lamp. It is very useful in the hands of the man who has been appointed to examine the working places. Most modern lamps go out in an explosive atmosphere, and this is well, for they are placed in the hands of the workmen whose energies are directed to coal-getting, &c., rather than in watching their lamps. With the fireman it is different. He is, or at least ought to be, a prudent and careful man, who has had considerable experience of firedamp, and has probably been appointed on that account. He usually detects the gas with the Davy without igniting it, and withdraws from the point of discovery quietly, unless the quantity appears large, in which case he puts out the flame himself by drawing the wick down into the oil-holder with the pricker. He on no account attempts to blow it out. It must be remembered that if the Davy lamp be kept for a short time in an explosive atmosphere the gauze becomes red-hot, and in that state the flame will pass, and ignite the gas outside. Again, if the lamp be placed in a strong current of air, the flame will be forced out of its proper position, and the end of it will then be directed towards the gauze, the wires of which will become heated to such an extent that the flame of the gas inside may pass to the outside. A velocity of about 6 feet per second is sufficient to produce this effect in a very short time. The Davy lamp is, therefore, not a safety-lamp in this current; nor would it be so if, while in this condition, it were subjected to a jerk or violent motion on the part of the person carrying it in a still explosive atmosphere. Another source of danger arises from the fact of inflammable material getting on the gauze. In a dry mine coal-dust adheres to it, and if gas should then be fired in the lamp the dust may be inflamed, and fire the explosive mixture outside. The same thing may occur from gauzes smeared with oil.

The Davy lamp consists of an iron wire gauze cylinder about 1½ inches in diameter, and 6 inches high, fixed to a brass ring, and screwed on to the oil-vessel. For better protection the gauze is doubled at its upper part. Outside the gauze 3 iron bars, placed at equal distances apart, connect the bottom and top rims, the latter having a metal roof, attached to which is a ring for carrying the lamp. Through the oil-vessel, in a close-fitting tube, a wire called the pricker passes, the purpose of which is to trim the lamp, and draw down the wick when it is desired to extinguish the flame. Sometimes a shield of tin is placed round

⅔rds of the Davy lamp, it being fastened to two of the upright bars, on which it may be made to slide up and down. Besides its defectiveness in respect to passing the flame at a rather low velocity, another great objection to the Davy lamp is its insufficient light, and most of the more modern lamps give a better light, glass being introduced at the bottom instead of gauze for this purpose.

An examination for fire-damp with the Davy lamp should be made at first without reducing the wick-flame. With the flame at its full size, the upper portion of the lamp is filled with the products of combustion, to the exclusion of a large volume of ignited gas, and a small volume burning within does not so readily pass to the outside, whereas a reduced wick-flame has a tendency to facilitate passage of the flame to the surrounding atmosphere. The reduced wick-flame is only necessary for low percentage of fire-damp. The lamp should, of course, be kept in an upright position, so that the flame and the smoke do not impinge against the sides of the lamp. The gauze cylinder should not exceed 1½ inches in diameter, for the larger the lamp the greater will be the force of an internal explosion, and consequently the greater the liability to drive the flame through the gauze from within. Thus, what are called Scotch lamps, which are really large Davy lamps with gauze cylinders about 3 inches in diameter, can hardly be classed as safety-lamps at all.

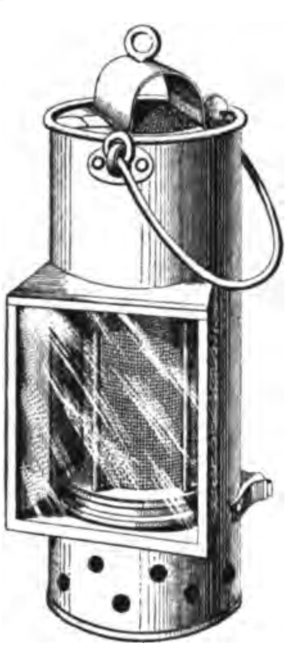

Fig. 515.—Davy Lamp in Case.

The weight of an ordinary unshielded Davy lamp is about 1½ lbs., while that of a "tin can" Davy (see Fig. 515) is about 2¼ lbs., without wick or oil in either.

Owing to the greater velocities of air currents in mines now than formerly, and to the discovery that an unprotected Davy lamp is not safe in an explosive mixture having a velocity of 6 feet per second, the use of the lamp under such conditions is prohibited by law unless it is specially protected. In the North of England this has led to the use of the Tin-can Davy shown in Fig. 515. This consists of an ordinary Davy lamp enclosed in a tin case provided in front with a flat pane of glass. The tin is perforated at the bottom for the admission of air. The case should extend from the bottom to the top of the lamp. Near the top, which is open, the handle is fitted. The lamp is locked to the case, and thus made it was one of the safest tested by the Royal Commission on Accidents in Mines, 1886. In their Report, the Commissioners say that "The Davy lamp enclosed in a case of any form which completely shields the gauze from the direct action of the current, may apparently be trusted in currents not exceeding in velocity 2,000 feet per minute (about 33 feet per second), but for higher velocities the greatest care is necessary in designing the case and its relations to the enclosed lamp."

Cases with inlet and outlet shut-offs may be arranged which would form additional safeguards. Cases without such appliances render the light steadier and somewhat improved. At best, however, the Davy gives a weak light, owing to the opaque wire surrounding the flame.

In different districts Davy lamps are made with movable metallic shields of different heights, which partly surround the gauze, and the dimensions of makers' lamps differ slightly.

External glass cylinders reaching to different heights up the gauze are also used, and lamps so shielded are usually known as "Jack lamps." With short glasses there is sometimes no provision for the air to pass underneath, but the long glasses usually rest on an india-rubber ring with projections, or on four metal pegs, so as to allow of a passage for the air to feed the flame under the glass. The glasses may be moved up and down outside the gauze, unless they extend throughout its whole length.

To be efficient in a current of 10 feet per second the glass cylinder surrounding the main gauze must reach above the double gauze formed by the smoke gauze cap.

Fig. 516 shows an arrangement of Davy as a combined examiner's and shot-firing lamp, sometimes used where there is not a specially devised shot-firing lamp. Here the glass cylinder extends about half-way up the gauze, and above it is fixed a metallic shield, partly surrounding the upper portion of gauze above the top of the glass. For the purpose of firing the shot the glass is lifted sufficiently to allow the wire to be inserted between the meshes of the lamp-gauze, in order that its temperature may be raised in the flame. When the wire has been heated and withdrawn, the glass is lowered, and, together with the upper shield, it forms a great protection from the force of the air currents.

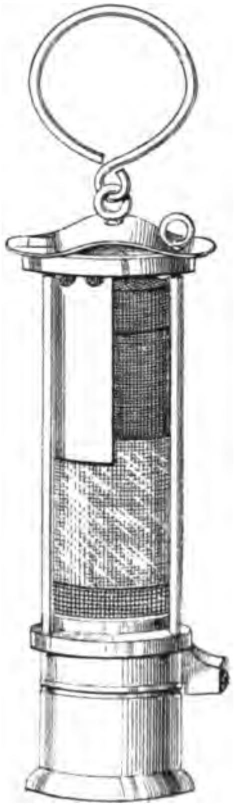

Fig. 516. Davy Lamp with Glass Cylinder and Half-round Upper Shield.

The daily examination of the mine before work begins is often made by the help of ordinary safety-lamps. The Davy, protected by a bonnet and a movable glass on the outside of gauze (to comply with the requirements of the 9th general rule of the Mines Act, 1887), is usually preferred by the examiners to other forms of lamps. In order to make the cap more distinct and thereby aid the observation it is usual to reduce the flame in size by drawing down the wick with the pricker in low percentages of firedamp. The luminosity of the flame being then less, its elongation, or the pale blue light of the cap, becomes more distinctly visible. An estimate may thus be made from the appearance of the lamp flame of the quantity of firedamp present in the air. The amount of firedamp present must exceed 3 per cent. for the cap to be distinctly visible with an ordinary gauze Davy lamp. Lower percentages cannot be detected by it, nor can 3 per cent. with certainty. With 5 per cent. of firedamp present, the cap reaches to the cover of the lamp, where it spreads out and fills the upper part of the wire gauze cage for a depth of 2·6 inches from the top. The result of these experiments will be modified with Davy lamps having gauze cylinders of varying diameters.

The *Stephenson* Lamp.—In 1812, George Stephenson, the eminent engineer, obtained an appointment at the Killingworth Colliery, about 7 miles to the north of Newcastle-on-Tyne. In August, 1815, he requested Mr. Nicholas Wood, who commenced his apprenticeship as a colliery viewer at Killingworth Colliery, in the year 1811, to prepare a drawing of a lamp from a description which he gave him. There is no doubt, therefore, that Stephenson was experimenting with lamps simultaneously with Sir Humphry Davy. The result was the invention of a safety-lamp, the characteristic feature of which was a glass cylinder about two inches in diameter, surmounted by a cap of perforated copper. The feed-air was conveyed under the glass cylinder by means of small tubes. On Oct. 21, 1815, Stephenson's first lamp, which had only one tube to admit air, was tried at a blower in Killingworth pit, and burned well. On Nov. 4 of the same year a second lamp was tried. This had three capillary tubes to admit air, and in consequence it burned better than the first. On Nov. 30 of the same year, Stephenson's third lamp was tried in the mine and found to be safe, and to burn well. In this lamp the air was admitted under the glass by a double row of small perforations. These lamps had no iron wire gauzes.

The question of priority as between the Davy and the Stephenson safety-lamp has been a subject of bitter controversy. It is clear that the subject occupied the attention of the two great men at the same time, and each simultaneously produced a lamp on much the same principle. Great credit is therefore due to both inventors. To Sir Humphry Davy's researches, however, we are indebted for the discovery of the principle of safety found in the iron wire gauze, which still constitutes the means of security in modern lamps, and which Stephenson himself adopted as a means of protecting his glass cylinder, in the place of a perforated cover over the glass previously used. The application of glass to safety-lamps was condemned by Sir Humphry Davy, on account of its liability to breakage.

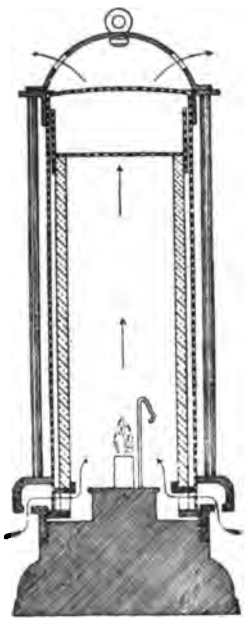

Fig. 517.—The Stephenson Safety-lamp.

The Stephenson or "Geordie" lamp, as it is sometimes called, as at present made is shown in Fig. 517. The cylinder of glass there shown is about 5 inches in height, and rather under 2 inches in diameter. It is covered by a cap of perforated copper, and placed within the wire gauze cylinder, which is slightly over 2 inches in diameter, and about $6\frac{1}{2}$ inches in height. A smoke gauze cap, which is a short wire gauze cylinder, placed over the top of the main gauze, is sometimes added to the main gauze cylinder, which it then overlaps for about $1\frac{1}{2}$ inches. The glass cylinder generally rests loosely on a brass ring, and can, by a jerk, be shifted vertically along the gauze, the amount of play differing in lamps from about $\frac{1}{4}$ to $1\frac{1}{4}$ inches. In some varieties of the Stephenson, the glass cylinder is fixed in its place by the gauze being slightly nipped in a ring, or by its reaching to the top of the gauze cylinder. The lamp weighs about 2 lbs. without oil or wick. The air to supply the flame is admitted through a number of small holes of about $\frac{1}{20}$th of an inch in diameter, formed in the under side of a hollow collar, and then through the main gauze which is kept in place by a ring below as indicated by the arrows. The ring of metal keeps the bottom of the glass cylinder about $\frac{1}{4}$ of an inch above the bottom of the main gauze. It is perforated with large openings to admit the air under the glass after it has passed through the main gauze. Unless these inlet perforations at the bottom of the lamp are kept free from dust and oil they soon become choked, and prevent the lamp from burning well. Occasionally the metal top to which the handle is attached is a perforated dome, but more often a bridged top is fitted, either of which allows the products of combustion to escape after passing out of the main gauze.

In an explosive atmosphere which is still or has only a low velocity, owing to the air entering from below not containing sufficient oxygen for the support of the flame, and, owing to the glass above the flame being filled for the most part with the products of combustion, the flame is soon extinguished. This may happen without igniting the mixture at all, or there may be a feeble burning at the bottom of the lamp. The quantity of gas burning is too small to raise the temperature of the gauze to a dangerous degree, but it may break the glass at its lower edge. With a broken glass a Stephenson becomes practically a Davy lamp.

In a strong current of inflammable gas and air, however, the force across the gauze above the glass cylinder causes a current to descend on one side of the glass, while the products of combustion are ascending on the other. Under such circumstances the gas will ignite at the top as well as at the bottom of the lamp, thus causing the gauze to become red-hot, and igniting the external gas mixture. In currents having a velocity of from 9 to 12 feet per second the Stephenson lamp becomes unsafe.

The *Clanny* Lamp.—Dr. William Reid Clanny's first lamp, which is often mentioned as the father of all safety-lamps, has been already referred to. After Sir Humphry Davy's discovery, iron wire gauze was applied to the lamp which had been devised by Dr. Clanny as well as to that of George Stephenson. Previous to this, it is probable that Clanny had adopted some modification of his lamp which he explained in a paper read by him on Aug. 19, 1839, before the members of the South Shields Committee. The Transactions of the North of England Institute of Mining Engineers, vol. xvii., p. 37, contains a description of the introduction of a short cone or cylinder of safety gauze, within the glass cylinder, and reaching to a little below the top of it so as to surround the flame. An illustration of the Clanny lamp as at present made is shown in Fig. 518. It consists of a lower cylinder of stout glass surrounding the flame, and an upper cylinder of wire gauze of less diameter. The feed air passes through the lower part of the wire-gauze cylinder, then down the inside of the glass cylinder, the products of combustion ascending inside the cold air-currents, and escaping through the upper part of the wire gauze. The oil-holder, with pricker, is the same as in the Davy lamp. Protecting bars are placed vertically around the glass and others around the gauze above. The main gauze is surmounted by a smoke gauze cap, and at the bottom is flanged to fit against a brass ring over the glass. The greatest care is required in fitting the glass cylinder. At its top and bottom sometimes washers of soft leather, indiarubber, asbestos or other suitable material are interposed between the glass and the metal rings, asbestos being the most effective. There is a great difficulty in maintaining the joints air-tight at all temperatures to which they are exposed. If no washers are used and a firm connection is made, the difference in the expansion between the metal and the glass is liable to cause a breakage when the lamp is burning; if the connection is made too loosely, a dangerous opening may be left between them. If washers are used the application of heat may cause them to shift their position and require frequent renewal unless asbestos is used. The glasses must be carefully selected, only those with truly parallel bearing surfaces and free from chipping should be used. A good washer cannot remedy defects in the glasses or

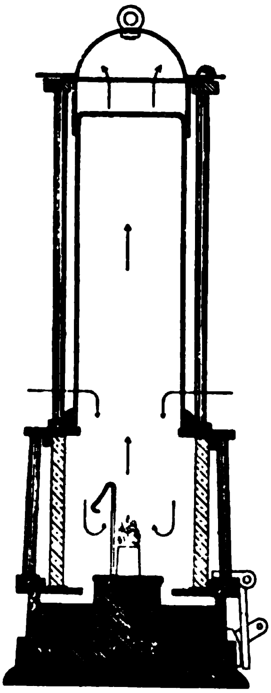

Fig. 518.—The Clanny Safety-Lamp.

irregularities in the metal surfaces. These remarks apply to all lamps in which glasses are used.

Lamps with glasses have a greater illuminating power than the Davy, but whenever glass or solid metal replaces any portion of gauze, then the internal explosions, when they take place, are thereby rendered more violent, inasmuch as the escaping gases have less area to pass through and attain a higher velocity. The danger of passing flame is thus increased. For this reason high glasses are undesirable, as are also those of large diameter.

The Clanny lamp is but slightly safer in use than the ordinary unprotected Davy, and will cause an explosion in a readily explosive gaseous mixture if the velocity of the current is only from 6 to 10 feet per second. With a percentage of only about 4¼ of marsh gas in the atmosphere, however, Clanny lamps are self-extinguishing. The Clanny is heavier to carry, but gives a little better light than the Davy, and can be carried without being blown out, in an air current which would extinguish an unshielded Davy. The risk of breakage to the glass has deterred its adoption in some instances.

When testing for gas with the Clanny, the first examination should be made without drawing down the wick. The products of combustion from the flame, when at its ordinary size, nearly fill the interior of the lamp and admit of only a small quantity of explosive gas reaching the flame. If this search fails to discover the presence of gas, a more delicate one may follow by drawing down the wick sufficiently to reduce the size of the flame.

Well-devised tin cases or bonnets render Clanny lamps less dangerous in currents having a velocity up to 10 feet per second.

The weight of an unbonneted Clanny lamp is from 2 to 2½ lbs., and with shield about 3½ lbs. without oil or wick.

At the introduction of the Davy, and for some time after, it was regarded as being a safe lamp in the ventilating currents of usual velocity, but not so in rapid currents in restricted passages, or if allowed to become red-hot by the combustion of fire-damp within the gauze. As explosions still occurred from time to time in collieries entirely lighted by Davy lamps, grave suspicion arose as to their safety in currents of greater velocity which became necessary in more extensive workings.* Until comparatively recent times the Davy, Stephenson, and Clanny were the only lamps generally used in the United Kingdom, none of them being altogether trustworthy. Practical tests at different times left on the minds of investigators a strong impression of their defects and gave a stimulus to ingenuity which has led to improvements. Still, although we have a large number of lamps, not one of them even now is perfectly satisfactory or thoroughly safe under all circumstances.

Inventors have great difficulties to overcome, arising not only from velocity of current, but from the rough usage lamps receive in the mine; the violent shaking and tilting they are subjected to, frequently accompanied by discharges from the oil chambers; the dust settling on them at the faces or finely suspended in the atmosphere and making it more explosive; water dripping from the roof on the glasses; oblique explosive currents at the face caused by falling stone or coal close to the lamps; and the grimy hands of the collier, which clog the gauzes and inlet feed-holes with dirt and grease.

Morgan's is an excellent lamp (see Fig. 519). Its chief features are a double perforated iron air-shield, and the construction of a shield at the top of the lamp, through which the products of combustion pass. The air for feeding the flame, after passing through the perforations in the outer and in the inner shields, goes through the three gauzes, then past the base of the inner chimney, and so to the flame, the products of combustion passing through the cone-gauze in the interior of the lamp, and up through the top shield.

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* In 1835, it was proved before a Royal Commission, that a greater number of lives had been lost in mines during the eighteen years succeeding the introduction of the Davy lamp than in the eighteen years immediately preceding the invention.

The outer shield has five horizontal rows of circular holes punched in it, the inner six horizontal rows of slits. The openings in one shield are opposite the solid portions of the other so as to prevent direct entry of the air which takes the course shown by the arrows. Of the three gauzes used the outer one is a cylinder without a top, the middle one is a Clanny type, and the inner one is curiously constructed, it being made up of an outer cylinder with a top which carries an inner short one carrying a metal chimney to improve the draught. The products of combustion have to pass through two gauzes and through openings in the top of the bonnet. Bars are placed round the glass cylinder to protect it from injury and connect the upper and lower rings.

When tested in an explosive current containing 8 per cent. of gas with a velocity of 47 feet per second, this lamp was found quite safe and went out in five seconds without exploding the external mixture. It is highly probable, therefore, that the lamp is capable of successfully resisting explosion in the mine in any velocity of current likely to be met with. The weight of the lamp without oil or wick is about 3 lbs., so that it is slightly heavier than some other lamps which are perfectly safe under ordinary conditions. The light given by it is good and steady in a still clear air, but strong currents reduce its illuminating power. It is not extinguished by a moderate amount of tilting or oscillation.

The lock for this lamp is a lead plug which differs in form from the ordinary lead rivet, and may be more quickly dealt with. There is an opening through the projection C on the oil chamber and a recess in the upper projection B. When the oil chamber is properly screwed on, the two vertical holes are opposite each other. In the lower is a small spring catch, D. The lead plug, A, is cylindrical, but has a portion cut out for the spring to fit into as shown in the illustration. The plug A is made to fit accurately in the openings C and B. To lock the lamp it is passed vertically upward through the lower opening until it is prevented from going further by the closed end of B, when the lower extremity is level with the bottom of the opening in C. In its upward progress it forces the spring D back out of the way, and when fully inserted the spring falls out under the cut portion of the plug. This prevents it from being drawn back out, while the covering at B arrests its further upward movement. To be unlocked the lead plug must be cut.

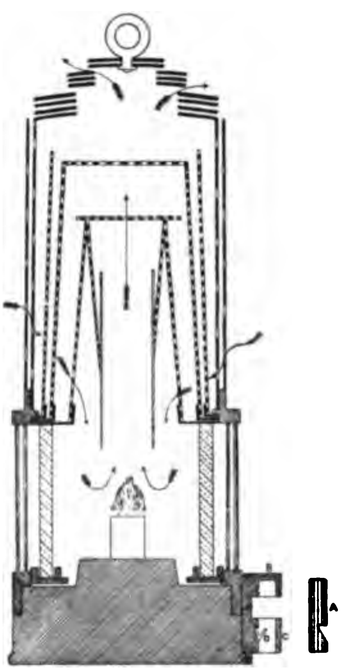

Fig. 519.—The Morgan Safety-lamp.

The *Protector* is also a good lamp. In it colzaline is burned instead of the vegetable or animal oils used in other kinds of lamps described. It makes no smoke or soot, will not clog in the gauze, does not require any pricker, and very readily shows the presence of inflammable gas. Inside the Protector lamp-bottom is a sponge which, in trimming the lamp, simply requires to be saturated with the spirit, the superfluous colzaline being poured out again. A permanent asbestos wick is placed in the upper part of the wick-tube, and the liquid absorbed in the sponge passes through an intermediate short piece of wick, which is renewed about twice a week, to the permanent packing above. The illuminating power of the lamp is excellent. This system of lighting is applied to many other lamps besides the Protector lamp, which term (Protector) refers more to the mode of locking than to the lamp itself. The Protector construction is applied to the Davy, Stephenson, Clanny, Mueseler and Marsaut lamps. Its peculiarity consists in the fact that, when a lamp has been locked, it cannot be opened without putting out the light. There is not much advantage in this, however, for if a person were likely to open a safety-lamp at all where this ought not to be done, he is also likely to carry matches, so as to re-light the wick.

With petroleum spirit a long wick-tube can be used, because the liquid rises in it sufficiently. This wick-tube is made with an outside thread throughout its length, as shown in Fig. 520. An extinguisher tube, A A, flanged at its lower end, is screwed over the wick-tube. This extinguisher tube, in turn, has a thread cut at the outer end of the flange. The oil vessel is screwed to this flange, after which the flanged tube is kept securely in its position by a bolt. A spring prevents the bolt from being withdrawn until after the removal of the oil-chamber. The extinguisher tube is screwed over the wick-tube before the wick is lighted.

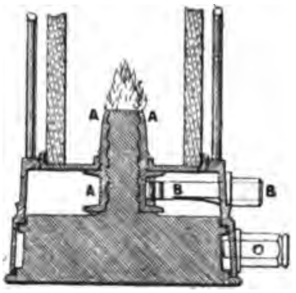

Fig. 520.—PROTECTOR SAFETY-LAMP LOCK.

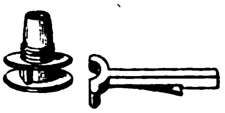

Fig. 521.—PERSPECTIVE VIEW OF A A AND B B.

After lighting the wick and screwing on the oil-chamber, the bolt B is pushed in, when the fork at the end seizes the extinguisher tube so as to prevent it from turning. (See perspective view in illustration, Fig. 521.) The oil vessel can easily be removed, but for that purpose it must be unscrewed. The act of unscrewing this causes the end of the wick-tube to traverse the closely fitting extinguisher tube for a distance of from 1 to 1½ inches, and when wholly unscrewed the flame is extinguished. By this arrangement it is practically impossible to open these lamps without extinguishing the flame. A bolt and lead plug lock is fitted for additional security, which allows the oil-cup to be unscrewed a little way to regulate the height of the flame while it prevents it from being altogether removed.

The Royal Commissioners on Accidents in Mines, in their Report, 1886, call attention to the danger of using the petroleum spirit sold under the name of "colzalene." (See Appendix I.) The danger arises from the absence of definite instructions which are absolutely essential to safety in the storing, transferring to smaller vessels, and to charging the lamps. The excess of spirit poured into the reservoirs not absorbed by the sponge fragments should be allowed to drain off by inverting the reservoirs in a proper place and by allowing them to remain a sufficiently long time. If this excess is not drained off, subsequent slanting or jerking of the lamp might cause it to run out, become ignited, and the flame pass through the gauze; or without such usage the reservoir would become warmed a few minutes after lighting the wick, and a flaring of the flame would follow. The spirit in the reservoir becoming greatly heated, might be driven out by the vapour and ignited at its exit.

Out of all the safety lamps experimented on by the Commissioners, only four were deemed worthy of recommendation by them as combining a high degree of security with good illuminating power and simplicity of construction. These were Gray's, Marsaut's, the bonneted Mueseler, and Evan Thomas's modification of the bonneted Clanny lamp.

A drawing of *Gray's* lamp is shown in Fig. 522. In it the air reaches the flame by four tubes, down which it passes from near the top of the lamp as shown by the arrows in the drawing. The tubes terminate in a cylindrical chamber under the glass. The inner wall of the chamber consists of a strip of gauze and through this the air passes on its way from the chamber to feed the flame within the glass. The products of combustion pass up the chimney, which terminates in a short cone. A gauze diaphragm, having about double the area of the section of the chimney, closes the cone. Above is the perforated dome through which the escaping gases pass. The lamp gives a good light, the flame not being affected by oscillations of the lamp, nor by rough jerks in an upward or downward direction, and is considered to give considerable security in explosive currents. Its weight without oil is 2¾ lbs. With 8 per cent. of gas in the mixture forming the current moving at a velocity of 47 feet per second it did not produce an explosion when tested by the Royal Commissioners, but after being in the current 45 seconds the glass cracked, and at the end of 100 seconds was withdrawn in a dangerous condition. The principal defects pointed out by the Commissioners are (1) danger to the glass from heat produced by gas burning at the cylindrical strip of gauze immediately under it; (2) in currents of low velocity this ignited gas heats the lower edge of the glass strongly, and in currents of high velocity a stream of ignited gas passes completely across the lamp from the windward side and plays on the glass on the opposite side, causing the glass to crack; (3) the top of the lamp may be easily tampered with; and (4) the gauze at the outlet is liable to be obstructed by soot if the flame should smoke.

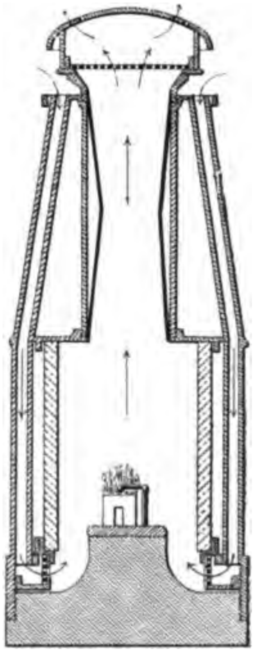

Fig. 522.—Gray Lamp.

The *Hepplewhite-Gray* is a modification of the Gray lamp just described. Fig. 523 is an illustration of Ashworth's patent Hepplewhite-Gray deputy safety-lamp. In it a flat wick is used as in the Gray, but two of the four tubes down which the supply air passes have openings in them close to their base. A suitably sized slide may be pushed up and down each of the tubes provided with openings, so as to cover them or leave them exposed to the surrounding air. In a later modification three inlet tubes are used; one of them, considerably broader than the others, serves the purpose of a reflector. The breadth of the wick has been increased to $\frac{9}{16}$ inch.

The cap at the top extremity of the lamp may be turned partially in a horizontal direction, and with it moves a plate which covers the upper ends of all inlet tubes at the same time, when the cap is turned in one direction, and leaves them free when turned in the other. By this means the supply air may pass from the surrounding atmosphere down the tubes from the top, or, it may be shut off from admission there, and allowed to pass through two of the tubes at the bottom. This arrangement renders the lamp an excellent gas trier, as by closing the lower openings any explosive mixture near the roof or in holes above the roof is discovered, whilst in sluggish currents, the air may be admitted through the lower openings in the tubes. When both top and bottom openings in the tubes are closed at the same time, the supply air is shut off entirely, and the lamp goes out immediately afterwards. This may become necessary when testing for an explosive mixture. The glass instead of being cylindrical in form is larger at the base, the top being the same size as the coned bonnet and gauze above it. A conical gauze cap is used in the Hepplewhite-Gray instead of the gauze diaphragm in the Gray shown in Fig. 522.

A shield outside the gauze cap extends from the glass to a point slightly above the gauze, and terminates in an outlet cone which reaches to the level of the shield-plate C., its upper extremity being mid-way between the two horizontal rows of perforations B, B′, by which the hood is pierced for the escape of the products of combustion. The shield plate C is of a size to cover the inlet openings. The circular row of holes in the crown of the lamp are of the same size as those in the shield-rim. A stiffened cover-plate is placed over, to protect the products of combustion at their point of exit from direct currents. By these additions and improvements the discharge is regulated as well as the inlet so that they bear what is considered a proper proportion to each other. The restriction of the discharge when the lamp is in use causes the top of the gauze to be kept in a bath of carbonic acid gas, so that if an internal ignition of gas takes place it will not continue in the lamp. The first test for fire-damp should be made with this lamp when either one or both of the shutters over the holes near the base of the inlet tubes are pushed up, leaving the holes open. The searcher then slowly and steadily raises the lamp, without lowering the wick flame until the presence

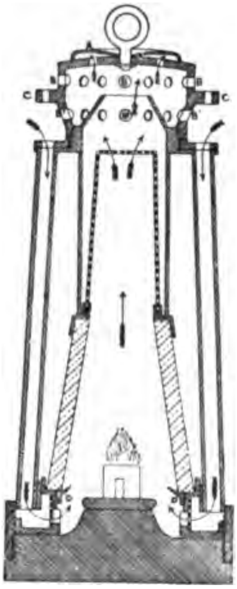

Fig. 523.—Hepplewhite-Gray Lamp.

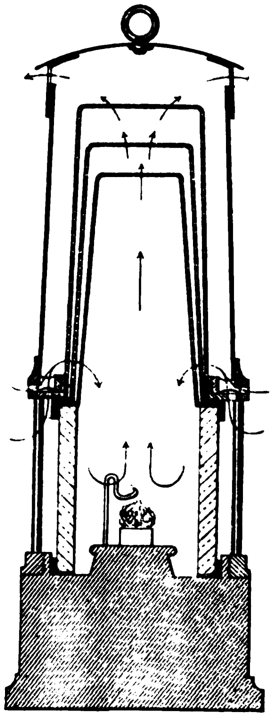

Fig. 524.—Marsaut Lamp.

of gas is observed by the "spiring" at the flame, or until the lamp reaches the roof, or cavity above. If gas is not found during this test, the holes are gradually closed by the thumb and forefinger of the hand holding the lamp when the supply of air to the wick flame is cut off from below, and any small percentage of gas in the roof or hole over the roof will be drawn down from the top of the tubes and cause the flame to spire. No lamp should be suddenly plunged into fire-damp. To facilitate the cleaning of the lamp, the ring D, which holds up the glass, is screwed on to the vertical plate forming the annular air inlet chamber. On unscrewing the lower gauze ring the inside parts of the lamp come out, without other labour, ready for cleaning. The light given by this lamp is good, and the conical glass admits of the light reaching the roof. In ordinary bonneted lamps the shields are larger than the glass below them and act like shades, which prevent the light from striking upward. The conical glass and retreating shield of the Hepplewhite-Gray admit the rays of light to the roof with but little obstruction without tilting the lamp. Its illuminating power therefore is not reduced during an examination of the roof, which can be carried out more quickly and satisfactorily than with any other safety-lamp.

It has been successfully tried in currents of high velocity.

The *Marsaut* Lamp.—M. J. B. Marsaut spent much time in making tests and observations with safety-lamps, and afterwards made many practical suggestions in consequence. He was one of the first to make a shielded lamp. The Marsaut is made with either two or three conical gauze caps. In Fig. 524 it is shown with three. It has a thick glass cylinder as in the Clanny lamp. The three gauze caps fit close together at their lower extremity on top of the glass, and gradually diverge from each other in proceeding upwards. The gauze caps are protected by a bonnet of sheet iron, screwed on a flange above the glass. Near the upper end of the bonnet is a gauze diaphragm, containing about 400 apertures to the square inch. Just above this diaphragm at the top of the bonnet and immediately below the top plate of the lamp, are a series of large holes through which the gases from combustion escape. Near the lower end of the bonnet are also a number of holes by means of which the feed air passes through the gauze caps immediately above the glass cylinder, and these openings are arranged so as to prevent the direct entry of horizontal currents of air into the lamp. No provision is made for preventing the removal of the bonnet. The gauze used in the caps has about 934 apertures to the square inch. In most other lamps the mesh of the gauze used is the same as that in the Davy lamp. A flat wick is used in the Marsaut lamp, which gives a better illuminating power than a round one. The lamp goes out in an explosive mixture. When used with two gauzes the lamp has an illuminating power of about two-thirds of a standard candle, but with three gauzes the illuminating power is reduced to about one-half of a standard candle. When used in an explosive mixture with only two gauzes, it is just possible for the flame to pass through the gauzes whilst the glass remains uninjured. When three gauzes are used the lamp is much safer, although the illuminating power is less. Its weight is about 2¾ lbs. without oil. When tested, the three-gauze lamp did not produce an explosion in an explosive current containing 8 per cent. of marsh gas moving at a velocity of about 50 feet per second. A two-gauze Marsaut under the same conditions exploded. The lamp is liable to become much heated owing to the gases from combustion impinging on the bonnets, which are in good metallic connection with other parts of the lamp. A considerable amount of heat is therefore conducted even to the bottom of the lamp when used where the air current is not sufficient to keep down the temperature. Other bonneted lamps become heated in the same way, but no objection can be urged against this so long as very volatile illuminants are not used.

The *Mueseler* lamp is constructed on a similar principle to the Clanny. It has a short cylinder of thick glass round the flame, above which is a gauze cap, Fig. 525. Immediately above the flame and extending within the gauze cap is a central conical metal chimney, the top of which in some forms of Mueseler lamp is covered with wire gauze, but in the Belgian pattern is open at the top as well as bottom. The chimney is carried by an attached ring of gauze fixed to its outer circumference between the top of the glass cylinder and the bottom of the gauze cap.

The feed air, as indicated by the arrows in the drawing, passes under ordinary circumstances first through the lower part of the gauze cap and then downwards through the horizontal gauze ring, continuing its course afterwards between the metal chimney and the glass cylinder. The inlet air has thus two obstructions to its passage, the outlet only one, if the chimney is not covered with gauze. The object of the chimney is to create a strong upward draught and to insure the inlet air being drawn down close to the inside of the glass cylinder and thus keep it cool. A great drawback is its liability for the light to go out if the lamp happens to be held with its axis at a slight angle to the perpendicular, as in that position the flame drives back the inlet air on one side and baffles the current. Fig. 525 shows the Mueseler lamp as made in Belgium, where it is constituted the legal lamp for use in fiery mines. The base of the chimney is ·85 of an inch above the top of the wick-tube. The gauze diaphragm which supports the chimney is 1·9 inch above the wick-tube. The height of the chimney is 4·6 inches, its diameter at the bottom being 1·15 inch, narrowing to 1 inch at the junction of the main cone, ·25 above the base, and diminishing to ·4 inch at the top. The diameter

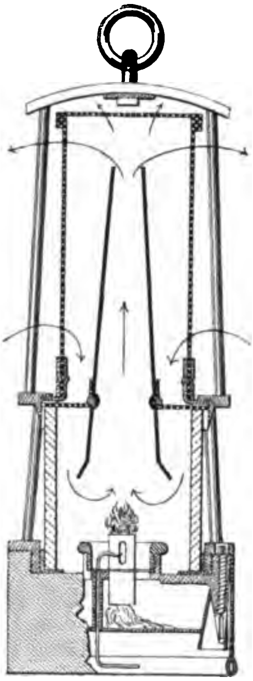
Fig. 525.—Mueseler Lamp.

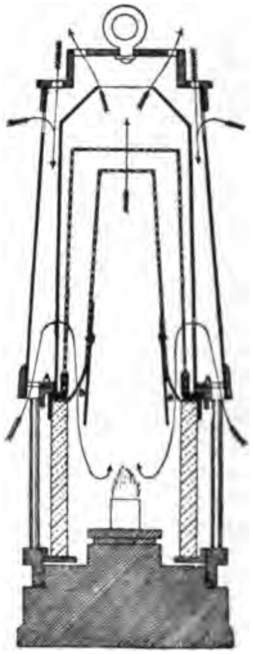
Fig. 526.—Ashworth-Mueseler Safety-lamp.

of the glass cylinder is 1·9 inch, and the gauze has about 900 apertures to the square inch. The top of the wick-tube is ·6 inch above the oil chamber. In some Mueseler lamps of Belgian construction a gauze with 1,024 apertures, and one with 1,296 apertures per square inch, are used.

If the lamp is held in a vertical position, with a flame of normal size during a test in a horizontal current, the gaseous mixture ignites under the gauze diaphragm, thus cutting off the air supply from the lamp flame, which consequently becomes extinguished. In weak currents the gaseous mixture also soon goes out, but continues to burn in strong currents, in the latter case endangering the glass. If a reduced flame is used the products of combustion do not fill the chimney and there is a danger of the ignited gaseous mixture extending to the upper part of the lamp. If the gauze cap outside the chimney is allowed to become filled with the ignited gas, whether the ignition is caused by a reduced lamp flame or by an oblique current acting on an ordinary flame and causing a portion of the feed air to reach the lamp flame by the chimney, the Mueseler lamp acts like a Davy in an explosive mixture, and will explode the surrounding current if at a velocity of about 6 feet per second.

A Belgian Mueseler tried in a vertical position in a horizontal current having a velocity of 47 feet per second, tested by the Royal Commissioners, became extinguished in a few seconds without exploding the surrounding gaseous mixture which contained 8 per cent. of marsh gas. Used under favourable circumstances, therefore, it is a very safe lamp.

When fitted with a bonnet to protect the gauze cap it is extremely difficult to ignite the gas in the top of a Mueseler lamp. The bonnet should closely fit the horizontal flange carrying it, and the air enter through a row of holes in the flange.

The weight of such a bonneted Mueseler lamp without oil is about 2¾ lbs.

The English-made Mueseler lamps have usual gauzes, and do not afford so much security as those made in the Belgian legal form, owing chiefly to the modified construction of the chimneys.

The Commissioners say: "As a general rule, if a bonneted Mueseler is exposed to a current with any velocity we could obtain, the gas ignites under the horizontal gauze diaphragm, and is speedily extinguished. It may, however, continue to burn, and in that case, as the lamp is ordinarily constructed, the glass is almost certain to be cracked in a short time." Again, in their report, they give as the sources of danger and the disadvantages arising from the use of the Mueseler lamp: (1) the glass cylinder is easily cracked by a blow, by the flame playing directly upon it, or by the impact of cold water while in a heated condition; (2) there are difficulties to be overcome in maintaining at all temperatures sufficiently tight joints where the metal and glass parts meet; and (3) difficulties arising from combustion and a tendency to smoke the glass and impair the illuminating power.

Ashworth's Mueseler, shown in Fig. 526, is an ingenious invention to guard against explosion in rapid currents, and has been successfully tested in explosive currents up to 100 feet per second. Its most striking feature is the double shield used to protect the gauze cap. The gauze diaphragm B is conical instead of horizontal, and the metal chimney of the Mueseler is replaced by a gauze chimney carrying at its lower extremity a very short inner metal one extending downward below the gauzes so as to increase the draught. The inner shield terminates in a cone which reduces the outlet passage above the gauze cap. This keeps the top of the lamp filled with carbonic acid gas during the combustion of the flame. If gas is ignited within the lamp the lamp flame is extinguished because the supply air is cut off. Air is admitted to the lamp flame as indicated by the arrows, from which it will be seen there is no direct entry. The outer shield is pierced by a row of seven holes near the top, the inner shield by a row of ten holes near the bottom. The entering air is thus deflected and prevented from striking violently against the gauze, through which it passes steadily. For gas-testing purposes there is a movable shutter A placed immediately above the horizontal row of inlet holes close to the top of the glass. When the shutter is closed the feed air is cut off from below and can only enter through the holes at the top of the outer shield and pass downward to the openings in the inner shield. By this means gas near the roof or in cavities above it is drawn down to the lamp flame.

The lamp burns badly in feeble air currents, and there appears to be some difficulty in lighting it, owing to the circuitous course of the feed air. As the lamp gets hot a proper circulation takes place in strong air currents.

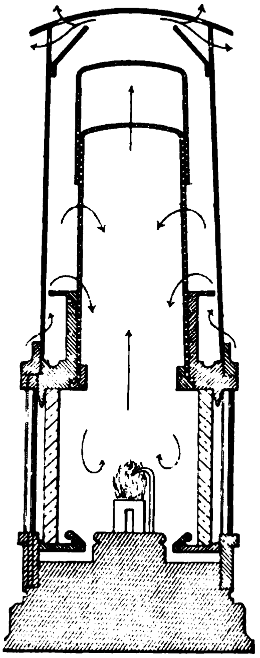

Fig. 527.—Evan Thomas's Lamp.

Fig. 527 shows *Evan Thomas's* No. 7 lamp, of which the Commissioners speak very highly. In principle it is a Clanny lamp, but is bonneted and a great improvement on the ordinary Clanny form of lamp. Within the bonnet is a brass tube, one inch high, which fits the main gauze cylinder closely at its lower end. This tube terminates at its upper extremity in a horizontal brass flange, which extends nearly to the bonnet. Between the edge of the flange and the bonnet is an annular space one-sixteenth of an inch wide. The inlet air reaches this annular space after passing through horizontal slits in the bonnet near its lower end. The feed air in continuing its course passes through the main gauze cylinder immediately above the brass tube and descends to the flame. The products of combustion escape through holes near the top of the bonnet, which are protected by a shield secured to the bonnet. The main gauze cylinder is protected by a gauze cap closely fitting it. The bonnet is locked by a sliding bar, which cannot be withdrawn until the oil chamber is removed. The weight of the lamp without oil is 3¼ lbs.

In case of an internal explosion, the gas only burns in the gauze cylinder between the top of the brass tube and the lower edge of the gauze cap, so that only a small area of gauze is heated.

The Commissioners say:—"No. 7 seems to be a most efficient lamp. The flame is bright and remarkably steady in the strongest air current we can produce. In an explosive atmosphere, moving with a velocity of 3,200 feet per minute, it showed no signs of danger after an exposure of seven minutes and forty seconds. The gas continued to burn in the gauze cap, and a portion of the gauze quickly became red hot, but its temperature appeared to be considerably below that required to ignite the gas mixture. With current velocities down to 400 feet per minute the gas always burned continuously in the gauze, but the latter did not become visibly hot until the velocity approached 1,600 feet per minute. The lamp flame was in all cases extinguished in the gas mixture in a few seconds."

Again: "In this lamp the quality of safety, in a pre-eminent degree, is combined with simplicity of construction and with illuminating power at least fully equal to that of any of the lamps hitherto in general use, and there is no probability of the flame being extinguished under any circumstances attending ordinary use."

In another section of the Commissioners' Report appear the following remarks:—"This lamp seems very safe in all currents in which it was tested. In air moving with any velocity up to 3,500 feet per minute the flame burns very steadily when the lamp is either erect or inclined. The flame is scarcely affected by violent oscillations of the lamp or by rapid motion up and down in a vertical direction, and it is not extinguished by inclining the lamp until the latter is nearly horizontal."

The *Clifford* lamp, as shown in Figs. 528—530, is new since the publication of the Commissioners' Report. It is to be called the Phœnix lamp, is novel in construction, and is designed to give great resistance to the passage of flame in high current velocities. In its construction, the usual form of having a glass cylinder surmounted by a cylindrical gauze has been entirely departed from.

VERTICAL SECTION

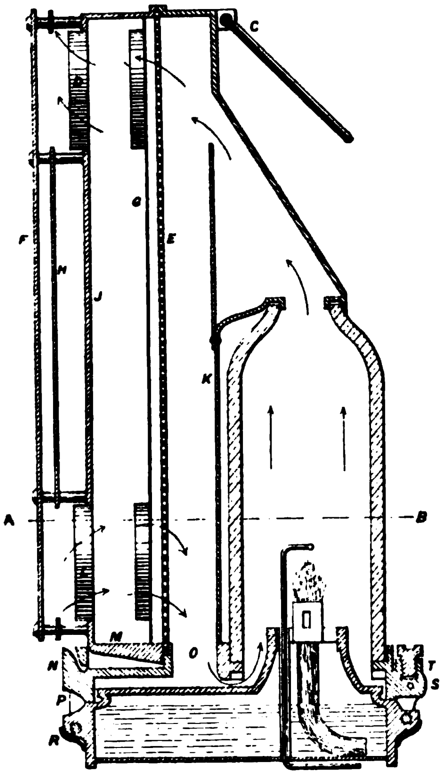

Fig. 528.

TRANSVERSE SECTION ON LINE A.B

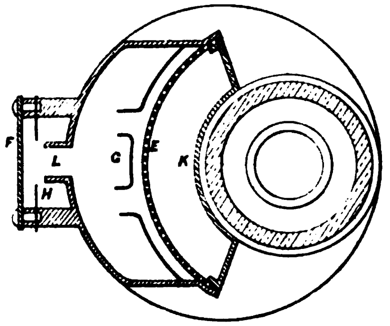

Fig. 529.

PLAN OF OIL CHAMBER AND ATTACHMENT

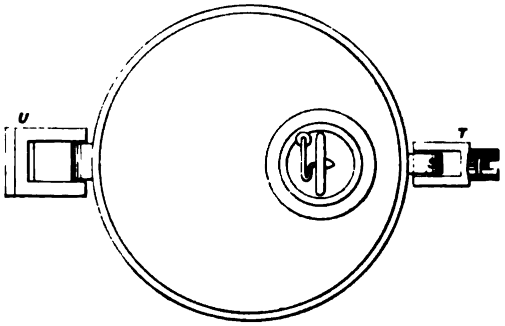

Fig. 530.

The Clifford Lamp.

Fig. 528 is a vertical section; Fig. 529 a transverse section on the line A B; and Fig. 530 is a plan of the oil chamber and attachments with the upper portion of the lamp removed. The glass cylinder surrounding the flame is firmly held in position by asbestos washers. The upper portion is contracted in diameter and is covered by a metal hood, which forms part of a segmental pillar or box, K, cast in one with the base portion P, or attached to it in any other reliable and substantial manner. The base is fitted tightly against the oil chamber by means of the shackle hinges U and T, which are of special construction, and jointed in a thoroughly safe way. The gauze E is a strip bent to fit the segmental portion (Fig. 529) of the box pillar K, and protected by three bonnets, J, G, F. The main bonnet J hinges at C to the pillar portion K, and, when unhinged, access to the gauze is obtained for the purpose of inspection and cleaning as required.

The gauze is forced between the segmental surfaces of the bonnet and the pillar K. The shackle U has an inclined face, which is made to press against the projecting inclined piece or wedge N, so that when closed by hinging from R, the portion M of the bonnet will be firmly pressed against the gauze to make a joint. The gauze may be secured to the segmentary edges of the box K in many ways, any of which renders it easily got at for inspection. After the yoke T is slipped over S, it may be strained up by screw or fastened with a lead rivet.

The air supply enters the bonnet J at the passage L, which is protected by the metal plate F so as to prevent direct entry, and passes on through the space formed between the bonnet J and shield G. It reaches the gauze E through the two apertures in the internal shield G which are opposite to L, but not in direct line with it. The air then descends into the space O and passes under the glass by the annular space shown, continuing its flow upwards to the flame. The hot products of combustion then ascend and pass out of the glass cylinder and reach the upper portion of the space enclosed between the gauze E and the pillar K through the hood. The outward flow is continued, as shown by the arrows, through apertures in the shield G and the passage D in the bonnet J, which is protected similarly to the air inlets.

It is claimed for this lamp that in it the light from the flame is free to strike upwards with but little obstruction above, and whilst the advantage of air supply from below the flame is obtained, the gauze is not placed under the wick, and thus risk of clogging it by spilling the oil is avoided. The light given is good, as a plentiful supply of air enters the lamp and the light illuminates the roof as well as the sides of the workings. It is not sensitive to tilting or violent movement. The bonnet arrangements prevent the possibility of a current of any velocity from impinging directly upon the gauze, and thereby renders the lamp safe in explosive currents. It is said that the lamp has withstood an exceptionally high velocity test.

It is too early to form an opinion of this lamp, as it must be practically used for some time in order to thoroughly test it; but judging from the drawings it does not appear perfect in construction. The shoulder formed on the glass cylinder where it is reduced in size is not covered by the hood, and although this may to some extent allow the light to strike upwards, it leaves the glass exposed to falling stones or coal from the roof or sides, a small piece of which is sufficient to fracture it. No upright bars or pillars are placed for protecting the glass cylinder, which renders it less secure on that account, and in wet places the lamp would be very liable to be cracked by water dropping on it.

McKinless's gauzeless safety-lamp, shown in Fig. 531, is another novelty in construction, and, like the Clifford lamp last described, is new since the publication of the Royal Commission's Report on Accidents in Mines.

The most striking feature of this lamp is that gauze is entirely discarded in its construction. The supply air is admitted through a large number of very small holes drilled in a belt or band, J, above the glass cylinder K and middle ring of the lamp H, and after passing downwards between this belt and an inner conical chimney, E, reaches the combustion chamber. At the top of the conical chimney E is fixed a cap, E′, to collect the unconsumed particles of carbon or soot.

The products of combustion pass upwards through the conical chimney and cap, and thence through a large number of very small holes drilled in a cap, B, and escape near the top of the lamp.

The outlet cap B is fitted and riveted to a cylindrical tube O, the lower end of which fits into a collar of the supply air belt J, on which collar the diaphragm of the conical chimney rests. This diaphragm separates the inlet and outlet currents, and renders certain the passage of the feed air into the combustion chamber. An upper ring, C, is placed level with the top of the bonnet, through which the outlet cap is visible. The products of combustion thus escape without re-mixing with the supply air.

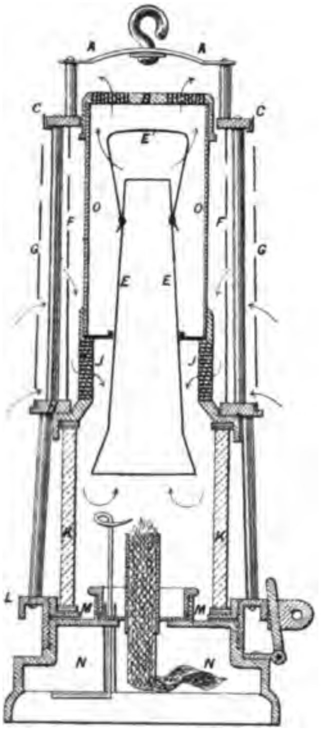

Fig. 531.—McKinless's Gauzeless Safety-lamp.

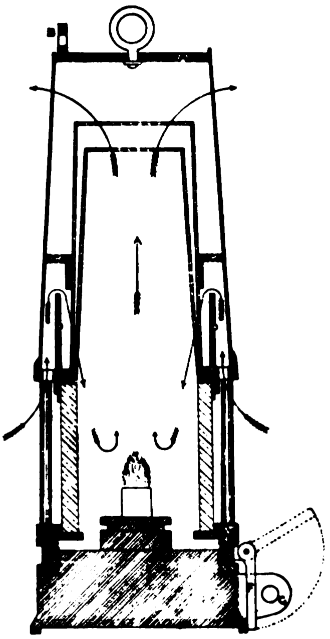

Fig. 532.—The Howat Deflector Safety-lamp.

The feed air belt J is protected by two shields, G and F, which are placed outside it. The shields have large openings to allow the free ingress of air. The arrangement of these openings prevents the force of a current from striking directly against the small feed air holes and so disturbing the flame. The outer shield is easily removed when damaged, and after being repaired can as easily be re-fixed, being fastened by simply turning in four or five tongues of its own metal under the upper ring C. The combustion chamber and the drilled belt J are kept securely in their place by the ring H; L is the bottom ring which secures the whole to the oil chamber N; K is the glass cylinder, and M a ring for holding the glass in its position. The metal in the belt J and in the outlet cap B is $\frac{1}{8}$th of an inch thick, and each hole is therefore $\frac{1}{8}$th of an inch long and $\frac{1}{26}$th inch diameter. The three lower rows of openings in the outer shield G may be covered with asbestos cloth or leather, and the supply air then enters at the top of the lamp; thus used it is an excellent gas tester. The lamp weighs 3 lbs. It is stated that it is not sensitive to tilting, swinging, or jerking, and that it has been tested in explosive mixtures of high velocity without the outer gas being fired.

This lamp requires to be used for some time before forming a decided opinion about it; it should, however, be remarked that the gauze is not the only weak part of a safety-lamp.

The Howat Deflector derives its name from the "deflector" used in it to throw the feed air down to the flame as it enters the lamp. Fig. 532 is an illustration of the lamp. The supply air passes upward through a series of holes in the horizontal flange carrying the shield as shown by the arrows. A vertical cylinder of brass, D, $1\frac{1}{4}$ inches high, guides the air upwards, and prevents it from passing directly through the base of the two gauzes. About $\frac{1}{4}$ of an inch above D is the bottom of the angle ring C, the top of which is horizontal and extends right across between the outer gauze cap and the shield so as to completely fill the space, whilst its vertical flange extends downwards in close contact with the outer gauze and terminates near the top of the brass cylinder D. The cylinder D is placed about midway between the gauze and the shield, so that there is a space for the air on either side of it. This on reaching the deflector C is thrown down through the opening around the top of the cylinder D through the two gauzes on to the lamp flame. After the lamp has burned a short time the deflecting ring and gauzes become hot and cause a good air supply to enter for the combustion of the flame, which benefits by the inlet air being directed to it. The lamp top, A, is made solid, the products of combustion passing up through the two gauze caps and out at the openings in the top of the shield, shown by the arrows. The shield is securely fastened in position by means of a lead rivet, B. Three objections have been made to the use of a fixed bonnet, viz., that the collier after receiving his locked lamp is unable to examine it to see that all the gauzes are in their proper position; that in case of an internal explosion in which ignited gas reaches the top of the lamp, he is unable to see any resulting alteration in the state of the gauze; and that it increases his difficulty of examining the roof. The lead rivet fastening to the bonnet on the deflector lamp removes the first objection, as the collier can first satisfy himself that the gauze caps are not omitted before the bonnet is locked. It, however, necessitates the use of two lead rivets, as one must be used for the oil reservoir lock. The oil chamber is provided with a projecting boss, G, to receive a hasp, F, hinged at the top and fixed to a loose collar, E, surrounding the oil vessel. This collar can be turned round so as to compensate for the wear of screw on the oil vessel resulting from ordinary use. By this means the hasp F can always be brought exactly over the projection G, and then dropped over it. A hole is bored in the projection G to receive the lead rivet horizontally after the hasp is lowered.

The weight of the deflector lamp without oil is about $3\frac{1}{4}$ lbs.

The *Thorneburry* lamp is the invention of Dr. James Thorne as a result of suggestions made in the Report of the Royal Commissioners in 1886. It has two concentric glass cylinders, and through the annular space between them the inlet air, after first entering through perforations in the flange carrying the bonnet, descends to the gauze F (see Fig. 533), continuing through a hollow ring to the combustion chamber G. The entering air thus passes through three gauzes, viz., an outer one at B, which is a short piece used as an additional protection against rapid currents, the main gauze A, and the horizontal one F at the base of the glasses. The products of combustion ascend the metal chimney C, which leads directly from the inner glass, through the top of the main gauze A, which leads from a ring above the outer glass, through openings in the top of the metal shield which encloses the whole.

A heavy mineral oil having a specific gravity of not less than ·83, with a flashing point of 250° Fahr. closed, Abel's test, is burnt in a cone similar to those used in paraffin lamps, and is said to be as safe as animal or vegetable oils, or to the mixture prescribed by the Mines Commission, and to burn without smoking. A flat wick is employed, and the height of the flame is regulated by a simple rack and pinion movement controlled from the outside by means of a small key fitting the square shaft of the pinion. A pricker is required to keep the wick of the usual types clean, but as no carbon cap forms on the wick of the Thorneburry lamp, the services of a pricker are dispensed with. A very superior light is given by the lamp, which will burn for 14 hours in actual use in the mine with the flame at a uniform normal height.

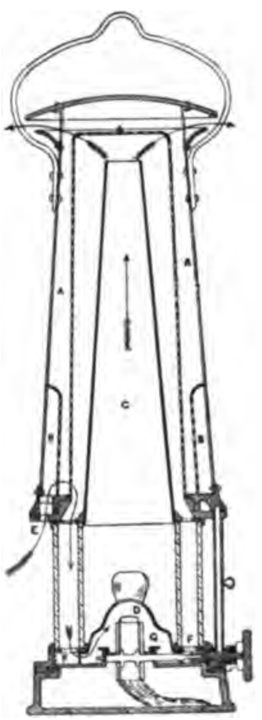

Fig. 533.—The Thorneburry Safety-lamp.

Objections have been made against the use of lamps with two concentric glasses on account of the inner glass being so near the flame that the additional heat from an internal explosion is almost certain to crack it, thereby rendering the lamp much less secure, and also on account of the difficulty in making accurate and tight joints with them. In cleaning the lamp the glasses have usually to be taken out and much time is occupied in refitting the parts of a large number of lamps. To overcome the latter objection the glasses in the Thorneburry are left undisturbed when the lamp is taken apart to clean, and, with regard to the former, the cylinders are of specially-prepared virgin glass properly annealed. This, it is said, unless subjected to sudden changes of temperature, may melt but will never crack by the application of heat.

An internal explosion of gas extinguishes the lamp flame, after which the ignited gas may remain for a few seconds burning at the wire gauze ring F protecting the air inlet, but this in turn soon becomes extinguished owing to the nitrogen and carbonic acid gases given off by combustion in the cone. Slow currents do not cause dimness as in ordinary lamps, and it bears the swinging about incidental to the use of a safety-lamp in the mine. It is simple in construction, containing no more parts than an ordinary Mueseler, which facilitates cleaning, and the gauze and chimney can be seen without unlocking the lamp. The lamp has been tested in horizontal and inclined explosive currents, the velocities of which varied from 10 to 50 feet per second, with the result that it was always automatically extinguished without firing the external mixture.

An objection to the lamp is its weight, which when ready for use is 3 lbs. 10¼ oz. It also gets very hot if allowed to stand in a still atmosphere.

Marshall's Automatic Extinguishing Lamp.—This lamp, shown in Fig. 534, is new since the Report of the Mines Commission, 1886, and is remarkable for the ingenuity of its design. It requires no bonnet, so that the gauze cylinder is fairly open to view, and a partially unobstructed examination of the roof can be made with it if long glasses are used. An outer "sleeve gauze" cylinder fits closely over the main gauze from the lower end of the latter to within about ¾ of an inch of its top. This sleeve gauze slides over the inner one and can be raised in a slow current or in descending the pit, thus allowing the feed air to pass more freely under it, having then only one gauze to go through. In strong currents the sleeve gauze can be lowered, and the double gauze then presented to the air checks what otherwise would be a more violent entry. If desired, the sleeve gauze can be removed from the lamp and its use dispensed with.

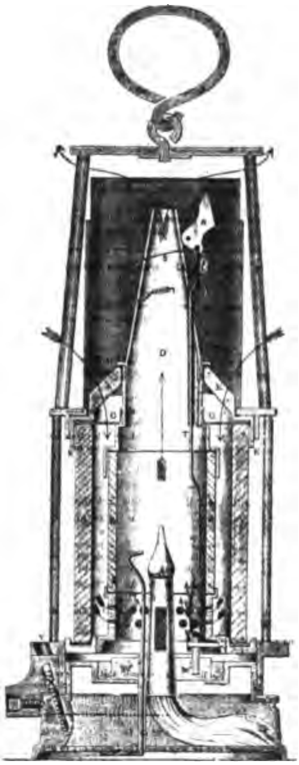

Fig. 534.

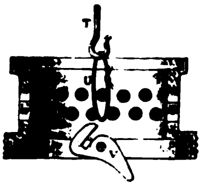

Fig. 535.

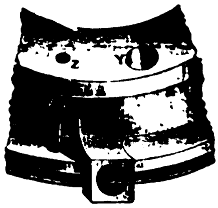

Fig. 536.

Marshall's Automatic Extinguishing Lamp.

The feed air enters at the bottom of the gauze cylinder as shown by the arrows; passes through openings between the connecting bars in the rings K and the horizontal gauze diaphragm G; continuing its descent past the base of the metal chimney D, and through the annular space between the outer glass A and inner glass B, below the latter of which it is drawn through a double row of gauze protected openings in the metal inlet ring C to the flame. The inlet holes are about thirty in number and ⅛-inch in diameter. The products of combustion ascend the metal chimney D, and after passing out at the top of the main gauze above, are deflected to the side by the solid flat brass top, as shown by the arrows. The upper protecting bars around the gauze cylinder support the lamp top. The most striking characteristic of the lamp is its double-acting extinguisher, which is automatically closed by the ignition of an entering gaseous mixture, or mechanically in the act of unscrewing the oil vessel. This shut-off or closing appliance consists of a cap, R, hinged to and closely fitting the open top of the chimney, D; a steel spring, S, fixed in a socket at one end on the outside of the chimney, the other end being fastened to the wire loop *t*; a main wire T passing through a tube in the chimney, the upper extremity of which is bent over to hold securely the wire loop *t*, while its lower extremity is hooked to receive a loop of tape or string, U; a wire loop *t* passing through a hole in the cap R and connected with the moving end of the spring S; a hook and lever, V, which, when the lamp is in ordinary use, holds down the loop U so as to resist the pull of the spring above, but which is shown detached in the enlarged illustration, Fig. 535; two raised lever blocks, W, fixed next the wick tube, which in screwing up the oil vessel push aside and pass the lever V, but, in unscrewing the oil vessel, press against the lever V so as to displace its hooked end and release the extinguisher, which rises in response to the tension of the spring and covers the chimney top, thus putting out any flame within the glass or chimney before the lamp can be opened ; a check screw X is screwed into the oil vessel through a hole Y at the last turn of the oil vessel, so that one turn of the oil vessel is required before the check screw X can be extracted, thus extinguishing the flame ; a screw-hole Z, shown in the enlarged illustration, Fig. 536, for the check screw when it is not in use. If desired, this screw, which is more difficult to open without a proper key than an ordinary screw lock, can be discarded for lead rivet locking, or the use of the extinguisher and the mechanism actuating it may be dispensed with in favour of a lead rivet lock, thus considerably reducing the cost of manufacture and complication in fitting up when cleaned.

When the explosive mixture ignites within the lamp, it burns inside the gauze-covered inlet ring surrounding the wick tube, and therefore must burn the string loop U, which is consumed in a few seconds. The wire T is thus released; the spring S instantly turns the cap R over the chimney top, and at once extinguishes all fire by shutting in and throwing down the carbonic acid gas. As an additional precaution an ordinary screw-lock is fixed where shown at the side.

The two rings, K, are made in one casting with connecting pieces between them. They support the chimney in place, and prevent it from tilting, while the horizontal gauze, G, causes the tension and allows the inner glass to expand with the heat. A close joint between the top and bottom ends of the inner glass, B, and the metal rings, is thus always maintained under all changes of temperature. The rings, K, and chimney with extinguisher attached, are so combined by a screw ring as to form one fitting only. The outer glass is kept cool by the feed air.

There is no smoke gauze cap over the main gauze, which is of copper, as is also the round wick tube. All gauzes used are of ordinary mesh, having 784 apertures to the square inch. The upper half of the chimney is made of copper, while the lower half is of brass. The outer glass A is nearly ¼ of an inch thick, the cylinder being 2¾ inches in diameter outside. The inner glass B is 1¾ inches in diameter outside. The outer glasses are 3, 3½, 4, and 4½ inches long; with the longer glasses, the chimney and upper parts are shortened. The length of the inner glasses varies from 1¾ inches to 3¼; they should be well annealed. A stock of string loops must be kept ready for use, but a new one is not required every day. They are easily prepared for use by being tied upon a tapering peg nailed to a bench.

To clean the lamp, it is taken apart by unscrewing the bottom ring plate, when the whole of the parts come out together. In fitting up, a small "setting hook" is used for attaching the string loop U to the extinguisher wire T and hook V, the lamp being held upside down for the purpose. When the loop has been attached to the wire T, and is pulled slightly, the cap comes off the chimney. The loop is then secured to the hook on the lever in the position shown in the drawing.

In the unlikely event of the extinguisher failing to act, or in the still more unlikely event of the gas flame passing between the glasses, the collier can turn the lamp bottom upwards, and shut in the carbonic acid gas, which effectually extinguishes the fire-damp flame in from 2 to 6 seconds in currents of any velocity.

The weight of the lamp with oil is about 4¼ lbs., and it well stands the jerks, oscillations and swinging inseparable from its use in the mine without disturbing the light, which remains clear and steady in strong currents of air.

In explosive currents of great velocity the lamp is very safe and has been tested in currents varying from 15 to 50 feet per second without failure. The lamp was exposed to horizontal explosive currents, to currents inclined upwards, and to currents inclined downwards, and at the expiration of a few seconds, the combustible connection with the spring was burnt, the extinguisher operated and put out all flame, without an external explosion taking place, and without cracking he inner glass, which is able to resist a much greater heat than that necessary to burn the loop spring. The lamp also successfully stood the above tests without the extinguisher attached.

Notwithstanding its safety in rapid currents and its ingenious design, the lamp is not likely to come into general use on account of its weight and cost, and the complication in construction and fitting up which prevents it from being quickly cleaned and put together. Some anxiety, moreover, is naturally felt respecting the wear and tear of all such mechanism, even when comparatively simple, and a strong feeling of uncertainty as to its always working satisfactorily. The lamp, however, if well constructed and fitted with long glasses, is well adapted for illuminating places where it could be left hanging without further attention. Its usefulness in this respect might be increased by fitting a circular reflector on the upper ring of the lamp frame so as to reflect the light downwards. An excellent light would be obtained by making the lamps a little larger in diameter and using broad flat wicks, which can be regulated by a screw and cog from the bottom of the oil vessel. Such regulators are usually very objectionable, because of the danger of their being accidentally moved by a jerk, or a touch of the sleeve, but this danger would be obviated if the lamp were suspended in a safe place.

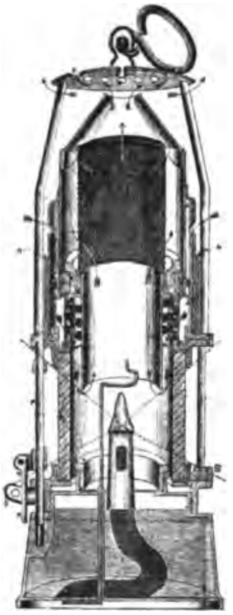

Fig. 537.—Marshall's Second Lamp.

Marshall's Second Lamp.—The objections raised against the lamp just described cannot be urged against this one, which was invented and patented by Mr. Thomas Marshall in 1887. Fig. 537 shows the lamp, which has a steel shield of peculiar construction. The lower end of the inner half of the shield which forms one of the most important features of the lamp fits closely round the top of the gauze-covered inlet ring. The inner half of the shield is fitted with a conical outlet top firmly riveted to the shield, to regulate the area through which the products of combustion escape. The outer half of the shield is of larger diameter than the inner half, and is pierced by a double row of holes, which admit the feed air. Just above these inlet holes a shoulder is formed on the outer half of the shield, and it is here riveted to the inner half. An annular space is thus left between the two halves of the shield for some distance. The ingoing air, after entering at the inlet holes in the shield as shown by the arrows, descends through the annular space formed in the shield to the metal inlet ring outside the chimney. The inlet ring is placed just above the glass and has four rows of gauze-protected circular openings, each hole being $\frac{1}{8}$th of an inch in diameter. A short outer gauze cylinder encloses the whole of the inlet holes as an additional protection. The feed air continues its course through the openings in the inlet ring and down between the top of the glass and the cylindrical metal chimney to the flame of the lamp. The products of combustion pass up the metal chimney, which terminates above the inlet ring, and up through the gauze cylinder, which fits over the metal chimney. After leaving the main gauze they pass through the outlet cone surmounting the shield, then dividing, escape freely through perforations in the flat lamp top and out at the sides as shown by the arrows. An outside flange encircles the chimney near the top, and the main gauze cylinder fits closely around the chimney, with its flange resting on that formed on the chimney, which again rests on a flange inside the top of the gauze-covered inlet ring. The chimney and main gauze are thus supported and are kept steadily in place by a screw ring which is passed over the gauze cylinder and screwed into the top of the gauze-covered inlet ring.

The bottom of the shield is riveted to a ring which screws to the lamp frame. This screw ring supports four pillars, which are bent inwards above the shoulder formed on the shield to carry the top of the lamp and the handle. The close joint between the lower end of the upper half of the shield and the inlet ring, together with the flange on the latter, effectually separate the ingoing air from the outgoing gases. A round wick-tube is used, and the oil chamber is screwed on to the upper portion of the lamp in the usual way, but the locking device is a great improvement on the screw lock. It consists of a pillar lock P, which locks both the oil vessel and the shield. A hasp, R, is fixed to the pillar P, and to lock the lamp this is dropped over the projecting boss O. This can only be done when the pillar P is raised or lowered to the proper level, and then the upper end passes through a hole in the flange, which carries the shield so as to prevent it from being unscrewed, while at the same time the lower end enters partly into a hole bored in the rim of the oil-chamber. The projection O has an eye in it for the reception of the lead rivet when the hasp is turned down sufficiently for the purpose. If desired, a padlock may be substituted for the lead rivet, or a screw lock hasp can be hinged to the pillar P, it being slotted to pass over the projecting boss O, and the point of the screw locking sideways through the eye in the projecting boss O. Screw locks and padlocks are, however, easily picked and re-locked. To open the lamp, the lead rivet is removed which allows of the hasp R being raised clear of the projecting boss, and then the pillar lock P can be raised and lowered so as to permit the oil chamber and shield to be unscrewed. The oil vessel and shield rims can have a series of holes or longer slot holes in them to compensate for wear of their screw-threads. A single glass is used, and in order to overcome the danger of its working loose when screwing on the shield, a screw-lock S is provided in the lower ring of the lamp. The point of this screw when locked is against the side of the angle ring used for fixing the glass cylinder, thus removing a great source of danger found in some safety-lamps. An asbestos washer is placed above and below the glass cylinder.

Resting on the flange which supports the shield is a vertically movable steel band N, which in one position is in close contact with the lower half of the shield to a point just below the inlet holes where the band ends. This forms the "shut-off" to be used when gas testing, if there should be an internal explosion. The shut-off is raised by hand until it covers the inlet holes so as to prevent further entry of the gaseous mixture, following which the lamp is extinguished, or the same effect is produced by turning the lamp bottom upwards. The locking of the shield and oil vessel may be left until the last moment so as to give the examiner or the collier an opportunity to unscrew the shield and examine all the gauzes which are then exposed to view, and if necessary to tighten up the main gauze. With a little ingenuity the gauze and glass may be cleaned and examined without removing them. When the shield and oil vessel are unscrewed, a hand brush can be passed through the chimney to clean the gauze, or the chimney and main gauze can be removed from the top when the shield is unscrewed without removing the glass, which may remain in until it is broken or cracked. An advantage claimed for this lamp is that if tilted the flame burns low, and will therefore not crack the glass; upon righting the lamp the flame recovers itself. The lamp is very small and handy. In the smaller of two sizes made, the glass cylinder is 2 inches high and 2 inches in external diameter, the thickness of glass being $\frac{3}{16}$ths of an inch. The weight of this lamp without oil is 2¾ lbs., and the height is about 8½ inches to the flat top. A size larger has a glass cylinder 2¼ inches high, 2¼ inches in outside diameter, and weighs without oil about 3¼ lbs. The smaller lamp has been tested in explosive currents of great velocity, which it withstood without failure.

This lamp as well as the one last described may be made by any lamp-maker without permission of the inventor. But in order to ensure safety and satisfactory illuminating power, further directions, which the inventor will be happy to supply on application, must be strictly adhered to.

Porch or Pit-bottom Safety-Lamp.—Fig. 538 shows a bonneted Clanny or other glass lamp made by Messrs. John Davis & Son, Derby. In it an admixture of one-third best paraffin and two-thirds best colza oils is used, and the illuminating power of the lamp is increased by the introduction of a second flat wick, each being ¾ of an inch wide. The enamelled steel reflector shown in the illustration is used to throw the light down, as it is suspended in a high position.

Many safety-lamps have been invented, besides those here described, which give a large margin of safety in proper use. These, however, have been chosen as illustrations of the best lamps of various types now in use.

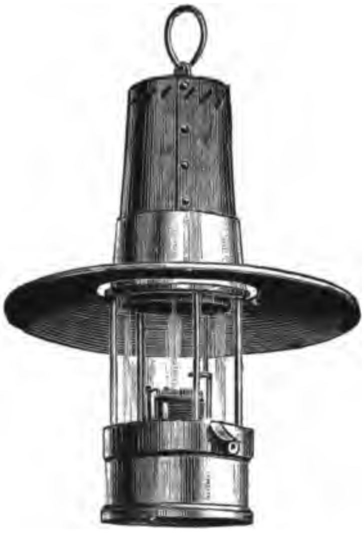

Fig. 538.—Porch or Pit-bottom Safety-Lamp.

In all oil safety-lamps using a glass cylinder, there is a possibility of the glass becoming cracked, or injured. This may be done by a blow, by carelessly holding the lamp out of its erect position so as to allow the flame to play directly on the glass, or by cold water coming into contact with the glass when the latter is more or less heated.

The cost of oil and wick for safety-lamps is about one halfpenny per lamp per shift of eight hours, but the cost varies with circumstances.

Locking Lamps.—All safety-lamps have locks to them. The oldest form of lock is still used on some lamps, notwithstanding its acknowledged insecurity. It consists of a screw bolt with a square head which is turned by a key until it has entered a hole bored in the oil chamber, which is supposed to prevent it from being unscrewed until the screw bolt is withdrawn. Unfortunately, this can be easily done without the application of a key, and if the re-lighting of a collier's extinguished lamp necessitates the loss of an hour's work or more, he is often tempted, at all risks, and in defiance of the Coal Mines Regulation Act, 1887, to resort to any ready means of withdrawing the screw bolt. This may be done by a couple of nails filed to fit the bolt, by pieces of wire bent for the purpose, or even by the point of a pick. After a few months' use a screw-lock often becomes so insecure that a sharp tap or a violent shake will so loosen the screw that it may be unscrewed by the fingers. A lamp in this condition cannot be said to have been *securely locked*, as it requires to be to comply with the 10th General Rule of the Coal Mines Regulation Act, 1887. Reckless tamperings with locks have frequently been proved by evidence given in the police courts. Ignorant men and boys, having screw-lock safety-lamps placed in their keeping, frequently fail to appreciate the danger attending the opening of their lamps and the exposure of an open light in the workings. The flame of the Davy lamp has also been drawn through the gauze by placing a tobacco pipe close to the gauze and sucking it. The same thing may be done with an unbonneted Clanny, if the wick be first raised so as to abnormally enlarge the flame by sucking the pipe against the gauze immediately above the glass. Possibly the knowledge of these facts made it seem useless for many years to introduce better locks, but now there are many good lamps, the shields and locks of which are not so easily tampered with.

For the most part, however, they afford little more security than the original device of the screw-lock. Many of them can be opened by easily extemporized mechanical means, while others are rather more difficult. It is easy to understand the temptation to tamper with the lock which the miner feels, whose lamp burns faultily or is accidentally extinguished. The legitimate method of opening and of readjustment may be available only on the surface, and that possibly a mile away, involving the loss of an hour's work, besides the toil of going to and fro. But even this, while it explains, will of course never justify the picking of a locked lamp in a fiery mine where the momentary exposure of a naked flame may cause a fearful explosion involving the loss of hundreds of lives, and a vast destruction of property. The difficulty would be met, and the temptation indefinitely lessened, by having relays of lighted lamps at stated places which would be available to the miner in case of need.

Most of the improved locks are supposed by the inventors to require the locking bolt to be withdrawn by the aid of an air-pump, or powerful magnet or water-pressure, it being previously kept in position by a spring. Some even of these, however, can be overcome without the proper appliances.

Purdy's Lamp Lock.—Purdy's lamp, of which the Royal Mines Commissioners in 1886 speak highly, has a pneumatic lock of ingenious device, which the inventor thus describes:—" A small cylinder is fitted at the side of the oil vessel, having a piston and rod, or bolt, which passes through and projects above the rim of the oil vessel. This rod or bolt is held up by a spring, after the principle of a cornet valve. When the top of the lamp is screwed on to the bottom, the piston bolt becomes inserted in a recess in the upper neck of the lamp, which comes opposite to the bolt at the moment of contact of both parts of the lamp. For the purpose of unlocking, a hollow screw is fitted in the side of the cylinder, which can be withdrawn only so far as to admit a small hole in the side of the hollow screw becoming accessible to an air-pump. When it is desired to draw down the bolt and unlock the lamp, the hollow screw is partly withdrawn and inserted in the mouth of the air-pump, by one stroke of which the air is exhausted, and by a simultaneous turn of the upper part of the lamp the two parts are disconnected."

Craig and Bidder's Lamp with Magnetic Lock.—This is one of the oldest, if not the first, of the magnetic locks for safety-lamps, it having been patented in 1869, and is described in the Transactions of the North of England Institute of Mining Engineers, Vol. XIX., p. 15-16. One arrangement of the lamp lock is shown in Figs. 539 and 540. The gauze frame is screwed on to the oil vessel A, by the ring C, in the usual way. This ring has a hole, B, bored partly through the under side of the metal, so as to receive the bolt D, attached to a piece of soft iron F, and pressed upwards by the spring J. At the last turn of the oil vessel screw, the bolt is forced by the spring into the hole B. In the brass bottom of the oil chamber M are two circular pieces of iron, HH, let into it, which correspond to the two poles KK, Fig. 541 of an electro-magnet LL. To unlock the lamp, the circular pieces HH are placed on the top of the corresponding circular projections KK, Fig. 541 of the magnet LL. The electric circuit can be completed by means of a knob placed conveniently for the operator, which, when contact is made, renders the projections KK the poles of a powerful magnet. The magnet attracts the bar F, overcomes the resistance of the spring J, draws the bolt D clear of the hole B, and thus permits the oil vessel to be unscrewed.

Another modification of the lock is shown in Figs. 542 and 543. In this the poles of the magnet are closer together. The bolt D is attached to the soft

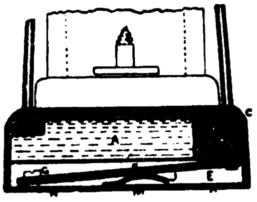

Fig. 539.

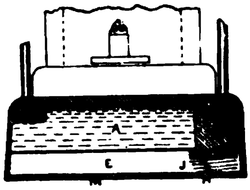

Fig. 542.

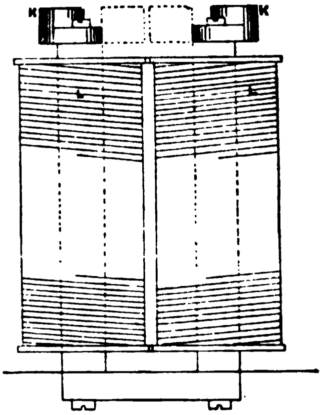

Fig. 541.

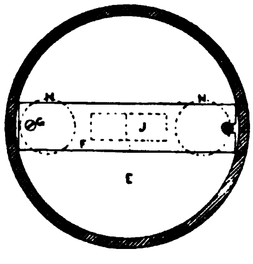

Fig. 540.

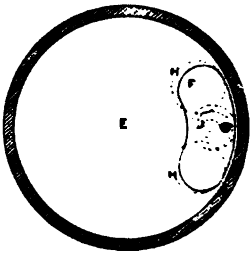

Fig. 543.

Figs. 539—543.—Craig and Bidder's Electro-magnetic Lock for Safety-lamps.

iron bar F, which is maintained in its place by the spring J. HH are the circular iron pieces let in the brass bottom. The locking arrangements may be varied.

The first or lever arrangement only requires half the battery power to open the lamp with the same strength of spring under the armature. It can be applied to other lamps.

An ordinary Leclanché battery is used, the maintenance of which only costs a few shillings per annum.

The electro-magnets are wound with covered copper wire, and are arranged under a brass ring let into the lamp table, in the cabin, so as to be away from risk of injury.

An objection to this lock was the slight hold the bolt had, thereby rendering it more easily displaced. It is very questionable whether this lock afforded any more protection than the screw lock as at first introduced.

In France, where Mueseler lamps with magnetic locks were used previous to 1877, the galvanic batteries to lock and unlock them had a power of 22 lbs. After that date gramme machines of double the power were used instead of the batteries, but even these did not altogether prevent the men from surreptitiously opening the locks.

The Royal Mines Commission, 1886, say that of magnetic locks, Wolff's was the simplest and most satisfactory examined by them. A magnetic lock, however, even if it cannot be opened without suitable appliances, is apt to get out of order by ordinary wear and tear, and is a cumbersome addition to a lamp.

Cuvelier's patent lock for miners' safety-lamps is very ingenious in construction. The following are the particulars of it as supplied by Messrs. W. P. Thompson & Co., Patent Agents, Manchester:—

The invention is designed to absolutely prevent the opening of the lamp by any unauthorised person, or in any but the proper place.

It is an undoubted fact that the lamps at present in use are opened by the miners, notwithstanding the seals and screws by which they are fastened.

The lock invented by Mr. Cuvelier is said to be most effectual in preventing the opening of lamps in the workings. It consists of a ring opened and closed by the force or pressure of a liquid acting upon it internally, which causes it to contract and expand on exactly the same principle as that adopted in the Bourdon pressure-gauge, so largely used.*

The bolt of the lamp consists of a metal rod, A, formed with a shoulder, B (Figs. 544–547). It is placed in a chamber, C, at the side of the oil reservoir, R, and pressed back by a wire spring, D. Below the reservoir is placed the lock, N, comprised of the tube, E, hermetically sealed at the ends by the solid end-pieces, F, and the inlet-pipe, H, with a small pin-hole, L. at the bottom. The ends, F, when closed, rest below the shoulder, B, and lock the bolt. To open the lamp, water is forced into the tube, E, through the hole, L, by means of a pump or accumulator (placed in the lamp-room), which expands the tube, drawing the ends, F, clear of the shoulder, B, which is then shot down past them by the spring, D. The elastic pressure of the tube draws the ends together again, under the shoulder, B, as soon as the bolt is pushed up, by means of a wire, introduced into an orifice in G (Fig. 546), as shown by the arrow, to again lock the lamp. No special appliance is needed for locking the lamp. A plate or false bottom, J, is soldered in below the tube, to keep it safe, and prevent it from being tampered with. This lock can be fitted to any ordinary lamp.

M	Lamp Casing.
N	Lock.
R	Oil Reservoir.
A	Bolt.
B	Shoulder on bolt, A.
C	Casing of bolt.
D	Spring.
E	Ring forming lock.
F	Solid end of lock-ring.

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* The principle of the Bourdon tubes is fully described in Chapter XVI., under the head of "Pressure and Vacuum Gauges."

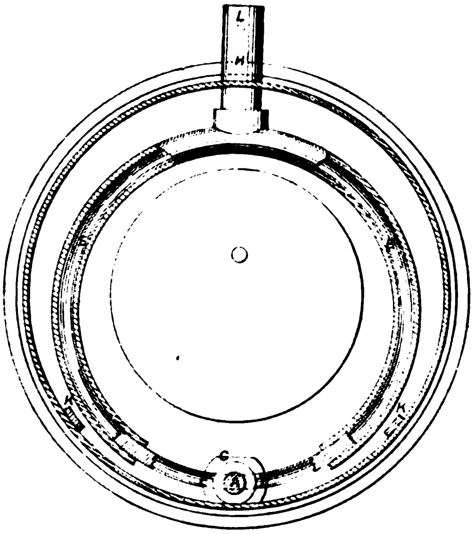

Fig. 544.

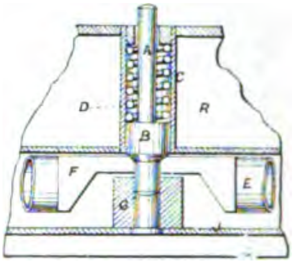

Fig. 545.

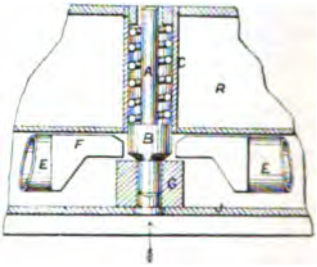

Fig. 546.

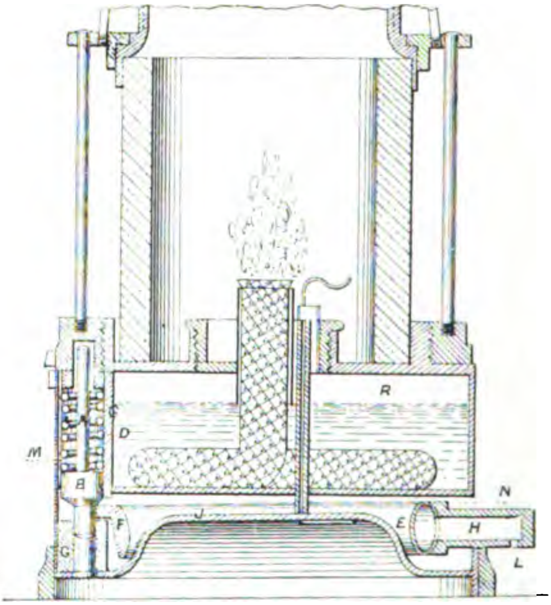

Fig. 547.

Cuvelier's Lock for Safety-Lamps.

H	Inlet-pipe to lock-ring.
L	Orifice in pipe, H.
J	Plate to protect tube.
K	Stop-pieces to prevent undue expansion of tube.

Fig. 544 is a plan (looking from below). Fig. 545 is a sectional view showing the position of the bolt, A, when the lamp is locked. Fig. 546 is a sectional view showing the position of the bolt when the lamp is unlocked; and Fig. 547 is a sectional view showing the lower part of a safety-lamp with the lock applied to it.

Possibly the want of simplicity of this lock and the dread that it may also be tampered with have prevented its coming largely into use.

Extinguishing Locks.—These locks have been devised by inventors who thought thus to remove from the collier the temptation to unlock his lamp. The "Protector," and that on Marshall's automatically closing lamps already described, are forms of extinguishing locks, and another, *Mercier's*, is shown in Fig. 548, applied to a Marsaut lamp, which has also Mercier and Hart's shut-off shield. The lock is arranged for the lamp to burn mineral oil in a vessel with sponge in the ordinary way, the light being regulated by a mill-headed screw. An aperture K is cast or otherwise secured to the lower collar G, through which the bolt M works. A chain L is attached to the end of this bolt, and secured at its other end to one of the bars, to prevent loss of the bolt. Two levers work on centres secured to the oil-vessel, one of which, E, has a cap D, and the other, F, is toothed at the end. The bolt M also terminates in a tooth pointing in the contrary direction to that of the lever F. When the bolt M is pushed in these teeth become locked together, and can only be disengaged by the partial withdrawal of the bolt M, which pulls the lever F with it, and at the same time actuates the lever E, causing the cap D to go over and extinguish the flame. A steel wire spring H, secured to the toothed lever F, tends to pull the latter back and prevent the withdrawal of the bolt in case the lamp should be turned upside down. A rivet or other lock may be passed through the hole N in the bolt M and projection K. To unlock the lamp, the bolt is partly withdrawn to bring the teeth clear of a stud on the lamp bottom. Whilst in this position the oil chamber is turned partly round to allow of the separation of the teeth, when the bolt may be completely withdrawn.

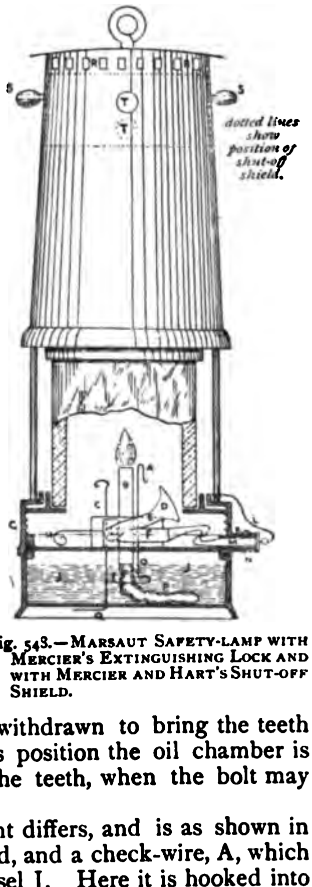

Fig. 548.—MARSAUT SAFETY-LAMP WITH MERCIER'S EXTINGUISHING LOCK AND WITH MERCIER AND HART'S SHUT-OFF SHIELD.

To burn colza, rape or seal oils the arrangement differs, and is as shown in the drawing. In this case there is a pricker, C, added, and a check-wire, A, which passes through a tube on the burner into the oil vessel J. Here it is hooked into the wick P so as to prevent the lamp from being unlocked when the wick is raised too high. The wire A requires to be adjusted about twice a week. When passed through the wick, the latter is pulled down as far as the check-wire permits, and the wick is cut off level with the top of the burner B. As the wick is raised the wire A is also raised, and when above the burner it prevents the cap D from passing over, which must therefore be lowered, and in being lowered the wick falls with it. This prevents the lamp bottom from being opened, for unless the cap passes over the burner the bolt cannot be withdrawn.

It is probable that the wire in this lock would give trouble in use, and it may be said that the security of the lock depends on the lead rivet N. Anyone careless enough to remove this in the workings would probably be provided with matches, so that the lamp's becoming extinguished in the act of unlocking seems of little moment. It could easily be re-lighted and re-locked. It appears, therefore, to be useless to add complicated fittings to lamps for the purpose of extinguishing them in unlocking, and such locks are not likely to become at all popular.

Lead Rivets.—On account of their simplicity and ease of treatment, as well as from the measure of security they afford, lead pins are coming more and more into use at collieries. The pins are moulded with a head at one end, and fit openings in the two parts of the lamp to be held together. When in place, they are firmly riveted and punched with some device at both ends. This is very important, as it forms the means of detection, if the lamp should be wrongfully opened. A very common lead rivet arrangement is shown in Figs. 518 and 532, and consists of a hasp which is lowered over a projecting boss, bored with a hole to receive the lead rivet horizontally, as has already been described when dealing with Fig. 532. The application of a lead rivet to the shield separately will be found described at the same place. When the rivet is placed horizontally the examination of the stamped ends is facilitated. But this even has a drawback, as after a rivet has been cut and the lamp surreptitiously opened, it is possible to replace the rivet so as to prevent detection. With the rivet in a vertical position the facility for such replacement is not so great. The letters or marks used may be varied from time to time. If a plain rivet were used it could be removed and replaced without its being noticeable. The rivet head can be cut off with a pocket-knife, and the pin can be re-riveted, or a fresh rivet may be substituted with the simplest of tools. The unauthorized reproduction of a stamping device is not so simple a matter, and it therefore affords considerable protection. Where, however, the colliery is extensive, and the number of lamps large, a critical examination on their return, which is necessary in order to detect the absence or alteration of the device, is difficult, because of the rapidity with which they are handed to the lampman. The lead rivet, though by no means infallible, is at present the best means of protection to be had.

In many instances a sliding pillar lock locked by a lead rivet is used, such as that shown in Marshall's lamp in Fig. 537, which locks both shield and oil vessel together with one rivet. Ryder's lock, drawn to a large scale and shown in Fig. 549, is another of this type. A is the upper horizontal ring of the lamp frame, on which the shield is screwed. D is the bottom ring to which the oil vessel is attached. B is the sliding pillar pierced with a hole near the top for the passage of the lead rivet C. This is shown in solid lines in the Fig. as being in the position it occupies when locked. To unlock the lamp the rivet must be removed, and then the pillar can be raised to clear the oil vessel, which is then unscrewed. The sliding pillar is lowered to where shown by the dotted lines, and then the shield may be unscrewed, as there is no portion of the sliding pillar in a position to prevent it. To lock the lamp the sliding pillar remains in its lowest position until the shield is screwed on, after which it is raised as shown in dotted lines, and locks the shield. This allows of the oil vessel being screwed on. The bar is then lowered to about mid-way between its extreme movements, and in this position the bottom now fits into a recess in the oil vessel. This lowering does not take the bar entirely out of the shield; the bar remains partly in it as well as in the oil vessel. In this position it is secured by the lead rivet, thus locking both oil vessel and shield.

The lead rivet is used to lock lamps in other ways, but when the oil vessel is screwed home, there must always be two parts pierced with holes brought opposite each other for the reception of the lead rivet.

In order to deal rapidly with a large number of lamps with lead rivet locks, the *Wolstenholme* lamp locking and unlocking machine has been devised, and is supplied by Messrs. John Davis & Son, Derby. Fig. 550 shows the machine, which consists of an iron casting bolted to a table. It has two openings in it, A and B, in which are the locking and unlocking devices. The same motion of the hand lever shown at the side works either the locking or unlocking arrangement. The lamp is held in one hand, while, with the other, the operator grips the lever, which on one side of the machine B causes the advance of a pin to drive out an existing lead plug. This is done without having first to cut the head of the rivet with a nippers. The lamp is held in such a position that when the lever is pressed down the pin in the recess descends on the central portion of the rivet and, being smaller than the head, forces the lead downwards, leaving a slight detached ring of lead severed from the head, and quite freeing the lock rivet from the lamp. There is no obstruction below, and the discarded rivet is thus pushed out. On the other side of the machine the lamps are locked by means of the lever. At the bottom of the recess A is placed an engraved device, which may be changed as often as required, and a punch above the die descends when the lever is set in motion. The punch has a device similar to the die. The lamp is held in such a position with one hand that the lead rivet in the lamp rests on the die in the recess of the machine, and when the lever is pressed, the descending punch wedges up or rivets the lead plug, and, at the same time, stamps the device upon its two heads.

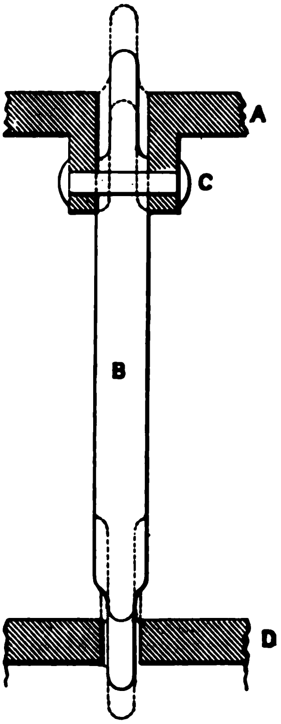

Fig. 549.—Ryder's Patent Lock.

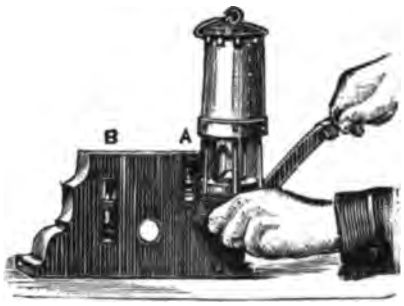

Fig. 550.—Wolstenholme's Machine.
A. Side for Locking or Riveting Lead Plugs.
B. Side for Unlocking.

To facilitate the locking and unlocking of lamps it is intended that this apparatus shall be fixed at various safe points in the workings, such as the lamp locking stations, so that the authorized colliery official may be able to relight the lamps in the pit without the inconvenience of carrying with him closing plyers and cutting nippers. A supply of fresh rivets must, however, be available at the stations.

The lead rivets may be bought ready for use, but it is more economical for the colliery to be supplied with a lead rivet moulding machine. This is an extremely simple apparatus, consisting only of two castings, the lower of which is a plain piece of iron with a surface made smooth by planing, and is usually bolted down to a bench. The upper casting has about 60 circular holes drilled through its section, with an enlargement in each hole at the bottom of the casting to form a head on the rivet. This casting has a smooth surface underneath, and rests on that bolted to the bench without any space between. Its upper surface has a raised edge at the sides and ends for the retention of lead, which overflows in filling the holes. Molten lead is poured from a ladle into the mould; it fills up the recesses and forms rivets, the number of which corresponds with the number of holes. On the lead becoming cool, a chisel of the same width as the upper casting is used to cut the surplus lead from above and around the holes, by guiding it from end to end between the edges of the casting. If the upper casting be now raised, the lead rivets drop out with the heads downwards, or they may be retained in the mould, by raising the upper casting before using the chisel. The cutting is then performed while the mould is placed over some receptacle into which the rivets drop. The lead not formed into rivets is melted and used again.

There is very little waste with lead rivets, as the portions pressed out in unlocking are melted and recast in the moulding machine.

Lighting and Re-lighting Locked Lamps.—A great many lamps become extinguished at the working faces from a variety of causes, and this has led to designs of lamps which may be quickly re-lighted without their being unlocked. The designs may be divided into two kinds, in one of which lamps may be re-lighted anywhere by the user; the other, only by an authorized person at properly appointed stations, by the application of electricity.

Wolff's lamp is one of the first kind, the re-lighting arrangement being adapted for the volatile illuminant, benzoline, used in it. The lamp contains a reel of paper on which fulminating spots are placed at intervals of about a quarter of an inch, each of which can, in turn, be brought opposite the wick, and, at the same time, be struck by a small spring hammer, operated by a button at the bottom of the lamp: the composition explodes and ignites the benzoline vapour surrounding the wick. This would not be the case if the lamp were not supplied with spirit. When all the caps have been struck, the old paper strip may be taken off and re-placed by a new one. The gauze on this lamp has 1,296 openings to the square inch, and it has a moderately good magnetic lock. The lock has a ratchet which is pressed against teeth on the oil-vessel by means of a spring. This prevents the oil-vessel from being unscrewed until a powerful magnet overcomes the spring. The ratchet then being withdrawn, the lamp may be opened.

Elsom's method of re-lighting is another example of the first kind, but is applicable to lamps using vegetable oil. It consists of an arrangement by means of which matches within the lamp may be struck to re-light the lamp in case the flame becomes extinguished. It is not likely that this invention will ever come into use, for, as at present framed, a rule at collieries using safety-lamps very properly prevents matches from being taken into the mine. It is also very undesirable that power to light their own lamps should be in the hands of the miners themselves, either by matches or explosive compounds. If the lamp were extinguished by a fall, it may be injured and, in this state, be re-lighted by the collier, instead of having been subjected to examination on becoming extinguished. Again, if the lamp were extinguished in any other way, and subsequently filled with an explosive mixture, the sudden flash of re-lighting and the internal explosion might cause sufficient pressure within the lamp to force the flame through the gauze and fire the external gaseous mixture.

An example of the second kind of design for lighting locked safety-lamps is that of Mr. E. M. Hann.* It consists of a pair of rods or pole CC, Fig. 551, extending vertically from the bottom of the lamp through the oil vessel to a level with the wick, and insulated from the lamp and the oil. A fine platinum wire, D, is stretched across and secured to the upper part of the rods CC, one of which can be partly rotated, thus carrying the platinum wire to touch the wick or to be withdrawn from it. This pole is prevented from being turned too far by means of a cam-shaped collar which comes into contact with the outer wick-tube. The poles CC terminate below in two flat metallic heads BB, which fit on two corresponding metallic flat heads AA fixed on a table top. The heads AA are attachable to wires leading from a small battery. To light the lamp, after it has been trimmed and locked, or before locking, it is placed on the table with the heads BB resting on the metallic portions AA, thus completing the circuit through the platinum wire D, and the flowing electric current heats the platinum wire sufficiently to light the wick. On removing the lamp the platinum wire is moved clear of the influence of the lamp flame, which can be adjusted in the ordinary manner.

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* See Proceedings of the South Wales Institute of Engineers, vol. xv., pp. 74, 75.

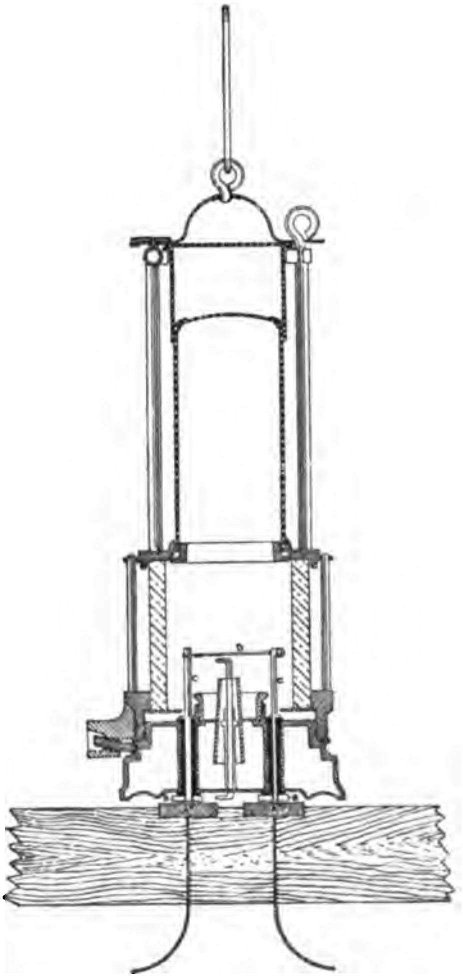


Fig. 551.—Illustrating Hann's Method of Lighting Safety-lamps while Locked.

The electricity used is of low tension, capable of producing only feeble sparks, and the battery used is small, portable and inexpensive. The appliance is shown in the Fig. on a Clanny, but is applicable to almost any kind of lamp, and as the platinum is raised to a white heat any kind of oil may be used. The apparatus is intended for lighting lamps at lamp stations in the workings by an official in charge, and not inside at points nearer the working places on account of the danger of sparking when the lamp is connected to the battery. The lamp stations, therefore, should not be made any nearer to the working faces than is quite safe for the use of naked lights. Whenever a lamp becomes extinguished in the workings it should be carefully examined before being re-lighted by the official in charge at the lamp station. If this can be done without unlocking, as may be the case with some unbonneted lamps, there may be great convenience in an apparatus of this sort, but with lamps in which the shield and oil vessel are locked at the same time by a lead rivet, a proper examination can only be made by taking the lamp to pieces, and little, if anything, is gained by the use of such a lighting arrangement, especially in small workings: indeed, its use may be fraught with danger, as there would be a temptation to light and afterwards use a lamp without the trouble of examination.

Best's is another apparatus of this kind, in which the electricity is supplied by a dynamo capable of lighting four lamps at the same time at the four windows of a lamp-cabin on the surface as the workmen come forward to receive them, or if preferred, one lamp at a time may be lighted.

There appears to be great economy of time and of oil by calling into use this electrical means of lighting lamps. About 500 lamps can be lighted in half-an-hour. One lamp can be put out and re-lighted six times in ten seconds, none of the lamps, therefore, need be lighted an hour or two before being wanted. Sparks are not so likely to be given off, as there is no electric force until the lamp has been attached and the current switched on; and even if they did, the sparks are not so objectionable in a surface lamp-cabin.

Shields.—The first shield was used on the Davy lamp and consisted of a very thin piece of tin or other metal bent half round outside the gauze and secured at its edges to two of the bars, up and down which it might be moved at pleasure. It reached some 2 or 3 inches up from the bottom of the gauze cylinder, where it rested on the ring screwed to the oil-chamber. If the lamp was carried in a direct strong current this afforded the light some protection from the wind. This primitive production was the precursor of the glass cylinder and tin-can shields already described. The shields of glass lamps are usually of metal, and surround the gauze above the glass. They may either be fixtures on the lamp, or—what is far better—screwed on and afterwards locked. They may be single or double and pierced by inlet-holes for the air in a variety of ways already described. They are also mostly provided with a row of outlet holes near the top for the products of combustion. The inlet or the outlet holes or both may have shut-off appliances to close them so as to exclude the surrounding atmosphere or to enclose the carbonic acid gas. To enable the user to see the inside of a lamp openings are sometimes made in the shield, which are covered by movable shutters sliding along grooves formed of two ribs of tin for each groove. If the lamp should become very hot there is risk of the ribs being melted off, and as there is no lock to the shutter there is a temptation to the collier to leave it open for the better ventilation of his lamp. A better arrangement of the shield is that of the Sight Lamp Company. Their shield is perforated by a large number of holes and lined inside by a glass made for the purpose. The glass is well protected from injury by the metal, and there is no space between the two. The inside of the lamp may thus be seen, as it is a single shield and the light is not obstructed to the same extent it is with a solid metal shield. The state of the gauze may always be seen while the shield is locked. If the lamp should be in an explosive mixture, this is of great advantage.

Shut-offs.—These are appliances for preventing either the surrounding atmosphere from entering a lamp, or the products of combustion from escaping, or both. They may be applied to the inlet or outlet holes of the shield, or to different positions in the inlet tubes, as in the Hepplewhite-Gray lamp, Fig. 523, and are then operated by hand at pleasure. In some instances they act automatically to close the outlet, as in Marshall's automatically closing lamp, Fig. 534. Their object is to ensure the extinguishing of all flame in a lamp tried in an explosive mixture. This is most speedily attained where both an inlet and an outlet shut-off are used. If only one shut-off is used that which closes the outlet is preferable, because the flame is more quickly extinguished by shutting in the carbon dioxide than by excluding oxygen as is done by closing the inlet. Beside which, when the inlet is closed there is danger lest the flame should be partly fed through the outlet.

The shut-off appliance invented by Mr. Arthur H. Stokes, H. M. Inspector of Mines, consists of two short brass cylinders closely fitting one inside the other with concentric holes drilled through them; the inner cylinder is fast to the bonnet, and forms the top of the lamp; the outer one is held and governed by a slot and pin arrangement, and may be rotated $\frac{3}{16}$ of an inch, when the holes in the inner cylinder are completely closed by the walls of the outer cylinder. This extinguishes the light, or the ignited gas within the lamp, in less than 30 seconds, by the precipitation of carbonic acid gas, after which the outer cylinder may be turned back again.

Fig. 548 shows Mercier and Hart's shut-off combination. Here the dotted lines near the top of the lamp show the short movable cylinder which is within the bonnet and fits it closely. There is an opening, T, to form a window in the shield. When the short inner cylinder is down, this window is closed and the outlet-holes R are uncovered; when raised, the window is unobstructed and the outlet-holes are closed, but if the inner shield be now partly revolved the outlet-holes R and window are both closed, and the products of combustion are retained within the lamp, causing the extinction of all flame. The buttons S, by which the shut-off is moved, being made of wood, always remain cool to the touch. If desired, the inlet-holes may also be closed.

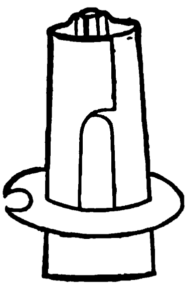


Fig. 552.—Wick Tube.

Wick.—Lamp-wicks were at first invariably round, in tubes from $\frac{1}{4}$ to $\frac{1}{3}$ of an inch in diameter, but now flat wicks are frequently used, about $\frac{3}{8}$ of an inch in width. With a round wick in a glass lamp there is an equal distribution of the heat of combustion all round it, and a consequent uniformity in the temperature of all parts of the surrounding glass cylinder when the flame is in its normal position. With any other shape of wick some edges of the flame must be nearer the glass cylinders than others, and an unequal expansion of the glass takes place. The unequal expansion in the glass, however, is only slight, and such as the glasses can well bear. A flat wick flame has a better illuminating power than a round one, and this is even more marked in Davis's corrugated wick-tube, shown in Fig. 552, made at the suggestion of Mr. A. H. Stokes. It is guttered along one side, and when a wick rather wider than the tube is introduced it is compelled to take a corrugated form. A longer surface of flame is thus obtained and a freer supply of oil to the wick. Wick should be well dried before use, as moisture in the fibre interferes with the free passage of the oil. The length of wick should only be a little more than enough to reach the bottom of the oil-vessel, and a fresh piece should be inserted each time the lamp is prepared for use, as its feeding capacity is reduced if allowed to remain soaking in the oil many hours.

Illuminants.—Vegetable or animal lamp-oils were at first used in safety-lamps, but now mineral oil is occasionally used. The objection to mineral oils is that they are volatile *per se* without decomposition. Their safe use then cannot be secured without certain precautions. Some of them are volatile at very low temperatures, and as their vapour forms an explosive mixture with common air, they are quite unfit for use in safety-lamps. The great heat generated in the lamp would be sufficient to bring up the mineral oil to the flashing-point, and make it highly dangerous. Even with heavy mineral oils of high flashing point, such as would warrant their use in safety-lamps, great care is necessary in their storage and in the filling of lamps. Mineral oils are more perfectly fluid, more combustible and give a better light than vegetable or animal oils, but unless there is a good supply of air to feed the flame they are apt, owing to the large amount of carbon they contain, to deposit much soot. The petroleum spirit, called colzalene, used in the Protector lamp has already been commented on, as also has the mineral oil of very high flashing point used in the Thorneburry lamp. In the Wolff and in some lamps made by the Protector Company, benzolene is used, but it is very volatile, requiring the greatest possible care. A sponge is saturated in the oil-vessel, and a bright clear light is given by this illuminant without the emission of smoke, but the precautions which must be taken in the filling-room have prevented it from coming into general use. These precautions are much the same as with colzalene. One mineral oil company has made lamps to burn solid paraffin, but as this requires to be melted round the wick and within the wick before it will burn, the object sought is frustrated.

The two staple illuminants for safety-lamps are best refined rape, that is colza oil, and seal oil. These give off no vapour under a comparatively high temperature. The highest value of these oils is only obtained from careful preparation, and cheap productions are to be studiously avoided. In their Report the Royal Mines Commissioners, 1886, draw attention to their experiments on the burning qualities of these oils and to the fact that a superior light is secured by a mixture of either colza or seal oil with petroleum or paraffin oil of a flashing point not lower than 80° F., in the proportion of not more than one-third part, by measure, to two-thirds of the vegetable or animal oil. The cost of good refined rape oil in London was from 2*s.* 2*d.* to 2*s.* 6*d.* per gallon, good seal oil about 2*s.* per gallon, and petroleum of a suitable quality from 7*d.* to 9*d.* per gallon. The mixture then, while giving a better illuminating power, is cheaper than the best colza or seal oil, and its use is attended with the further advantage that the wick does not so readily become charred and clogged at the top by a hard carbon crust.

Glasses.—For the most part glasses are cylindrical in form. That of the Hepplewhite-Gray is conical, thus letting the rays of light strike upwards better. The glass cylinder on the Clifford lamp, shown in Fig. 528, has a shoulder formed on it, and is reduced in size at the top. A cup-shaped glass is used on one of Col. Shakspear's lamps. A section of the inner glass of one of Evan Thomas' lamps is curved so as to make it broader at the bottom than at the top. The glass on the Eloin lamp is peculiar, having a concave exterior surface, a straight inside surface, and is broader at the top than the bottom. No form is stronger than the plain cylinder, and this is the most popular, as it admits of being fitted more accurately in place.

To ensure a good fit the edges are polished or ground and must be approximately parallel. All chipped or irregularly edged glasses are discarded. The perfect ones selected are to some extent protected from the effects of changes of temperature by the use of asbestos washers placed between the glass and the metal, as already stated when describing the Clanny lamp. The usual size is $2\frac{5}{16}$ inches in diameter outside by $2\frac{3}{8}$ inches long, the sides being perfectly parallel and the thickness of glass regular. Glasses with polished ends are less liable to crack and fray the asbestos washers than those with ground ends. For ordinary use entirely clear glasses are preferable, so that the light may be shed equally in all directions. For some purposes the reflector glasses made by Messrs. John Davis & Son, Derby, and shown in Fig. 553, are better, as they are enamelled white one-third of their inside surfaces to reflect the light more strongly through the clear part and render objects in front more easily seen. Conical reflector glasses are also made for the Hepplewhite-Gray lamps. For surveying blue, green, or red glasses are made, but these, of course, greatly reduce the light. Any glass lamp now intended to be used for the detection of fire-damp usually has a portion of the inside surface of the glass painted black so as to form a background which renders " caps " more distinctly visible.

In Mr. Albert Smith's lamp a tube of mica is used instead of a cylinder of glass. It is constructed with longitudinal metal joints, and is finished at each end by a short brass tube. Strips of mica are also used in Pendleton's lamp. Although the mica does not crack when the flame is allowed to reach it, if great heat is generated in the lamp it becomes brittle and, owing to the joints, untrustworthy. It is not likely, therefore, that mica will supersede glass for the cylinders surrounding the flame.

Cleaning Lamps.—At small collieries having only a few safety-lamps in use the cleaning is usually done by hand, without the aid of mechanical contrivances to unscrew or remove the internal fittings. An ordinary hand lamp brush is used to rub the dirt from the gauzes. If the gauzes are smeared with oil the brushes are sometimes dipped into powdered magnesian limestone, or the gauzes are steeped now and again in a solution of caustic potash. An examination of the wires is made during the cleaning by holding the gauze cylinders to the light at intervals. Dirty patches thus revealed to view are rubbed with the brush, while the discovery of twisted wires or dents in the gauzes leads to their removal.

For the glasses and any metallic portions not cleaned by brushes and suitable compositions cotton waste is used or suitable cloths. On fitting the parts together again, the washers, where necessary, are renewed so that there is no slackness of glasses. Safety-lamps are sometimes taken home to be cleaned at the close of each shift by those using them, after first being unlocked in the lamp-room on the surface. In some instances only the tops are taken home by the men, the bottoms being left for the lamp-keeper to deal with. Where the cleaning is performed in a properly appointed cabin it may be done by a qualified staff of lamp-keepers, or by each user cleaning his own lamp; but this last plan is open to grave objections, as the number of brushes and the accommodation in the lamp-room are limited, and the men, already tired with their day's work, and irritated at having to stand about and await their turn to use the brush, hurry over the work as quickly as possible instead of cleaning their lamps thoroughly.

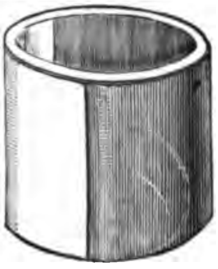
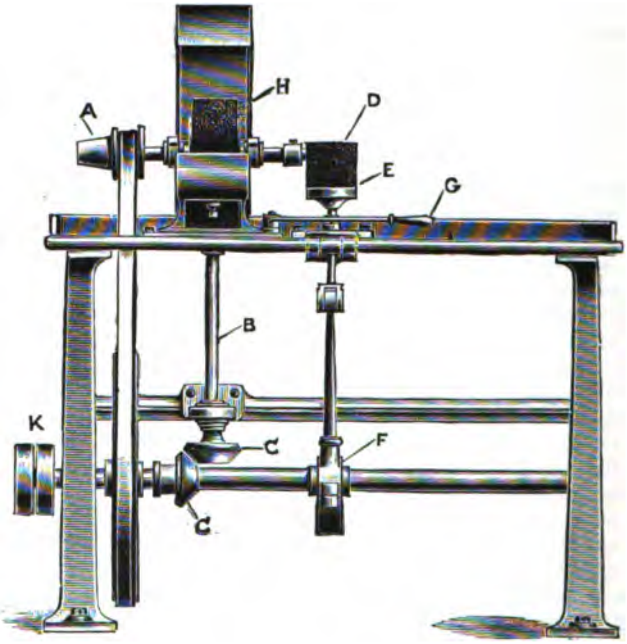


Fig. 553.—Patent Reflector Glass.

Even when the workmen clean their own lamps there is at least one lamp-keeper at each colliery to replenish the oil-vessels, renew the wicks and replace worn washers or broken glasses, except in the very rare instances in which the men themselves supply all safety-lamps and afterwards maintain them in a perfect state of repair. In order to comply with the Mines Act, 1887, there must always be an examiner appointed to see that the lamps are in safe working order, which they cannot be if any parts are omitted, and that they are securely locked before being taken into the workings. The user should also examine the cleaned lamp he receives from another person to make sure for himself that all its parts are in proper order and that the lamp is well trimmed.

Some collieries have as many as 5,000 safety-lamps in daily use, comprising lamps of different types for gas-testing, shot-firing, or general use. These large numbers must seriously tax the energies of even an efficient staff of keepers. Their labours, however, are lightened by the use of special machines for cleaning safety-lamps, by means of which lamps are rapidly unscrewed, cleaned, and put together again. Fig. 554 shows Wolstenholme's safety-lamp cleaning machine, supplied by Messrs. John Davis & Son, Derby. It is provided with a table 4 feet 6 inches by 2 feet 6 inches and is self-contained. The motive power may be obtained through the belting from the fast and loose pulley, shown at K, or a self-contained engine of the "Brotherhood" type can be fixed to revolve the shaft; or if the colliery is provided with electric current, a little self-contained motor can be attached to the shaft. A is a cone which in revolving unscrews the inside ring of the lamp. There is a cone for screwing up the inside ring, and as soon as pressure is removed this cone ceases to revolve, motion being transmitted by friction cones at C C. An oscillating brush at E cleans the gauze while it revolves on the spindle, without unduly wearing the gauze or brush, and the lever G imparts a sliding movement to this oscillating brush so as to allow for wear. The oscillating movement of the brush E is obtained by means of an eccentric, F, on the lower shaft. At H is a circular brush which runs in a suitable composition for polishing the body of the lamp. The oil-vessel is not screwed to or detached from the top part of the lamp by the machine.



SCALE : ⅝ in. = 1 foot.

Fig. 554.—WOLSTENHOLME SAFETY-LAMP CLEANING MACHINE.

Howat's safety-lamp machine is worked by hand, and is used for quickly screwing or unscrewing the parts, and for locking or unlocking with lead rivets. The lamp is placed in a cup containing a number of slots, with the projecting boss in one of the slots. A handle is turned which by means of bevelled toothed wheels causes the cup to rotate. The lamp turns with it, but to unscrew the bottom the top portion of the lamp is held with one hand while the handle is turned by the other. To unscrew the inside ring the cup is taken off, this exposes a square nut which fits into projections on the ring securing the lamp glass. The lamp is placed on this nut and the handle again turned until the inside ring is unscrewed. To screw the parts on again a reversal of the above operations takes place. The same machine rivets the lead pins or removes them from the lamps, but it does not clean the lamps.

Lamp Room.—This is much better on the surface than underground, where the light is not so good. It should be situated near the pit, so that the workmen on their arrival in the morning may not have far to go out of their way, and should not be blocked by other buildings or obstructions to the daylight. The lamps, moreover, will often have to be cleaned by artificial light, which must be good. The staff of workmen must be both sufficient and efficient. They should be careful and experienced men, who will ensure the lamps being properly tended and securely locked before being sent out. The walls of the cabin are fitted with narrow shelves properly numbered on the outside edge to receive correspondingly numbered lamps as they are brought back. An entry of each owner's name is kept in a book opposite his number. A bench in the cabin is necessary for cleaning on, whether this is done by hand or by machinery. The number of windows will depend upon its size. There may be two, three, or more in order to pass the lamps out to the men quickly. The passage outside leading to and from each window should be formed to compel the men to file past one by one and be barred by a self-registering turnstile securely locked when not in authorized use. The number of men as ascertained thus each day should correspond with the number of lamps taken out. At times, unfortunately, it becomes of great importance to know the exact number of men descending the shaft in any particular shift, as well as their names and places of residence. When the workmen take both top and bottom portions of their lamps home in order to clean them, it becomes more difficult to know who are at work in any particular shift.

Examining and Testing Lamps.—Modern lamps are more complicated in their construction than the first type, and the number in use is vastly increased. The work of examining these lamps at large collieries is, consequently, considerable. In many cases the lamp is half covered with a bonnet, hiding from the attendant's view the most important parts. The large number which must be taken to pieces before they can be examined, and afterwards restored to working condition, entails much labour, and it is possible that some small but important part may be omitted in the operation. It is very difficult for the eye to detect defects in the adjustment, and impossible—without resorting to other means—to know whether the joints between the glass cylinder and the gauze are perfect under the varying temperatures at which a lamp is worked. If a thoroughly reliable and simple means of testing could be devised, it would be well to test each lamp before it is taken into the pit. At some collieries this is done daily in the lamp-room by placing the lamp (lighted), after examination, in a mixture of gas and air previously ascertained to be a properly explosive one. This test should be slow and deliberate, and the lamp allowed to remain some time in the mixture.

The Royal Commissioners draw particular attention to this point. They say: "With the safer and more complicated lamps it is still more difficult to detect by eye defects in adjustment. Indeed, in many instances important parts cannot be seen after the lamp has been put together. During the course of our experiments explosions have been occasionally traced to imperfections so small that we have had some trouble in finding them even when certain of their existence. Experiments 723–725 of the Woolwich series are examples of this, as will be seen by comparison with those following them. We consider it, therefore, absolutely necessary that such lamps should be regularly tested in an explosive gas mixture before they are allowed to descend the shaft." In another part of their report:—"A lamp may be of the safest pattern, and yet small defects in the fitting of its parts may entirely deprive it of its power of affording protection. In preparing a large number of lamps for use in a mine it may happen even with the greatest care on the part of the lamp-man that a lamp in an imperfect condition may be allowed to pass. The detection of these imperfections by simple inspection is in many cases almost impossible. And we are convinced that the only way of avoiding the introduction into a mine of a dangerously imperfect lamp is to test every lamp in an explosive mixture of air and some inflammable gas before it is allowed to descend the shaft."

Mr. Patterson, of Newcastle-on-Tyne, has patented an apparatus for the supply of explosive mixture to a safety-lamp. The mixture (which should be of uniform nature) is conveyed through an upright pipe provided with a tap for regulating the supply, and discharged through the casing at the centre of a small fan driven by means of a belt from the fly-wheel of an engine. The fan outlet is connected with the test-box by a short horizontal pipe. The box rests on a table of suitable size, and has a hinged cover through which the lighted lamps are passed, and allowed to rest on the bottom of the box. A window is fixed in one side of it, through which the lamp-man watches the lamp he is testing at the same time that he keeps one hand resting on the handle of the supply-tap, so that he can watch the lamp, and regulate the amount of explosive mixture at the same time. In this way the mixture is forced round the lamp in every part, and defects detected.

Photometric Tests.—The relative illuminating powers of different sources of light are ascertained by instruments called photometers or light measurers. These are of several types. On account of the simplicity of the apparatus, Count Rumford's is frequently employed. In it the relative intensity of two lights, such as a candle and a gas-jet, or a candle and a lamp, is compared. Fig. 555 shows Rumford's shadow photometer. An opaque rod, of any material, such as blackened wood, is supported in a vertical position. Behind it is a calico or white paper screen, while the lights to be compared are placed in front, and at a little distance from each other. The two lights are shifted about until the two shadows cast from the rod upon the screen appear to be of equal density. The more powerful light will then be further from the rod than the other, and when two sources of light produce equal intensities of shadow at unequal distances, their illuminating powers vary with the square of their distances from the shadows on the screen. Theoretically this method is perfect when applied to lights of the same colour. No generally useful idea of the intensity of the illumination of different candles or other sources of light is conveyed without reference to some standard of measurement. In 1860, Parliament fixed this standard as the light of a certain candle. This candle is such that six of them weigh one pound, each burning 120 grains of sperm per hour. This is the legal standard of light in England and in America, and is used at the Board of Trade for measuring the illuminating power of gas, electricity, oil, or other illuminant. It should have burned some ten minutes before use, and the wick should be bent over and have a red tip. The candle is not a scientific standard, because it consumes irregularly if exposed to draughts, or burned in a confined space, or placed in varying temperature.

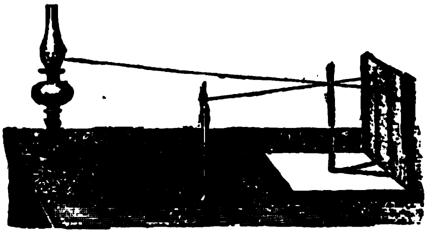


Fig. 555.—RUMFORD'S SHADOW PHOTOMETER.

Viole's standard of illuminating power was authorized by the International Congress of 1881. The very expensive apparatus required has prevented this standard from being widely accepted, and carcel or candle standards are mostly adhered to. The carcel lamp is the standard of light used in France. The wick is cylindrical, giving an Argand or central draught flame. It is woven with 75 strands, and weighs 3·6 grammes (55·5 grains) per decimetre (3·9 inches) of length. The chimney is 29 centimetres (11·3 inches) high, 47 millimetres (1·88 inch) in diameter at the bottom, contracting just above the wick to 34 millimetres (1·36 inch). The carcel is the light yielded by this standard lamp, burning 42 grammes (648 grains) of colza oil per hour, with a flame 40 millimetres (1·57 inch) in height. One carcel is equal to about 9·5 candles. A carcel gas-jet is a standard Argand gas burner, made with proper rating to give the light of a definite number of carcels illuminating power. Cognizance must be taken of the quality of the gas as well as of the burner used. An Argand burner will not, during a long series of experiments, differ 1 per cent. in light value. It appears, therefore, that this is a more unvarying standard than the sperm candle used in England. The results of the illuminating powers of tested lights are obtained by comparing them with a standard candle, and if, in the experiments, the latter burns irregularly, a correction is made to the normal consumption of 2 grains per minute. An average of ten or more observations is taken.

The following list of measurements of illuminating power of some safety-lamps is taken from the Report of the Mines Commission. The oil burned was seal or colza.

Lamp.	Illuminating Power, Standard Candle = 1.	Lamp.	Illuminating Power, Standard Candle = 1.
Davy	·07 to ·22	Gray	·22 to ·54
Stephenson . . .	·07 to ·17	Marsaut (3 gauzes) .	·33 to ·54
Clanny (unbonneted) .	·11 to ·58	„ (2 gauzes) . .	·62 to ·72
„ (bonneted) .	·13 to ·64	Mueseler, Belgian . .	·13 to ·49
Morgan	·50 to ·51	Evan Thomas, No. 7 .	·37 to ·46

In testing a lamp against a standard candle or against another lamp, the two lights are placed on the same level, and only those rays which reach the screen decide the result. The different values of lamps in lighting upwards are not taken into account, and reflectors, if placed on lamps, would affect the result. Even without reflectors different tests seem to give varying values to the illuminating power, dependent on the quality of the oil used, the height of the flame, and the trimming and make of the lamp. With clear glasses in safety-lamps the light emitted is only on a certain plane, and there are always shades thrown by the top, bottom, and bars of the lamp. The illumination from a candle or open oil-lamp in gentle currents of pure air is much more satisfactory, the light being free to shed its rays upwards and all round.

Use of Lamps.—Safety-lamps provided with the best form of lock to prevent their being tampered with by ignorant or reckless workmen at the face, and tested thoroughly before given out, will, if carefully used, afford considerable protection inthe mine. At convenient places near the working faces there should be recognised depôts for keeping reserves of locked lighted lamps ready for use. On a workman's lamp becoming extinguished another in working condition is thus available within easy distance of him. It is the practice at many collieries to keep boys for the purpose of travelling from the lamp-stations in the mine to the innermost roadways, carrying with them lighted locked safety-lamps, which they give to the workmen whose lamps have become extinguished during their work, and receive the extinguished lamps in return. When all the lighted lamps are distributed the boys return with those they have collected from the workmen to the lamp depôts, where they exchange them for more reserve lamps to be carried " in bye " again. These arrangements tend to the comfort of the workmen at the face, and give some little security against their tampering with the lamps. For these reasons they are to be commended.

The collier at the face is paid in accordance with the weight of the coal he sends out, and if his lamp becomes extinguished, from no fault of his it may be, the time occupied by him in going to a lamp-station to get it re-lighted, or of remaining in the dark until he gets another lamp, is a partial deprivation of his means of living. With far-off stations and frequent loss of light during the day, this deprivation will be very serious without some arrangement to relieve him.

The use of the safety-lamp is to give protection to the miners from explosive mixtures, whether by allowing of safe examinations by the firemen before the shift begins, or to prevent explosion from sudden eruptions or ordinary discharges of gas during the shift. This protection can only be maintained by constant vigilance and discipline in the mine, and by great care of the lamps. It is becoming more and more usual for the lamps to be looked after, cleaned, and trimmed by a properly qualified staff of lamp-keepers, in a cabin on the surface. The feeling is growing that the proprietors should find all lamps, and afterwards have them maintained daily in oil and wick, as well as examined and cared for when not in use. The workman's responsibility concerning his lamp is then confined to the time he is actually using it.

A very dangerous part of mining work consists in the repairing and timbering of main roads. Here it is that the illumination of the roof becomes so important a matter. A lamp which gives a good light to the collier at the face may be altogether inadequate here. As all lamps are prevented from throwing rays of light upward to some extent, even if a suitable reflector be used, they should bear a slight amount of tilting so that holes in the roof may be examined. The rays of the Hepplewhite-Gray lamp strike upward to a considerable extent, on account of its truncated conical glass, and it is, therefore, well suited for repairing work.

The colliers are often prejudiced in favour of the use of some particular lamp with which they have long been familiar, and strikes have been known to arise from attempts to introduce a new one. The same remark applies to the introduction of safety-lamps, instead of open lights. The workmen have a right to express their opinion on any particular safety-lamp, and to refuse to use a new one, but they are not so able to judge of the merits of a new lamp as the manager should be, and he has to bear the whole responsibility in this connexion under the Mines Act, and the Employers' Liability Act. There have been occasions, too, when both the owners and workmen have wished to continue the use of candles in collieries, deemed unsafe to be worked by open lights by Government inspectors, which have led to arbitration for decision.

Portable *Electric Safety-Lamps* have now become an established fact, and are in use at some collieries. They are what are called incandescent lamps. Incandescent lamps are simply small glass bulbs, containing a carbon filament, the ends of which are attached to two thin platinum wires, which pass through the glass, and conduct the electric current to and from the carbon filament. Wherever electrical energy is produced and passed through a conductor, it is deprived of energy, which is not destroyed or lost, but appears in the form of heat in the conductor. The amount of heat produced in the conductor is equal to the electrical energy not otherwise accounted for, or missing as such, at the end of the conductor, and that amount varies in accordance with the resistance offered by any conductor to the passage of an electric current. The current for an incandescent lamp is conveyed in by one wire, then through the carbon filament, and out by the other wire. The circuit is continuous, and the electric current in its passage meets resistance in the carbon filament, in overcoming which the electric current displays itself by generating its equivalent in heat, thus rendering the carbon incandescent. This white heat is produced in a vacuum, as every particle of air is pumped out of the small glass bulb containing the carbon filament before being hermetically sealed. There is no flame and no combustion in the incandescent light.

Platinum wires are used in preference to those made of other metals, because platinum comes closest to glass in its co-efficient for expansion by heat. If wires of copper, iron or silver were used, the probability is that the greater expansion by heat of these metals than glass, would cause the latter to burst or break, or, at any rate, prevent the possibility of retaining a good vacuum.

Another form of the electric light is known as the arc light. It is not suitable for underground use, as it requires a large electric current and a high electromotive force, but it is sometimes used at the pit bank. In the electric arc lamp the conducting wires are attached to two rather thick sticks of carbon, the ends of which are kept a short distance apart, so that there is a break in the continuity of the electric current. As the current leaps across the space between the two carbon sticks, a brilliant flame is produced, and the carbon sticks burn away.

The different forms of portable electric safety-lamps arise from the kind of battery used to supply the necessary current, the carbon filament in the glass globe being common to the different lamps. In all, the battery must be attached in a portable form, so that as the lamp is carried in the hand from place to place the battery may supply the electricity for a specified length of time. Portable batteries are either primary or secondary. In the former, electric energy is produced when zinc in the presence of another element is acted upon by a chemical compound, and is replenished by putting fresh plates and fresh chemicals into it. In the latter the energy is derived from some extraneous electric generator, and is stored in the battery at intervals.

The chief drawback to primary batteries is, that they are expensive to maintain, on account of the materials they consume. The objection to the secondary batteries is, the necessity of having another apparatus—a dynamo—to re-charge them.

Theoretically, whatever may be the form of battery, the lamp may be made to give any amount of light for 10, 12, or more hours, as may be desired, but, practically, the amount of light is limited by the weight it is convenient to carry.

The light is quite independent of the surrounding atmosphere, and it will burn equally well in any form of gas, which will not be affected by the light unless the glass globe be broken. On the glass breaking the lamp is almost instantaneously extinguished. It is plain, therefore, that no explosion can result from the use of the electric lamp unless the glass which surrounds the light be first broken, and the lamp at that instant happens to be in an inflammable atmosphere.

For surveying with the magnetic needle this lamp is not applicable, but for all other purposes it is unquestionably a great boon. As it is impossible to test for gas with the electric safety-lamp, a suitable fire-damp detector must be placed with it, in the hands of a fireman, or be used by any one wishing to ascertain if fire-damp be present in the mine. Liveing's, Swan's, or Lewis & Maurice's, or any other reliable fire-damp indicators, may be used.

Electric portable safety-lamps are in their infancy, and, although in practical use, they do not at present compare favourably in point of first cost or in maintenance with the best oil lamps, the latter costing only about one-half of the former to maintain throughout a shift. Possibly colliery proprietors who supply lamps for the use of their workmen are awaiting further inventions, and the survival of the fittest, before adopting it to any extent.

FIRE-DAMP DETECTORS.

In the early days of coal-mining fire-damp was detected when present in sufficient quantity by means of an open light. By this method a very slight percentage was either not detected at all or was disregarded. Later on the size and luminosity of the sparks given off by the steel-mill were supposed to indicate the presence or absence of fire-damp, but no trustworthy results could be so obtained even by experienced observers. All such methods of testing for fire-damp have happily passed away since the inventions of the safety-lamp and of special fire-damp detectors of great delicacy.

Ordinary safety-lamps themselves serve excellently as fire-damp detectors, if the amount of gas present in the air is about 3 per cent. or more. By drawing down the wick of the Davy lamp to reduce the light, an experienced observer can estimate with close approximation the presence of from 3 to 6 per cent. of fire-damp in the air by the height of the caps above the flame. The small pale caps are not so easily judged in clear glass lamps owing to the highly reflective surfaces behind the flame. The inner surface of the back of the glass may, however, be burnt to a dead black, which greatly aids in detecting caps on the flame. Under favourable conditions 2 per cent. of gas in the air may thus be detected. A cap is most easily examined when a black screen is placed behind the lamp-flame, whilst at the same time the flame is prevented from dazzling the eye by being shielded with the hand or some other screen.

Professor F. Clowes, of the University College, Nottingham, has given great attention to the reading of cap heights in different lamps. He gives the following results of experiments made by him with an Ashworth-Hepplewhite-Gray safety-lamp, burning colza oil and petroleum (water white) mixed in equal proportions, and provided with a glass the interior of which is properly blackened behind. Two different sized testing flames were used; in one the wick was drawn down until the luminosity was destroyed as nearly as possible without extinguishing the flame, and in the other the testing flame was reduced in size until the cap just became visible, and in this way a larger cap, or cap of maximum size, was obtained (see Fig. 556).

The heights of the testing flames and of the caps are as follow:—

Per-centage of Methane (Fire-damp) present.	Colza-Petroleum Flat Flame.			
	Small Blue Flame, height 3 mm.		Flame partly Luminous, height 6 mm.	
	Height of Cap, mm.	inches.	Height of Cap, mm.	inches.
2·0	7·5	0·3	7·5	0·3
3·0	7·5	0·3	7·5	0·3
4·0	12·0	0·48	24·0	0·96
5·0	29·0	1·16	41·0	1·64
6·0	67·0	2·68	Enters top of lamp	

From the recorded heights in the above table it is seen that when 3 per cent. or more of gas is present, a larger flame produces a correspondingly larger cap, the larger caps being represented in the Fig. by dotted lines. There is great difficulty in obtaining identical readings of the cap height, even in air containing the same percentage of gas, as the cap-heights differ through the minute variations in the adjustment of the flame in repeated experiments. Further difficulties arise from the charring and crusting of the wick, although this is lessened by mixing petroleum with the colza oil. The use of petroleum alone is disconcerting, because of the formation of a small cap or halo over the flame even in common air.

Testing with the best ordinary forms of safety-lamps is only useful when a greater percentage than 2 is present in the air; any lower proportion completely eludes detection by observations of the flame. The acknowledged danger of lower percentages in an atmosphere charged with fine coal-dust renders it necessary, however, that these should be detected and measured, and this has led to the invention of many fire-damp detectors, which may be broadly divided into two classes, viz., those in which the flame-test is used and those depending upon other means.

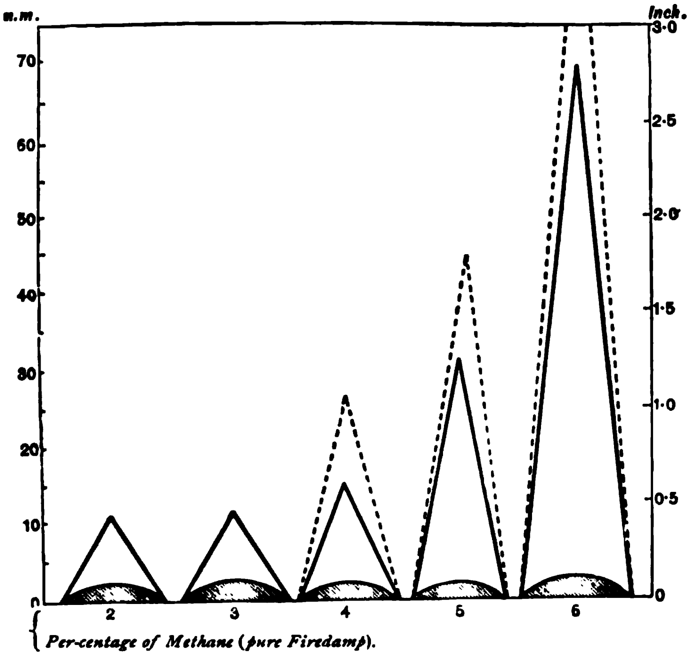


Fig. 556.—Actual Heights of Caps over Colza-Petroleum Flame.

The pale blue flame, 3 mm. in height, is shown shaded. The partly luminous flame, giving maximum caps, is not shown, but the cap heights over it are dotted in.

Ansell's Diffusion Apparatus is a detector of the latter class, which, under favourable conditions, is capable of indicating the presence of either fire-damp or carbonic acid gas. The apparatus was patented in 1865, and described by the inventor, Mr. George Frederick Ansell, at a meeting of the North of England Institute of Mining Engineers on June 2nd, 1866 (see Vol. XV., pages 165—173 of the Transactions). Its action depends upon the difference in the rate at which gases, which differ in density from air, spontaneously blend or mingle with air. If a vessel containing gas be made wholly or partly of a porous substance, such as thin graphite or unglazed earthenware, which will admit of diffusion, and if it be surrounded by air, the gas diffuses out of the vessel into the air through the porous walls more rapidly than air diffuses into the vessel. The relative rates of diffusion of two gases into one another depend upon their relative specific gravities. Thus the diffusion of hydrogen into marsh gas would take place more quickly than that of marsh gas into hydrogen; marsh gas into air more quickly than air into marsh gas; and air more quickly into carbonic acid gas than *vice versâ*. Each gas would exchange places with air according to a diffusion equivalent which can be calculated. In the case of air of uniform temperature, although diffusion would take place, the volumes passing in opposite directions would be equal, and the pressure inside the vessel would remain unaltered. If the vessel is filled with air and surrounded by a mixture of air and marsh gas at the same pressure as the air inside the vessel, the mixture passes more rapidly through the porous substance into the vessel than air passes out of the vessel. As a result of the inflowing volume being greater than that of the air flowing out in a given time, the pressure within the vessel is temporarily increased. Air mixed with fire-damp is lighter than air alone, the mixture becoming lighter in proportion to the percentage of fire-damp. The pressure within the vessel then varies according to the percentage of fire-damp contained in the external atmosphere. By observing the greatest pressure reached in the vessel, having previously noted the pressures produced by surrounding atmospheres known to contain certain percentages of fire-damp, an approximation to the percentage of fire-damp present at any observation is supposed to be obtained. If carbonic acid gas be present in the atmosphere instead of fire-damp, the mixture is heavier than air alone, and the diffusion which then takes place diminishes the pressure inside the vessel. Ansell's indicators are designed to show variations of pressure resulting from the property of the diffusion of gases, such pressures being made apparent in different ways.

The instrument is made in different forms suited to give warning either of a slowly accumulating quantity of fire-damp at a goaf, or of sudden irruptions of the gas, the increase being indicated either by telegraphic signals or by a syphon-gauge.

The particular form of Ansell's indicator, however, which has been most used, consists of a pocket aneroid barometer of the most delicate construction enclosed in a chamber the brass back of which has been removed and replaced by a plate of unglazed earthenware. This plate is sufficiently porous to allow of diffusion between the air inside the aneroid and that outside, but it prevents the entrance or escape of gas or air by mere mechanical pressure. The pressure on the outside of the vacuum chamber in the aneroid acts on a spring, the motion of which is multiplied by a system of levers and so causes a hand to travel over a dial face, which is graduated to inches as in the ordinary aneroid. The porous plate is protected by a brass cover to prevent diffusion taking place at undesirable times. To test for gas, the instrument is taken into the mine, and on reaching the place of trial it is first held by the ring handle until it has become of the same temperature as its surroundings. The brass cap is then removed from the porous plate, and the instrument is held in the suspected atmosphere. In about 45 seconds the maximum pressure is attained, and the index hand must then be read, because after the maximum point is reached, effusion or the passage of gas through the porous plate takes place, and an intermingling proceeds rapidly until the air on both sides of the partition has become similarly mixed with gas and is of uniform composition. All indications of pressure in the instrument then cease to appear and the index-hand travels back to zero. If, in this instance, say 2 per cent. of fire-damp was indicated by the instrument when the maximum pressure was reached, and it be now taken to another place in the mine, where there is, say, 3 per cent. of fire-damp present, only 1 per cent. will be indicated by the index-hand, and in time that indication will disappear because the intermingling here will result in the instrument being eventually filled with air containing 3 per cent. of fire-damp. The 1 per cent. shown on its face must be added to the 2 per cent. previously in the instrument to give the proper amount for the second observation.

This result may be confirmed if the instrument be taken into pure air in an intake gallery or on the surface with the cap removed; after a few minutes the cap must be replaced and the instrument again tried at the second point of observation, when it should indicate 3 per cent. of fire-damp. If pure air is not supplied to it at each trial made, it is obvious that the face indication is of no use but rather misleading, unless taken in connection with the previous reading. Even if pure air could be conveniently introduced into the chamber at each place of observation, correct indications could then only be obtained when such air was at the same temperature and pressure as the atmosphere of the place of observation, and when the air of the mine did not contain a mixture of different gases, or excess of aqueous vapour.

An external atmosphere containing carbonic acid gas is of greater density than the air enclosed within the apparatus, and produces a decrease of pressure within the aneroid during a test. This decreased pressure causes a movement of the aneroid indicator along the scale in a contrary direction to that which indicates fire-damp, so that the same apparatus indicates and measures both fire-damp and carbonic acid gas, or choke-damp in air.

Such an instrument cannot possibly give trustworthy indications of fire-damp under ordinary conditions in the mine, even if it can of carbonic acid gas in non-fiery collieries. The carbonic acid gas given off by lights and by the workmen, mingles with the air of the mine, where its presence tends to counteract the effect produced upon the Ansell indicator by emissions of fire-damp. The indication which would be produced in a mixture of fire-damp and air may be reduced, neutralized, or even reversed by the addition of carbonic acid gas, or the indication which would be produced in a mixture of carbonic acid gas and air may be reduced or neutralized by the addition of fire-damp to the mixture.

Again, the hygrometric condition of the air affects the indications. Water-vapour is less dense than air, and its presence tends to magnify fire-damp indications, or produces indications which may be mistaken for those of fire-damp. Further, the porous-plate in a dusty mine soon becomes clogged with fine dust and prevents diffusion. And it must be added that the apparatus is not sufficiently delicate to register slight traces of gas, since the presence of 1 per cent. of fire-damp in the air produces but a slight pressure record.

The pressure in inches of mercury corresponding to percentages of gas is given by Mr. Ansell in the paper referred to as follows:—

With rising Barometer.
Firedamp.

			inches.
1	per cent.	=	0·010
3	,,	=	0·030
5	,,	=	0·060
7	,,	=	0·080
7½	,,	=	0·090
10½	,,	=	0·130
12½	,,	=	0·170
15	,,	=	0·220
20	,,	=	0·320
50	,,	=	0·800
100	,,	=	1·680

With falling Barometer.
Carbon Dioxide.

			inches.
1	per cent.	=	0·015
3	,,	=	0·050
5	,,	=	0·080
10	,,	=	0·160
15	,,	=	0·240
20	,,	=	0·330
50	,,	=	0·820
100	,,	=	1·640

From this description of Ansell's indicators it is plain that although capable of revealing the presence of gas under certain conditions, their use in the mine is apt to mislead, because the recorded indications are produced by changes of pressure without there being any means of ascertaining how much of such pressure is due to any particular influence acting on the instrument.

The Forbes Damposcope.—This is the invention of Professor George Forbes, and is intended to determine the percentage of gas in the atmosphere by measuring the length of that part of a tube containing the air to be tested which gives the maximum resonance to a musical note of standard pitch. The length of the column of air in the tube depends upon its density, and this fact is made use of as a means of calculating the density of the air in the mine. The instrument consists of a brass tube 1 inch in diameter, and between 6 and 7 inches long. A tightly-fitting piston can be moved up and down the tube by means of a rack and pinion. The graduated pinion-head is 3 inches in diameter, and its complete revolution causes the piston to move 1 inch. An index attached to the tube over the graduated pinion-head enables readings of the piston movements to be taken. The tube has an open end, in front of which is fixed a tuning-fork, capable of vibrating 512 times per second. This tuning-fork is first set in vibration when the instrument is in pure air, and the piston is moved backwards and forwards until in the position which gives the strongest resonance. The index reading is then recorded, or, if preferred, it can be adjusted to zero. The tube is then filled with the atmosphere to be tested, the tuning-fork again vibrated, and motion given to the piston, until that position is found which again gives the most intense resonance. The graduated pinion-head is again read and compared with the previous reading, thus showing the increase or decrease in the length of the column from the open end of the tube. If the temperature has remained unchanged, and the air tested contain only marsh gas, the percentage of gas is calculated. A thermometer is fitted to the instrument, which, it is thought, to some extent enables corrections to be made to readings if changes of temperature occur, or percentages of gas corresponding to the value of the scale are recorded for different temperatures in a table based upon earlier experiments with air containing known percentages of gas.

The indication of gas by this instrument is subject to the same sources of disturbance and error as the Ansell aneroid, notwithstanding the attempts to apply corrections. There is also some difficulty in adjusting the piston in the exact position which produces the strongest resonance, and untrained observers will probably adjust differently. The necessity to ascertain and make allowance for the causes of disturbance before a proper estimate of fire-damp can be obtained by these instruments, makes them too complicated for ordinary use in the mine.

The Hardy Detector.—This is another acoustical indicator in the use of which the composition of the atmosphere is supposed to be determined from observations of the beats produced by two sounds nearly in unison, so that it does not require to be used by a trained observer with a sensitive ear, as in the case of the Forbes damposcope. When two streams of pure air are directed at the same time from two distinct organ-bellows to two organ-pipes, giving the same note, the sound produced is of a single note. If, instead of fresh air, one of the bellows be supplied with a mixture of air and gas, one organ-pipe does not give its natural sound, but a note modified by the column of air and gas in the tube. The two pipes thus acted on simultaneously give two sounds which are not in unison, and, therefore, distinct beats are produced. These beats are more or less frequent in proportion to the gas in the air, and are easily recognized and counted by an unpractised observer.

The Hardy detector consists of two separate bellows and two organ-pipes. One of the bellows and its organ-pipe are enclosed in an air-tight compartment containing pure air; the other bellows receives its supply from an atmosphere to be tested, which it directs to the other pipe. An experiment may be made in a few seconds. The organ-pipes both sound the same note in pure air. If the mixture under examination be composed of air and marsh gas, the following results are obtained :—

For 1 per cent. of marsh gas in air there is about 1 beat per 3 seconds.
„ 2 „ „ there are 3 beats 2 „
„ 3 „ „ „ 2 „ 1 „
„ 4 „ „ „ 3 „ 1 „
and so on, the beats increasing in frequency in proportion as the mixture contains more marsh gas. With 12 per cent. of gas there are about 9 beats per second; with 20 per cent. they become very rapid, but are quite distinct and clear.

It is intended that the instrument shall be used with organ-pipes of a higher note when testing for gaseous mixtures whose density is almost the same as that of pure air. Carbonic acid gas and fire-damp mixtures give similar results, and the instrument, therefore, although ingenious, is of little practical use. The amount of aqueous vapour, carbonic acid gas, and other gases which may be present in the atmosphere when testing for fire-damp, cause considerable disturbance in the results obtained.

Some attempts have been made to produce indicators in which the apparent weights of a body suspended in air, in fire-damp, and in a mixture of fire-damp and air, are made to determine the percentage of marsh gas in the mixture, but no practically useful instrument of this kind has yet been devised. The apparent weight of a body in common air is its true weight, less the weight of that atmosphere which is displaced by it. The density of marsh gas is less than that of air, at the same temperature and pressure, hence the weights of equal volumes of gas and air mixtures differ in proportion to the amount of gas present in such mixtures. The apparent weight of a body in these mixtures under given temperatures and pressures, will vary in accordance with the relative volumes of air and gas in the mixtures.

That such instruments are useless for general inspection of the mine is apparent when mixtures of air, fire-damp, and carbonic acid gas may be so blended as to make a given volume of precisely the same density as that of pure air from which, therefore, it could not be distinguished in the mine.

The Aitken Indicator.—In this, the suspected atmosphere in the mine is drawn through a tube in which are placed two thermometers, side by side. The bulb of one thermometer is naked, while that of the other is coated with platinum-black. If fire-damp be present in the air under examination, the platinum-black covering the bulb produces a slow combustion. The rise of temperature above that of the entering gaseous mixture partly passes from the coating to the bulb, in consequence of which a rise in temperature is registered by the coated thermometer above that of the uncovered one. The difference of temperature indicated by the two thermometers will increase as the proportion of gas present in the air increases.

There are many objections to this instrument. Mr. Aitken detected with it small quantities of ordinary illuminating gas in the air, and of artificially prepared marsh gas, but experiments with natural fire-damp gave unsatisfactory results. The gaseous mixture had to be raised to the temperature of boiling water, and this would be impracticable in the mine. The spongy platinum deteriorates by absorption of moisture, and by the deposit of dust upon its surface. The method of testing is incapable of giving standard results. There is also some doubt with regard to the safety of using the apparatus at all in the mine. In a high percentage of fire-damp the platinum may attain a red heat and so cause an explosion of the surrounding gaseous mixture.

Angus Smith's Air-Compression Syringe is another instrument for ascertaining the presence of fire-damp in the air. It consists of a glass tube closed at one end in which works an air-tight piston. The tube is filled with the atmosphere to be examined, which is then rapidly compressed by forcing down the piston by hand. The heat generated by the compression is generally sufficient to produce the combustion of the gas within the tube when not less than 5 per cent. of marsh gas is present. The flash may be seen in the tube through the glass. If a small quantity of platinum-black is introduced at the closed end of the tube, a flash may be produced in air containing a smaller percentage of marsh gas, but no flash occurs with less than 2·5 per cent. of the gas.

This instrument has never come into general use. An ordinary Davy lamp in skilful hands is capable of making tests as delicate as this compression syringe, the use of which would be attended with considerable danger of explosion if the glass were to be broken by the violent concussion inside the tube when testing. No measure of the different proportions of fire-damp present can be obtained by the compression syringe.

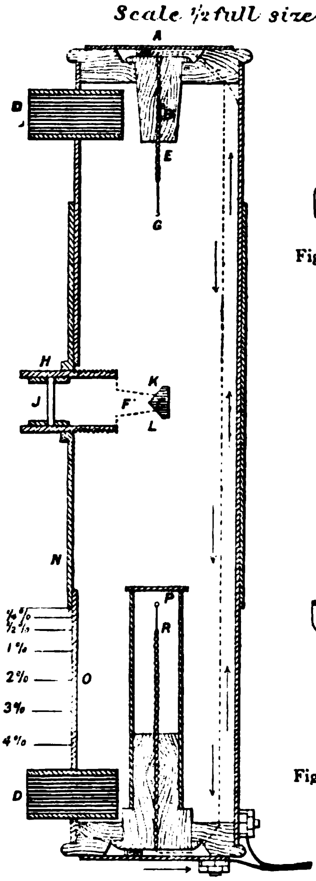


Fig. 557.

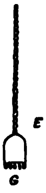


Fig. 558.

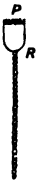


Fig. 559.

ENLARGED VIEW OF THE SCREEN C & SIDE TUBE F

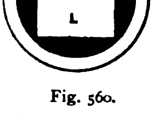


Fig. 560.
LIVEING'S FIRE-DAMP DETECTOR.

By the aid of *Liveing's* firedamp detector,* very small quantities of gas in the underground workings may be detected. It is shown in Figs. 557–560 and consists of a brass tube about eight inches long by one-and-three-quarters inches diameter, closed at the ends by discs of hard wood. Through these discs are passed the insulated connection wires. At G and P these wires carry fine platinum spirals, the one at P being enclosed in an airtight tube R, which is filled with pure air and provided with a glass cover at the end facing the second spiral, and that at G being exposed to the air within the instrument. Between the platinum spirals is supported a small wedge-shaped screen C by means of an arm F. The two surfaces of this screen are covered with white paper and can be seen through the glass disc J in the side tube H, to which the arm and screen are attached.

By means of a battery which accompanies the detector, an electric current is passed as shown by the arrows till the spirals attain a red heat. One side of the screen C is illumined by the covered wire P, the other by the working wire G, the wires themselves being hidden from view whilst the instrument is in use.

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* See Transactions, North of England Institute of Mining Engineers, vol. xxviii., pp. 167—170.

On passing the electric current under ordinary conditions of the air inside the instrument, the two surfaces of the screen C appear equally illuminated on being viewed through the glass disc J; but if inflammable gas is present that side of the screen which is opposed to the working wire G appears brighter than the other. To measure this inequality of brightness, the screen C is moved towards the covered wire P, the movement being effected by sliding the side tube H in that direction. After the screen is so adjusted as to make its two surfaces appear equally illuminated, the amount of movement given the side tube is made to read a graduated scale showing the corresponding per centage of gas present in the air from ¼ to 4, as marked on the drawing, Fig. 557. An index is fixed to the outer tube N by means of which the per centage is read. The side tube H which carries the arm F and screen C, is screwed into the outer tube N of the instrument and a slot is cut in the body of the instrument to allow the necessary travel for the side tube H towards the covered wire P. In no position of the screen is the slot in the instrument uncovered. D D are two Hemmings jets, which admit of the air being drawn into the instrument for examination through bundles of iron or copper wire in the tubes. This is effected by placing the mouth of the observer over the lower orifice and drawing in a breath. The outer air then enters the instrument. The platinum spirals are heated red by passing the electric current, and the photometer (light-measurer) is adjusted by the sliding movement given it if inflammable gas is present. When two or three per cent. of gas is in the mixture the difference in colour of the light causes some difficulty in judging correctly the precise position of the screen. To overcome this difficulty, one half M, Fig. 560, of the side of the screen next G is covered with yellowish red paper, the other half K being left white. For quantities up to 1 per cent. K is compared with L, but in larger quantities M is employed instead of K, as its tint resulting from the yellow surface neutralizes the very white light given by the working wire G and gives facility for accurately judging the position of the photometer. In case of an explosion inside the instrument it cannot be communicated to the outside, as the hot gases become cooled in passing through the bundle of iron or copper wires in each of the entrance tubes.

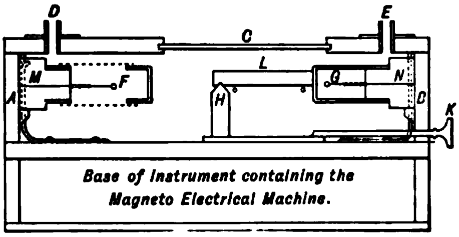
D
E
C
L
M
F
Q
N
A
H
B
K
Base of Instrument containing the
Magneto Electrical Machine.

Fig. 561.—Liveing's Electrical Indicator.

Another form of the Liveing Indicator suitable for use in the mine is shown in Fig. 561. Here the sloping surfaces of the block H can be moved by the rod K along the graduated scale L until the two inclined faces are seen to be equally illuminated. The position of the point of junction of the two screens upon the scale then indicates its position relative to the two spirals, and the percentage of fire-damp present in the air. There is a narrow glass window, C, to enable the observer to adjust the block H, and read the scale. The spiral G is enclosed in an air-tight metal tube containing pure air, and is, therefore, never reached by the gas. The spiral F is surrounded by a cylinder of wire gauze which removes the necessity of placing Hemmings' jets at the openings D and E. These are unobstructed, but if desired, an india-rubber tube can be fixed to one entrance tube having its open end at any place, such as a cavity in the roof, the atmosphere of which is tested by being drawn by suction from the mouth into the space in which the spirals and the recording apparatus are enclosed. The air at once reaches the spiral F, through the wire-gauze, while it is excluded from the spiral G. The light emitted by the spirals reaches the photometer through glass plates, which close the wire-gauze tube M and the metal tube N.

In the base of the instrument is a small magneto-electric machine. By turning a handle on the outside of the box which encloses the apparatus, an electric current is produced, and this passes through the two precisely similar platinum wire spirals, heating them to moderate redness. An increase in the velocity of rotation of the handle increases the electric current in the wires, and correspondingly raises their temperature and increases their brilliancy. The rate of rotation may vary within certain limits, but should be such as to heat the spirals to dull or moderate redness.

Fig. 562 shows the indicator being employed in the mine, where it has accurately shown the presence of fire-damp ranging from 0·25 upwards. When constructed, the scale of the instrument is divided to suit two precisely similar spirals, which in pure air are rendered equally bright by the same current. In its new condition the instrument is capable of giving accurate results for high and low percentage tests. After being in use a comparatively short time, however, the wire spiral which is exposed to the gas becomes permanently altered in its electrical resistance. The scale divisions then give erroneous percentages of gas present in the atmosphere tested, and it becomes necessary to set the zero of the scale afresh in air free from gas. The scale admits of being shifted to a slight extent, and when adjusted, the indications of gas it gives, although not strictly accurate, are sufficiently so for practical purposes. A difficulty, however, arises from the fact that the above alteration in the spiral is constantly increasing, and thus requiring re-adjustment. It may not be easy to obtain fresh air for the adjustment, if that becomes necessary in the mine, and, if the wire spiral undergoes considerable change during one round of testing, the adjustment of the zero may be impossible owing to the limited range of motion allowed to the scale. In that case two new spirals must be inserted in the place of those in the instrument, and this usually causes serious delay.

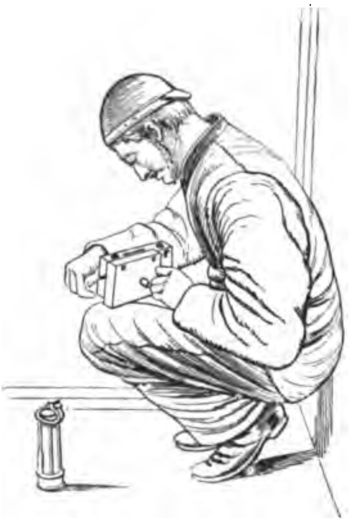

Fig. 562.—Liveing's Indicator in use in the Mine.

An objection to the use of the Liveing Indicator in the mine arises from the fact that it furnishes no light, and is so much dead weight to be carried about by the firemen or other official, in addition to a safety-lamp for lighting purposes. After being in use some time there has been difficulty in obtaining, by means of the magneto-electric apparatus, a current of sufficient steadiness and constancy to ensure accurate readings. In unskilful hands the handle may be too rapidly rotated and thus produce a current capable of fusing and rendering useless the exposed platinum spiral. A less rapid rotation of the handle may bring about the same result if much gas is present.

Lewis and Maurice's firedamp indicator (the invention of Sir William Thomas Lewis and Mr. A. H. Maurice) is shown in Figs. 563–566.* E is an air chamber, containing about 2 cubic inches of air; in it the mixture of gas and air to be tested is burned. A gauge J has an ivory scale attached to its upper limit F, graduated to read to $\frac{1}{4}$ per cent. of firedamp. A light brass shield H protects the upper limbs F G of the gauge. The bottom of the air chamber consists of a brass cap, fitted with a leather washer, to ensure its screwing on quite air-tight. The bottom of the air chamber is shown unscrewed in the sectional elevation in Fig. 563, but screwed on in the side elevation in Fig. 564. It is removed to admit of the entrance of the sample air to be tested. K is a platinum wire $\frac{3}{4}$ of an inch in length and of a size

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* See Transactions, South Wales Institute of Mining Engineers, vol. xv., pp. 39—43.

Scale. ½ Full Size.

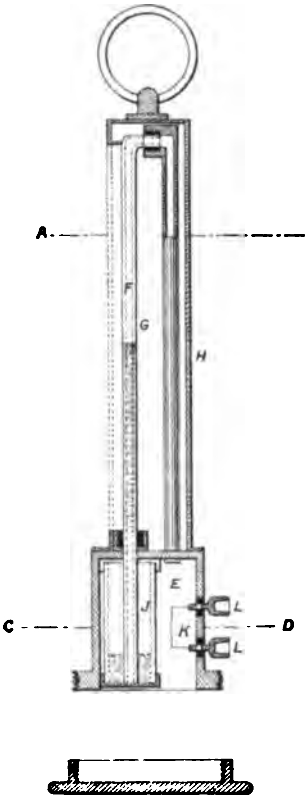

Fig. 563.

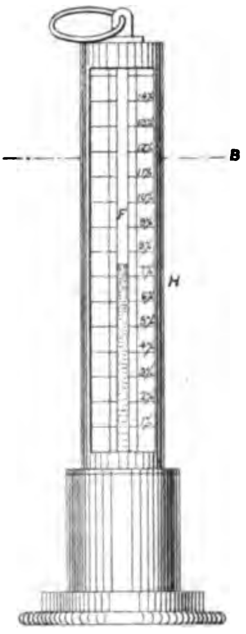

Fig. 564.

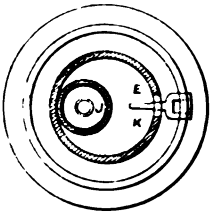

Fig. 565.—Section through C D.

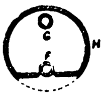

Fig. 566.—Section through A B.

LEWIS AND MAURICE'S FIRE-DAMP DETECTOR.

suitable to offer such resistance to the electric current passing through it, as to heat it to redness. It is placed in the air chamber as shown in the figure, and as the air chamber is made of brass, the battery connections are insulated at L L where they pass through the side of the instrument.

J is a glass cylinder forming part of the gauge, 2 inches high and ·8 inch in diameter, closed at the top and bottom with brass caps cemented on. A small glass tube F extends from within ·05 inch of the bottom of the glass cylinder J, through its upper end to a vertical height of about 7 inches, where it is bent back to form the descending tube G, parallel to F till it terminates within the air chamber. Coloured glycerine is placed in the gauge-glass J, till it reaches to a height of half-an-inch. The remaining portion of the gauge-glass cylinder J contains air at a pressure which exercises sufficient force on the glycerine to drive it up the glass tube F till it stands about half way up the tube. The sectional area of the cylinder J compared to that of F, is as 30 : 1, so that on the air in J expanding $\frac{1}{30}$th of an inch, the displaced liquid will rise 1 inch in the tube F. The height of the air column in the cylinder J is 1½ inches. When it expands 1 per cent. of its volume, the liquid enclosed in J will be displaced to an equal extent, or 1 per cent. of 1·5 inch = ·015 inch and it therefore rises in the tube ·015 × 30 = ·45 inch, but allowing for the back pressure due to the weight of the liquid in the tube F, it really rises ·4 inch. It is thus seen that a reduction of the normal air pressure of 1 per cent. in the air chamber E will cause the confined air in J to expand 1 per cent. and in doing so drive the liquid ·4 inch higher in the tube F. The reduction in air pressure in the chamber E is brought about by burning out the firedamp which forms part of the mixture being tested, resulting in a partial vacuum in the air chamber.

To make a test with the indicator, the bottom of the air chamber is unscrewed, and the instrument held on its side, with the open end turned to the air-current so that the air enters and fills it. If out of a current or where there is not sufficient motion in the air, the instrument should be moved to and fro a few times. When the air chamber is thus filled, the cap is screwed on and the index finger or pointer attached to the scale is moved along it till it is level with the top of the liquid in the gauge-glass. The indicator is attached to a battery which accompanies it, or a pocket accumulator designed for the purpose, and the current passed for 14 seconds. The per centage of gas present in the mixture (if any) will be shown by the liquid in the gauge-glass rising above the index finger, and this amount may be read off from the scale, which is graduated to ¼ per cent. but if smaller gradations are required they can be indicated.

The gas is not ignited, as in ordinary combustion, by the indicator, the temperature of ignition not being attained by the platinum wires, but as it in effect burns in the closed chamber a chemical change takes place. The platinum wire at a red heat burns the carburetted hydrogen. During the process, oxygen and hydrogen combine to form water which is dissipated in vapour and then condensed to liquid, and carbonic acid gas is formed. As a result of the chemical changes, the products of the combustion plus the remainder of the air, occupy less space than that occupied by the original mixture of air and gas, and as the burning takes place in a closed vessel a partial vacuum is formed in it. It will be understood how the creation of this partial vacuum disturbs the pressure in the confined cylinder J.

It is necessary to accurately time the passing of the current, as the 14 seconds is the time required to burn half the gas in the air chamber and in graduating the scale the gauge is made to show the full quantity of gas when only half has been burned out, the rate of burning being uniform.

By placing the cylinder forming the lower portion of the glass-gauge J within the air chamber E, the air in both is subject to the same temperature at all times. If this were not so, the expansion due to heating the air in the air chamber by the platinum wires during a test, would depress the liquid in the gauge, but this is obviated by bringing the enclosed air in J under the same heating influence as that in E. Both may expand, but they do so equally, so that the gauge is not affected when a test is made. Thus when the cap is unscrewed from the bottom of the instrument, the level of the liquid may be disturbed in the same way as that in a thermometer, but whatever level it assumes, after screwing on the cap with the test sample of air enclosed, the index finger is made to agree with it by sliding it along the scale.

The indicators of Monnier, Coquillion, and Le Châtelier belong to the same class as that last described, all depending upon the contraction in volume or the change of pressure effected by the combustion of the marsh gas present in an enclosed volume of air. The complete combustion of the fire-damp present in the air would reduce the original volume of the mixture of air and gas by very nearly twice that of the fire-damp present. The decrease of volume, instead of being measured, is usually indicated by the decrease of pressure it causes in a closed vessel connected with a suitable gauge.

Professor Monnier's instrument is too costly and too complicated to come into general use, but is a very ingenious invention.

M. Coquillion's consists of three glass tubes connected together, the central one being of small bore and graduated. Above it is a glass tube of larger diameter closed by an air-tight cover in which is a small stopcock. Two insulated platinum wires traverse the tube, and these are connected inside by a fine platinum wire. The glass tube at the other end of the graduated tube is attached to a small pear-shaped india-rubber bag. The glass portions have an outer brass casing with slits on opposite sides for reading the scale. Water is poured into the instrument until it fills the bag and rises to a certain mark on the graduated scale.

To make a test with the instrument the stopcock is first opened, and on the bag being squeezed with the hand the water rises up the tube and expels the air contained in it; on releasing the bag the water returns to its former position and the upper tube of the instrument becomes charged from the surrounding atmosphere. To ensure the effectual expulsion of the original volume of air, and that none be left to dilute the atmosphere to be tested, this process is repeated several times, after which the stopcock is closed. The position of the surface of the water in the graduated tube is then read and the fine platinum wire is raised to a white heat by a current of electricity produced by a battery or a magneto-electric machine, which continues for two or three seconds. This operation is repeated two or three times at intervals. The fire-damp in the instrument is thus consumed; the resulting aqueous vapour is condensed when the instrument cools, causing a decrease in the pressure in the tube above the water. The atmospheric pressure on the bag then causes the surface of the water in the graduated tube to rise. The height of the water is now read and compared with the previous reading, so as to enable the observer to calculate the percentage of marsh gas present in the atmosphere.

It takes five minutes to make a single experiment with this instrument, so that it could never be used largely in the mine. Moreover, the indications are rendered inaccurate by the exposure of the air containing the products of combustion to water. The carbonic acid gas produced by the combustion of the fire-damp is soluble in water and may be partly or wholly dissolved according to the proportion present and the length of exposure. The results, therefore, are doubtful.

Le Châtelier removes this source of error by substituting mercury for water. He further quickens the period of the experiment by surrounding the test-vessels with a water-jacket. The products of combustion are thus more quickly cooled, and the periods of waiting for the true pressure readings are lessened.

These forms of apparatus are not well adapted for underground use, because they are not sufficiently simple and portable. A safety-lamp has to be carried with them and an apparatus for producing an electric current. They are easily deranged, and it is impossible to take the same care of them underground as when conducting experiments in the laboratory.

Other forms of apparatus have been devised to ascertain the relative amounts of inflammable gas and of air which are necessary to bring the gaseous mixture to the lowest limit of ignition when placed in contact with a flame. The simplest form of this class of indicator is *Le Châtelier's Eudiometer*, shown in Fig. 567. It consists of a glass tube open at its larger end and closed at the top. The upper portion is narrower than the lower and is graduated so that each line of division marks $\frac{1}{1000}$th part of the capacity of the tube when filled to a mark near its larger end. If the gaseous mixture to be tested is not inflammable, the tube is filled with water and inflammable gas is introduced, the volume passed up being indicated in the graduated tube. The air to be tested is then passed into the tube until it is filled to the standard mark. The thumb is now placed over the mouth of the tube so as to close it and the tube is removed from the water. The tube is well shaken while the thumb is kept over the mouth, the water in the tube helping to mix the gases thoroughly. A light is then placed to the mouth of the tube in a darkened room. If the mixture ignites, another test must follow with an admixture of a smaller proportion of inflammable gas. These experiments are repeated until it is known what minimum addition of the gas is necessary to produce kindling by flame. The exact amount of this minimum volume being known, it is easy to estimate the proportion of fire-damp originally present in the air under examination.

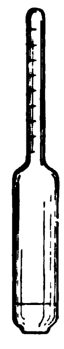

Fig. 567. — Le Chatelier's Eudiometer.

If the air which is to be tested is already inflammable, the mode of procedure is similar to that just described, except that pure air must be added instead of inflammable gas. Coal-gas is generally used for the inflammable gas, as a matter of convenience, but trustworthy results cannot be obtained if the coal-gas be of variable composition. In any case such results are slowly and tediously obtained. The method is altogether unsuited for use in the coal-mine from the fact that it requires a naked light, but if samples of air can be transported and tested in a darkened laboratory on the surface it may be of service. Some simpler means, however, of obtaining accurate results are more likely to be adopted.

The Shaw Gas-tester.—This apparatus determines the composition of the atmosphere on the same principle as the eudiometer last described. The apparatus occupies a surface of about 2 feet square; it is made of brass and iron, and consists of two pumps, one of which takes in pure air or the air to be tested, and the other pure gas or pure air, according to circumstances. The air-pump is stationary, the piston stroke being always constant, while the gas-pump is movable and can be set between two graduated bars, so that it will pump any desired percentage of gas in conjunction with air from the air cylinder. The sum of the two pumps always equals 100 parts and affords a basis from which to measure. A hand crank is used to set the beam in motion, and this in turn actuates the pump pistons, one of which has its stroke regulated by an adjustment of its position on the graduated lever. The product of the two cylinders is pumped through a mixer in which is a definite quantity of the sample to be tested, into an igniting chamber provided with an aperture on one side in front of a gas jet. The proportions are adjusted until the mixture is just brought to the lowest limit of ignition at the flame. There is a contrivance in the apparatus for automatically indicating the kindling. The expansion caused by the heat propels a loose piston head, held in place by a bowstring, against a gong in the chamber, and the sound can be heard by the observer. The apparatus is large and heavy, and is adapted for laboratory, not underground use. The sample of air is captured in the mine by means of a diaphragm hand-pump and a 6-gallon rubber bag, and is brought to the gas-tester. The Shaw gas-tester is an extremely sensitive instrument, and in its normal condition is capable of giving indications of great accuracy and delicacy. It is said that the addition or subtraction of only 0·1 per cent. of fire-damp will cause the bell to ring or remain silent. The lowest kindling proportion of the inflammable gas which is to be measured when mixed with air must first be determined by the apparatus. A calculation of the proportion of inflammable gas present in the sample of air and gas tested can then be made by the operator first reading off the proportion of air or gas actually added to and mixed with the sample under examination.

The apparatus is complicated and costly, and will probably require constant attention to be maintained in a state of efficiency. The collection and transport of samples of mine air are attended with considerable inconvenience and delay, and there is also risk of leakage or change of composition in the samples. By the time such samples reach the testing apparatus the condition of the atmosphere in the mine may have undergone a change, and no inference can be drawn of the state of any one point at the particular moment the test is being made. The apparatus is certainly very ingenious, but simpler and more portable forms of fire-damp indicators are to be obtained.

The Flame-test class of Detectors are certainly the most convenient forms of apparatus. They can be used at once in the mine with safety, and experiments can be made with them with ease and rapidity. The presence of carbonic acid gas, or of aqueous vapour within the limits in which these occur in the coal mine, do not appreciably interfere with flame-tests, and the results obtained are not appreciably affected by the variations of pressure and temperature of the air of the mine under ordinary conditions.

The Davy lamp has long been a means of detecting gas by means of the flame-cap. A very useful adjunct to the Davy and other lamps has for several years been in use in *Garforth's Ball Detector* (see Fig. 568).

The Royal Commissioners thus speak of Mr. Garforth's invention :—

" For extracting the gas from crevices Mr. Garforth uses a small hollow india-rubber ball provided with a metal nozzle. The ball being compressed in the hand so as to expel the greater part of the air contained in it, the nozzle is inserted into the crevice or cavity and the ball is then allowed to expand to its natural size. It of course becomes filled with the gas mixture existing in the place under examination, and this gas may be introduced into a safety-lamp through a gauze-protected channel by again compressing the ball. For thus testing the gas a bonneted lamp is used, through the oil cup of which passes, close to the wick tube, a narrow pipe containing gauze diaphragms, and terminating below in a tube which just fits the nozzle of the india-rubber ball. The external end of this tube is closed by a spring valve, and when the nozzle is introduced it opens the valve and the gas can be forced up the pipe and on to the lamp flame. This simple and extremely portable apparatus seems to answer its purpose perfectly, and the addition to the lamp of the pipe above-mentioned does not affect either its security or its illuminating power."

This detector may be very useful in obtaining samples of gas mixtures from holes above the roof too small for the insertion of a Hepplewhite-Gray lamp, but it is incapable of detecting percentages of firedamp below 2 per cent. in the air, and does not measure or register any proportion of gas and air. Its use is likely to decrease now that several lamps are made to take their air supply from the top.

Safety-lamps have been arranged to give an alarm with or without the aid of an electric current. In Mr. Hyde's alarm arrangement a large Davy lamp is enclosed in a metal case provided with openings which may be closed by shutters sliding in vertical grooves. On an ignition of gas in the lamp a loop of thread is burnt; the shutters should then fall and cut off the air supply. The falling shutters cause two clockwork bells attached to the case to ring at the instant the lamp is extinguished. This and similar appliances are altogether untrustworthy.

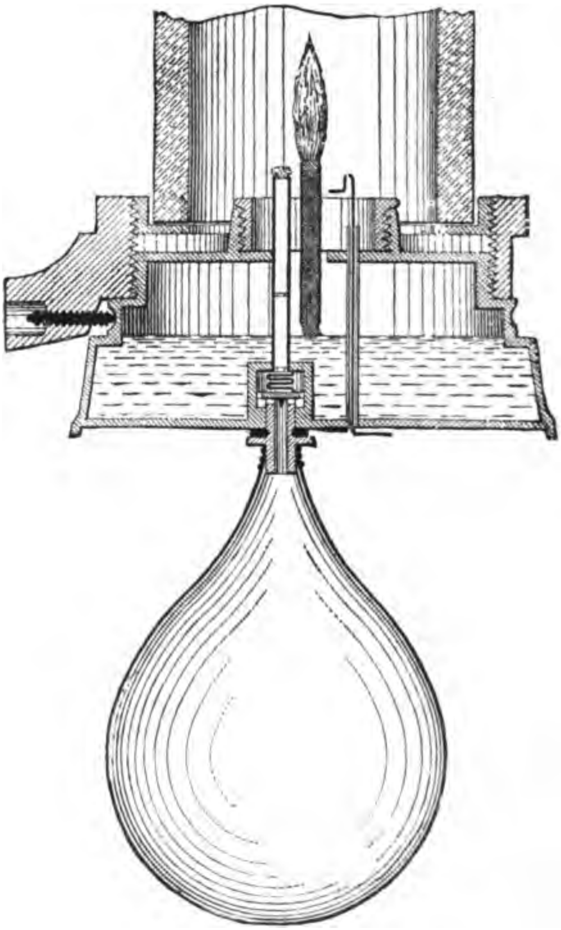

Fig. 568.—Garforth's Fire-damp Detector.

The reduced oil-flame, used for testing with the Davy lamp, is too luminous to allow of caps being seen when small percentages of fire-damp are present. The wire gauze surrounding the flame obstructs the light and prevents pale caps from being seen, and the background of the gauze gives no help toward making the caps visible. There is no means of adjusting the oil-flame to any standard size, and the size and appearance of the flame-caps seen over any particular flame vary considerably with the dimensions of the flame.

MM. Mallard et Le Châtelier's Safety-lamp.—The improvements these gentlemen made consisted in introducing metal screens to conceal the reduced oil-flame from the eye, as shown in Fig. 569. By this arrangement the flame could be brought approximately to standard height by drawing down the wick with the pricker until the tip of the flame was level with the upper edge of the screen. Instead of wire-gauze glass was used to surround the flame, and this facilitated an observation of the cap which was seen against a dead-black background eminently adapted to render it visible.

These improvements enabled lower percentages of gas to be detected than was possible with the Davy lamp. The charring of the wick in burning, however, prevented the formation of a strictly standard flame, and that obtained did not give distinct and easily visible caps in the presence of small percentages of fire-damp. An objection to the arrangement was, that the light emitted by the flame when raised to its proper light-giving height was obstructed by the screens, and its value for lighting thereby reduced.

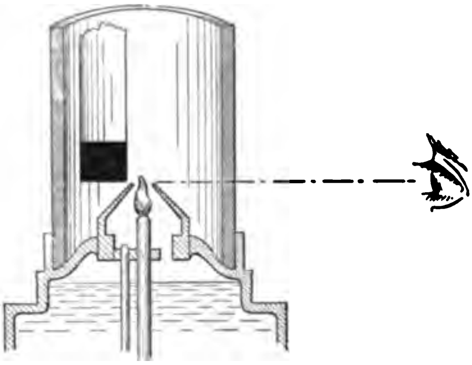

Fig. 569.—Metal Screens for the Concealment of reduced Lamp-flame.

The disadvantages arising from the necessity of using a reduced flame in order that it might not be luminous, led MM. Mallard et Le Châtelier to adopt a flame which was non-luminous, even when of larger dimensions than the reduced oil-flame. Alcohol and hydrogen flames were both experimented with. With a large alcohol flame 0·5 per cent. of fire-damp was detected, and with a hydrogen flame as low a percentage of the gas as 0·25. It was proposed by these gentlemen to use the testing-flame in a separate safety-lamp, and to carry an ordinary safety-lamp as well in the mine for illuminating purposes. They attempted the generation of hydrogen gas by chemical processes in the oil-vessel of a safety-lamp, intending, had they succeeded, to burn this gas in a regulated stream from a jet arranged in the lamp and connected with the reservoir. The attempt failed; the method of producing the gas was impracticable, and they were unable to regulate the size of the flame.

The *Pieler* Lamp, shown in Fig. 570, is used solely for examining the air in a mine. When first invented it was intended only to use this lamp in the laboratory to test samples of air brought out of the mine. It has the usual vessel, G, to hold the burning medium, which in this case is pure alcohol. To prevent the escape of vapour of alcohol the lamp is very carefully constructed. The reservoir containing the spirit becomes heated when the lamp is in use, and produces large volumes of spirit-vapour, which, on attaining a sufficient pressure, may force the alcohol up the tube in the form of spray, more especially if the wick-tube extends downwards to reach nearly to the bottom of the reservoir. To provide against this the reservoir must not be charged beyond a certain height, and in addition, the wick-tube and the tube which surrounds it above the surface of the alcohol are pierced by small holes, through which the vapour, as it is formed, escapes, to be consumed in the flame. A cylindrical wick, E, is used. It is made of silk, and passed over a tube, F, provided with a small nut inside. Upward or downward motion is given to the wick by means of the screw, D. A short conical chimney, C, open at the top and base, is attached for protecting the eye of the observer. Care must be taken to adjust the lamp-flame, so that its upper extremity is exactly on a level with the upper edge of the chimney. A gauze, B, similar to the Davy lamp, but of unusual length, is used. The conical gauze has 1,024 apertures to the square inch, is 2·3 inches in diameter at bottom, 1·75 inch in diameter at top, and 7·75 inches high. The top of the gauze cap is covered by a perforated metal plate. The object of the long gauze is to allow the cap to develop itself more freely inside when the lamp is in an explosive mixture. The lamp is provided with the usual bars or pillars, A, for supporting the top. The spirit-chamber is made large enough for the lamp to burn while used for a prolonged journey through the workings.

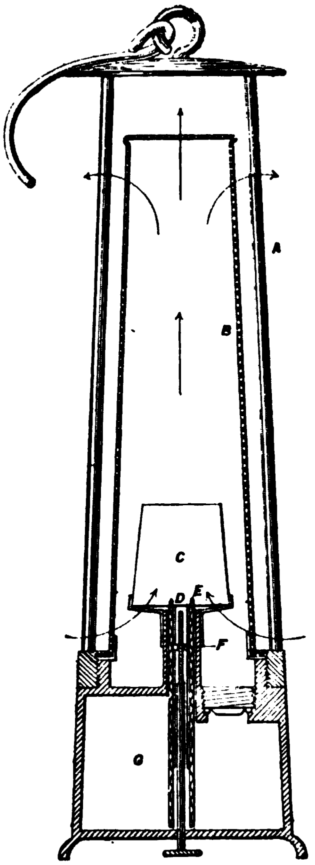

Fig. 570.—Pieler Lamp.

When the firedamp is mixed with the atmosphere in any proportion greater than ¼ per cent. its presence is detected by the Pieler lamp. The cone of light shown at the flame when in an explosive mixture is larger than that produced on the flame of any lamp using vegetable or animal oils. This sensitiveness renders it very valuable for some purposes.

According to experiments made :—

The cap with ¼ per cent. of firedamp is of a bluish-gray colour, and but slightly luminous; it has a height of 1⅛ inches.

With ½ per cent. the cap reaches 2 inches, is more sharply defined at the bottom, but fades away above.

With ¾ per cent. the cap reaches 3 inches, the edges are sharper, and the colour more blue.

With 1 per cent. the cap reaches 3½ inches; edges more sharply defined; colour deeper blue.

With 1¼ per cent. the cap reaches 4 inches.

With 1½ per cent. the cap reaches 4¾ inches.

With 1¾ per cent. the cap reaches the top of the lamp, the luminosity is increased in proportion, and the colour of the cap is deep blue.

With 2 per cent. the cap widens out at the top, and with higher percentages it continues to expand until the gauze is filled.

The lamp is thus described in the Report of the Royal Commission on Accidents in Mines, 1886:—

"Pieler's is a large Davy lamp, constructed to burn alcohol with an argand wick. The air supplied to the inner part of the flame is admitted by a tube, protected by superposed discs of gauze, which passes vertically through the vessel containing the alcohol. Around the flame is a short, conical chimney, open above and below, and the flame is so regulated that it does not appear above the chimney, its height being, therefore, from 1 to 1·25 inch. In gas this spirit-flame yields a much more conspicuous cap than can be produced by the flame of ordinary vegetable or animal oil. . . . The Pieler lamp is obviously a most sensitive gas-detector, but in its present form it is quite inadmissible for use in well-ventilated mines, for the following reasons. The flame is easily extinguished by a very moderate current, and if the lamp happens to come into an explosive mixture of gas and air an explosion is almost certain to be caused in a few seconds. The lamp could be rendered less dangerous for general use by enclosing it in a case, and, as far as we have been able to observe, its power of indicating the presence of gas would be but little if at all impaired." The weight of the Pieler lamp without oil or wick is about 2 lbs. Since it was first used the lamp has been rendered much safer by the addition of a bonnet, which prevents gas burning within the gauze from being blown through. The bonnet is provided with a glass window so that an observation of the caps can be made, but its use interferes with the visibility of the smaller and fainter caps, and the gauze as a background is unsuitable to throw up the caps.

The Pieler lamp is an exceedingly delicate means of testing for small proportions of fire-damp, such as would be found in return airways or other parts where it may be desirable to register the varying amount at different points where a cap does not show on an ordinary safety-lamp. Its indiscriminate use, however, as an ordinary testing-lamp in workings is one fraught with considerable danger, especially in inexperienced hands, on account of the large amount of flame within the lamp which results from an atmosphere containing more than 2 per cent. of fire-damp. The alcohol reservoir becomes heated by the combustion of gas within the gauze of the lamp, which may be sufficient, after a time, to boil the alcohol and lead to the production of a large and dangerous flame of burning alcohol vapour. The lamp should, therefore, never be taken into any part of the mine where there is more than 2 or 3 per cent. of fire-damp present. It is a most valuable lamp, to the careful observer, for detecting small quantities of fire-damp up to 2 per cent., but its use should be restricted to strictly safe examinations. Unfortunately, owing to the non-luminous flame produced by the alcohol which is used in it, the lamp is of no use as a guide in the mine, another safety-lamp being requisite for this purpose. Attempts have been made to construct the Pieler lamp with two reservoirs, one to contain oil and the other alcohol. By raising the wick of one reservoir when the other is lowered, it was thought that a luminous oil-flame for lighting purposes or a non-luminous alcohol flame for gas-testing could be commanded at will in the same lamp, and the inconvenience of carrying two lamps in the mine be thus obviated. The use of this composite lamp is attended with practical difficulties, however, which have hitherto prevented its adoption. The Pieler lamp cannot indicate more than 1·75 per cent. of fire-damp by the flame-cap, and the lighting-lamp is used to test for the higher proportions of gas.

Ashworth's Benzoline Lamp.—This is a lamp introduced by Mr. James Ashworth, and is intended to serve the double purpose of lighting and of delicate gas testing. The illuminant used is benzoline, and the form of lamp a modified Hepplewhite-Gray. Benzoline prevents the charring of the wick which occurs when oil is used. The lamp gives a bright light when used for illuminating purposes. A special burner is used, and when the flame is reduced for gas-testing purposes its temperature is higher than an oil-flame of equal size, and this tends to give a more visible cap. A very convenient and gradual rack-movement is used to reduce the light. When the flame is properly adjusted it loses its luminosity almost entirely, and maintains itself more satisfactorily than an oil flame. It is shielded from the eye, during an observation, by a permanent metal screen fixed around the wick-tube, which also serves for setting the flame to uniform height. The height of the screen is such that it just conceals the reduced flame from the eye. The conical lamp-glass is dead-black over two-thirds of its inner surface to completely deaden all reflection when testing for gas. The caps seen through the glass are small, but are clearly visible when there is from 1 to 1½ per cent. of fire-damp present up to 6. Percentages of gas lower than 1 are likely to be confounded with a halo or false cap always visible over the benzoline flame, but it is said that ¼ per cent. of fire-damp can be detected by the lamp.

Professor Clowes gives the following as the cap heights observed in experiments made by him with the Ashworth lamp, having a flame 3 mm. high (see Fig. 571).

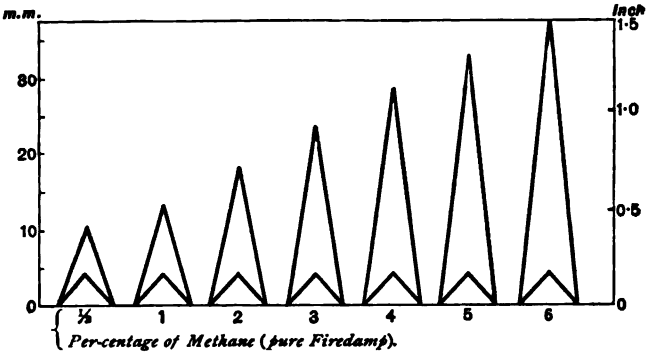

Fig. 571.—Flame and Cap (Benzoline). (Actual size.)

The heights of the test-flame and of flame caps are as follow :—

Percentage of Methane (Fire damp) present.	Ashworth's Benzoline Flame, height 3 mm. or 0·12 inch. Height of Cap.	
	mm.	inches.
0·5*	7*	0·28*
1·0	10	0·40
2·0	14	0·56
3·0	20	0·80
4·0	25	1·00
5·0	30	1·20
6·0	35	1·40

The presence of fire-damp causes the full-size benzoline flame to spire more distinctly than the oil-flame. This test is made by raising the flame until it is on the verge of smoking. If this be done and the lamp raised into a gaseous mixture an elongation of the flame follows. The elongation becomes more pronounced as the percentage of gas present increases, but any lengthened flame at once commences to smoke. This spiring is due to diminished supply of oxygen to the flame, but it cannot be made use of with certainty for detecting the presence of gas.

The Chesneau Lamp.†—This is a lamp invented by M. Chesneau to burn a large alcohol flame for gas testing (see Fig. 572). The reservoir for the alcohol is of brass, formed with a circular crown for the admission of air through double gauzes, on the Fumat pattern. The entry of air below the burner can be regulated; its course is indicated by the arrows. The flame is shielded from the eye by a screen, which is a solid cylinder of sheet iron fixed around the wick-tube and resting on the crown of the reservoir. This screen carries above it an iron-wire gauze $5\frac{1}{2}$ inches high. The gauze cylinder is protected by a sheet-iron shield which has an observing-window of mica with a scale engraved on it. The lower extremity of the shield has an annular diaphragm fitting close around the base of the gauze so as to prevent the surrounding air from reaching the gauze directly. The diaphragm rests on an asbestos washer to prevent, as far as possible, the heating of the lower part of the lamp when it is in a gaseous mixture and the shield gets hot. The openings in the top of the shield have protecting screens to prevent currents from striking the gauze. There is a short outer cylindrical shield of sheet-iron which is movable, and this shield is provided with a window. Its object is to protect, when required, the base of the observation-window in the main shield against currents, and so prevent moisture from forming on the interior of the mica by outside cooling. The window in the outer shield can be brought in front of the mica window when it is desired to make an observation. In a calm atmosphere the condensed moisture disappears some minutes after lighting, and a fresh deposit is not seen until the lamp is taken into a cold current.

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* This indication is very doubtful; it cannot be trusted.

† See "Transactions of the Federated Institute of Mining Engineers," vol. 4, pp. 617, 618.

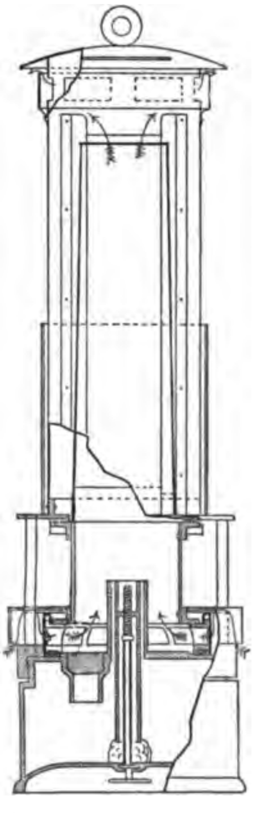

Fig. 572.—The Chesneau Lamp.

A small piece of cotton-wool is placed under the wick-tube, in the spirit reservoir, to prevent the rapid escape of alcohol if the lamp is accidentally overturned. This, however, is quite sufficient, for the lamp is extinguished if laid down horizontally, before the alcohol can spread; but this is not the case with an ordinary Pieler lamp. The hole in the reservoir, through which the alcohol is supplied, is closed by a screw plug, a tight joint being obtained by means of a washer of lead or leather. The weight of the lamp, with the reservoir full of alcohol, is 3·2 lbs.

The isolation of the upper part of the lamp from the reservoir prevents the dangerous heating of the alcohol which occurs in the Pieler lamp when the combustion of gas within the lamp heats the gauze strongly. By means of the screen the regulation size of alcohol flame can be maintained, and with it the lamp gives caps which only reach the top of the gauze with about 3 per cent. of fire-damp. With from 3 to 5·5 per cent. the alcohol flame elongates, the cap enlarges, becomes cylindrical, but lowers more and more from the summit, the products of combustion being given off too rapidly to be freely discharged. A little above 5·75 per cent. of fire-damp causes an elongation of the alcohol flame, which nearly reaches to the top of the gauze, but without appreciably heating it, as the air entering below is sufficient for the complete combustion of the gas and of the vapour caused by the heating of the lamp. Between 5·75 and 6 per cent. of gas causes the alcohol flame to disappear in the lamp, and the gas burns with a very pale flame in the crown only, without unduly heating the reservoir. If the inlet regulator be now closed the burning gas is extinguished in a few seconds. With about 6 per cent. of fire-damp the mixture is explosive and the Chesneau lamp is extinguished.

The height of the caps varies according to the quality of the alcohol used. The same alcohol should be used, therefore, for purposes of comparison. It is stated that the best for this lamp is wood spirit or methylated alcohol of 92·5 degs., by the Gay-Lussac alcohol-meter at 59 degs. Fahr.

One of the chief advantages claimed for the lamp is the introduction of a copper-salt, which is dissolved in the alcohol with a little hydrochloric acid to maintain it in solution. The flame and the caps are then coloured, and so rendered more easily visible. The salt at first used was copper chloride, but both it and the acid used with it, corrode the reservoir and wick-tube and lead to accretions of cuprous chloride, which soon choke the wick. Since this experience copper nitrate, together with some ethylene chloride, have been dissolved in the alcohol instead of the copper chloride. It is said that this renders caps visible when only from 0·1 to 0·2 per cent. of fire-damp is present.

It is necessary to carry another lamp in the mine with the Chesneau for lighting purposes and for reading the scale of the gas-testing lamp.

*Stokes' Alcohol-flame Safety Testing-Lamp.**—This safety-lamp is a modification of the Gray type, supplemented by appliances designed by Mr. A. H. Stokes, H.M. Inspector of Mines, for the detection of small percentages of inflammable gas.

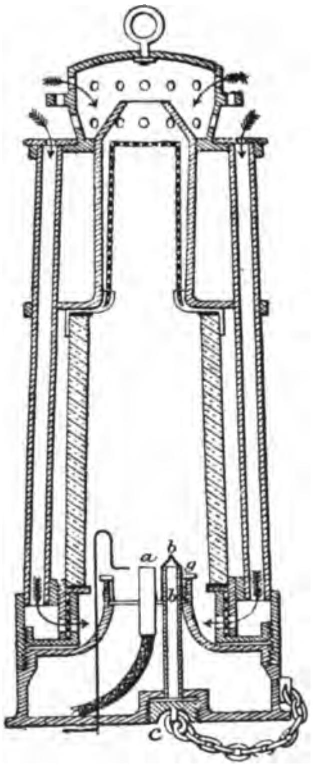

Fig. 573.—Stokes' Alcohol Safety-lamp.

The oil-chamber *d*, Fig. 574, is pierced by a small tube *b*, parallel and similar to the pricker-tube but of larger diameter. Its position is immediately behind the wick-holder of the oil flame and its upper end level with the top of the wick-tube *a*. The lower end of the tube *b* terminates in a circular recess, the top of which is sufficiently above the bottom of the oil reservoir to admit the screw-plug *c* when the tester is not in use. In Fig. 573, the screw-plug is shown screwed home, whilst in Fig. 574 it is removed to make way for the tester. The upper portion of the tube *b*, or that within the lamp-glass, is protected by a spring cap or extinguisher *g* so constructed that it opens for the tube of the alcohol tester to pass through and closes upon its withdrawal. In the act of withdrawing the tester the mouth of the tube *b* is firmly closed before the testing vessel is quite unscrewed from the thread of the plug recess. This prevents the passage of flame down the tube *b*, even if the lamp-user omitted to replace the screw plug *c* immediately after making a low percentage test; nevertheless, it is a further precaution to have the screw-plug permanently attached to the lamp by means of a short chain as shown on the drawing. Any forgetfulness to replace the screw-plug is brought under the notice of the operator by its dangling below the lamp bottom.

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* See "Transactions of the Federated Institute of Mining Engineers," vol. 5, pp. 462—473.

The alcohol tester, Figs. 575 and 576, consists of a small cylindrical brass vessel for holding the alcohol, with a long wick-tube *i* screwed into the top of the vessel. This tube is connected to the vessel by means of a circular brass plate, on the under side of which is placed a leather washer to ensure a perfect fit. The wick-tube is only 4 millimetres or 0·16 inch in diameter, and 55 millimetres or 2·2 inches long. The wick used in it may consist of two strands taken from an ordinary round oil-lamp wick. A cap or covering *f*, Fig. 575, is provided to go over the wick-tube *i*, so as to protect it from injury and from dirt when not in use. This cap screws nearly but not quite home to the alcohol vessel, and pressing upon the leather washer fitted to its under side, securely covers the air-hole. This leather washer and the one on the under side of the brass circular plate effectually seal the openings in the alcohol vessel, and allow of its being carried in the pocket without fear of leakage.

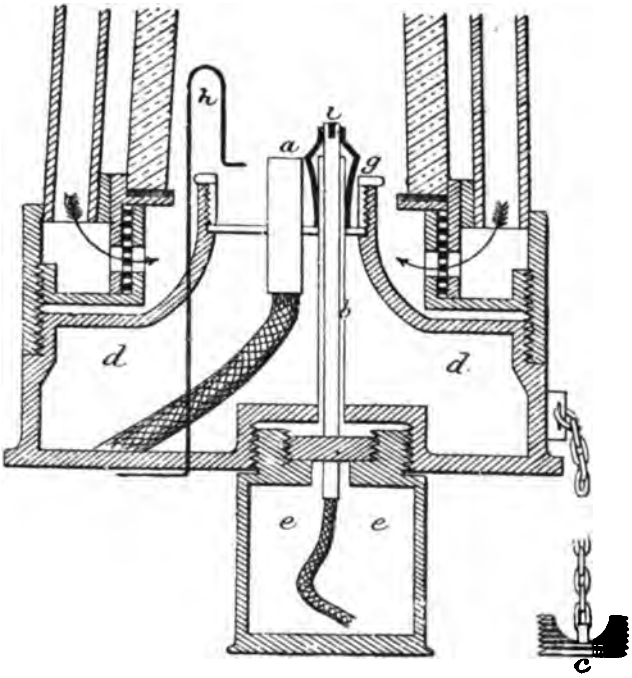

Fig. 574.—Stokes' Alcohol Safety-lamp.

The tester is charged with absolute alcohol before being taken into the mine; if filled and then burned continuously, the charge will last for about four hours. One charge, therefore, allows of 120 tests, of two minutes' duration each, before refilling is necessary. The wick of the tester is saturated with alcohol, after which it is drawn through and cut level with the top of the brass tube *i*.

Beside the alteration in the construction of the Gray safety-lamp already indicated, the glass is made considerably longer than usual, in order that the gas caps may be measured. Its diameter is, however, less than that forming part of an ordinary Clanny type of lamp, otherwise the large space for gas within the lamp might be dangerous when trying for high percentages in explosive mixtures. The whole length of the glass, for a breadth of 1 inch, is enamelled jet-black to form a background or screen, and thus render visible the cap due to small percentages of gas. Even when this background is held behind the light at the time of making a test observation, as it always should be, it is difficult, if not impossible, to see the cap formed by so low a percentage of gas as 0·5, and it has been found advantageous, before making delicate tests, to smoke the inside of the lamp-glass with a lighted taper for the full width of the black band so as to get a more dead black than the enamel. This should be done of course before going into the mine.

It is intended that this lamp should first be used to test for gas by means of the oil-flame in the usual way, and that the alcohol-flame should be used only if the other fails; in other words, search with the alcohol-flame begins where the oil-flame finishes.

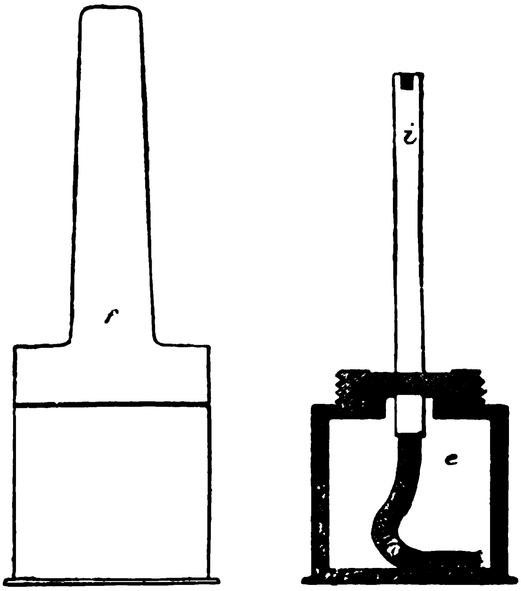

Fig. 575. Fig. 576.

STOKES' ALCOHOL SAFETY-LAMP.

The mine official, about to test, should first try with the full oil-flame, then with the same flame reduced in size. If this fails to indicate the presence of gas, he should restore the oil-flame to its normal height. The screw-plug *c* is then withdrawn from the bottom of the lamp and the alcohol-tester prepared by unscrewing the cap *f*. The tube *i* of the tester is now inserted through the bottom of the oil-chamber and screwed tightly up the full length of the screw. In passing upward the alcohol wick-tube *i* will open the spring cap *g* and appear above it. In a few seconds the heat of the oil-flame will cause the alcohol to ascend the wick-tube and become ignited, or to expedite the ignition, the lamp may be held slightly on one side, so that the oil-flame touches the top of the alcohol wick-tube. When the alcohol-flame has reached its standard height, as indicated by the slit *i'* across the tube, Fig. 574, the oil-wick is drawn down into its tube by the pricker and the light extinguished. There is then only the small pale blue flame of the alcohol-tester burning inside the lamp, by means of which the test is then made. If no gas cap appear above the flame, the atmosphere surrounding the lamp may be considered as containing below the least percentage of gas the lamp is capable of indicating. This is claimed as 0·5 by the inventor. If, on the other hand, a gas cap appear, its intensity of colour and length give the percentage of gas contained in the atmosphere undergoing examination.

To facilitate the test a small strip of black cardboard is placed round the front of the glass, of such a depth as to hide from view the whole of the alcohol flame, and long enough to reach across between two standards of the lamp between which and the glass it is held, or the palm of the hand may be used to hide the flames from sight.

On the completion of the test the result is noted in the pocket-book, the oil-wick is raised and bent over by the pricker until it touches the alcohol-flame, by which it is re-lighted. After adjusting the oil-flame to its proper light-giving height, the alcohol vessel is unscrewed. The withdrawal of the alcohol wick-tube *i* releases the automatic spring-extinguisher *g*, which immediately closes the opening of the insertion-tube *b* inside the lamp. The tester is withdrawn, and when the screw-plug *c* is inserted in its recess the lamp assumes its normal form and becomes an ordinary safety-lamp with oil-flame. The cover of the alcohol wick-tube *f* is screwed on and the apparatus returned to the pocket until required for further use, its weight when fully charged being about 4½ oz.

In the case of air-currents containing 3 or more per cent. of inflammable gas, the alcohol-flame is unnecessary, inasmuch as that proportion is clearly visible in the best forms of modern oil gas-testing safety-lamps.

A person accustomed to test for gas in a mine will have little difficulty in using the alcohol-tester, but he will require some training in this particular kind of observation, for the cap is very pale in colour in low percentages of fire-damp, and the thick circular lamp-glass through which the caps are viewed is detrimental to clear observation. Beside its curved surface a film, due to the humidity of the atmosphere of the mine, settles on the inside of the glass after the lamp oil-flame has burned a short time.

Mr. Stokes has made very careful tests of the lamp in ordinary coal-gas on the surface, and gives on his own authority the following results and conclusions:—

Alcohol-flame, with a Standard Height of 13 Millimetres (0·52 Inch).

Percentages of Gas.	Height of Standard Alcohol flame.	Height of Gas Cap.		Remarks.
	millimetres.	millimetres	inches.	
0·5	13	15	0·60	Very pale, not clearly seen.
1·0	„	25	1·00	Pale blue colour, can be clearly seen.
1·5	„	36	1·44	Pale blue colour, clearly defined.
2·0	„	42	1·68	Clear blue-coloured spiral.
2·5	„	50	2·00	Distinctly defined gas cap.
3·0	Oil flame.			

The top of the cap is, in all cases, so thin and attenuated that its termination for measurement requires careful observation.

In conducting tests on the surface it is absolutely necessary that they should be made in a perfectly dark room, equal to that which would be found in a mine.

With 0·5 per cent. of gas the cap is very pale blue in colour, conical in form, and can only be observed by carefully shading the light of the alcohol-flame. It is with difficulty measured, owing to the top part of the spiral cap being highly attenuated and pale.

With 1·0 per cent. of gas the cap is pale blue in colour, it can be clearly seen, and shows a defined spiral shape, which can be measured by shading the light of the testing flame.

With 1·5 per cent. of gas the cap is pale blue, but a little deeper in colour than with lower percentages, and is clearly defined.

With 2·0 per cent. of gas the cap is a clear blue-coloured spiral of the well-known description of cap due to inflammable gas.

With 2·5 per cent. of gas the cap is similar in colour to the cap obtained in the previous test, but rather more distinct and well defined.

CONCLUSIONS.

1. The lamp is an official's ordinary safety-lamp, and therefore can be used for light, and for the daily work of examining and testing the mine.

2. The alcohol tester is simple, easily applied, and can be carried in the waistcoat pocket.

3. The alcohol tester should not be used as a primary light for finding gas in goaf or in holes, but for detecting low percentages of gas after the ordinary oil-flame has failed to indicate its presence.

4. When making a test the flame of the alcohol-burner should be shaded from sight by a piece of cardboard or the palm of the hand. The flame can be adjusted to its standard height by the screw of the tester.

5. The colour and length of the cap should be noted, and unless the observer is an expert in testing the notes made in the mine should be compared with the copv of standard flame-tests for determining the percentage of gas in the atmosphere.

6. The standard size of alcohol-flame is not immediately obtainable by introducing the tester when the lamp is cold; but after the lamp has been burning some little time the oil-vessel becomes warm, due to the heat from the oil-flame. If the alcohol tester be then introduced it gives in a few seconds the standard height of 13 millimetres or 0·52 inch.

7. Cleanliness is very necessary for the successful working of the tester, as oil or grease upon the tube gives a yellow tinge to the alcohol-flame until burnt away. The vessel should be emptied after use and charged with a new wick and fresh alcohol when again required.

The Stokes' safety-lamp, with alcoholic tester, is supplied by Messrs. John Davis & Son, of Derby. The cost of the alcohol necessary to charge the tester and make 120 tests of two minutes' duration is about 3d. A decidedly inferior result is given by using methylated spirit instead of absolute alcohol, the spirit burning with a slightly yellow flame.

The lamp will principally be of service for the daily or more frequent testing of the return air-currents. When the deputy or fireman has finished his examination of the working-places, he should test the return air-current from his district or districts, the primary test being with the oil-flame.

Although it is not contended that the accuracy obtained by the analytical chemist in ascertaining the precise composition of the atmosphere, can be obtained by the use of this lamp, yet in competent hands in periodic inspection rounds, it will give closely approximate results, and serve as a practical means of ascertaining the varying rate at which gas issues from time to time.

The Gray safety-lamp is specially designed and arranged as a gas-tester, the standards or poles being hollow tubes, the inlet feed can be taken from the top of the lamp, so that the test can be made in positions as high as the top of the lamp can be placed.

*The Hydrogen Gas-testing Lamp.**—(Clowes', Redwood's and Waters' patent).

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* See "Trans. of Federated Institution of Mining Engineers," vol. 4, p. 441; vol. 5,

This is a suitable safety-lamp, furnished with special fittings to receive an appliance for delicate and accurate gas-testing. The appliance and its adaptation were designed by Professor Clowes and other gentlemen associated with him.

The form of safety-lamp employed is open to selection, but Professor Clowes prefers the Ashworth-Hepplewhite-Gray lamp, because he considers it to be one of the best existing forms of gas-testing safety-lamps.

The invention is largely the outcome of very careful experiments undertaken in 1891, by Professor Clowes, with the view of ascertaining the delicacy and accuracy of gas-testing safety-lamps then existing, and of deciding which is the most suitable test-flame for the purpose.

Professor Clowes first designed a special form of apparatus, or "test-chamber," which is particularly suitable for the observation of the caps which appear over different test-flames. This apparatus will be fully described later on. By the aid of this "test-chamber," he was enabled to observe and measure with accuracy the caps which were produced over the ordinary flames of oil, benzoline, and alcohol, in some of the lamps already referred to, when they were exposed to mixtures of air with fire-damp. Similar observations were afterwards made upon the caps seen over a naked hydrogen-flame. The result of the latter series of observations is shown in Fig. 577. A comparison of the effects produced convinced Professor Clowes that the hydrogen-flame was superior to all other flames for gas-testing purposes. Its advantages are the following:—(1) Owing to its non-luminosity, the flame never interferes with the perception of the caps, and the observation of the caps can, therefore, be made without shielding the hydrogen-flame from the eye. (2) It is the only safety-lamp flame which is fed without

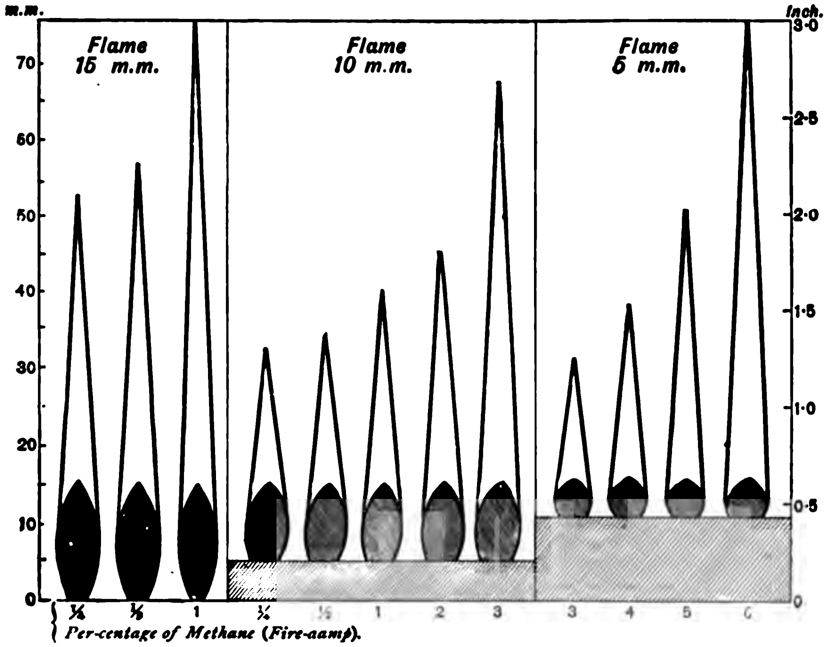

Fig. 577.—Actual Height of Caps over Hydrogen Flame

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pp. 265, 367, 375; vol. 6, p. 177, and vol. 7, p. 2. Also Journ. Soc. Arts, Feb. 17, 1893; and Cantor Lectures, Soc. Arts, Jan., Feb., 1894; Proc. Roy. Soc., vol. 52, p. 484.

a wick, and the gas supply can, therefore, be at once regulated with certainty and precision, so as to produce and maintain a flame of standard dimensions; this is impossible with wick-fed flames. (3) The hydrogen-flame differs from all other testing-flames by being clearly defined and entirely free from haze or halo at its tip; it can, therefore, be brought with great nicety to a scale-mark, and thus adjusted to an invariable standard height. (4) It gives larger and more visible caps than any other flame of equal dimensions, and in the presence of large percentages of gas it does not become luminous as the alcohol and oil-flames do. (5) The hydrogen-flame cannot be blown out by a strong air-current sufficient to extinguish all other flames; it is, therefore, able in a safety-lamp to resist rapid movements of the air, such as those caused by swinging the lamp against the ventilation current, by any rapid movement of the lamp, or by large stones falling from the roof close to the lamp. (6) The hydrogen-flame also requires for its extinction the presence of a proportion of carbonic acid gas four times as large as that which extinguishes other lamp-flames. (7) The hydrogen-flame is at once extinguished by the presence in the air of an amount of fire-damp which would give rise to explosion if it were kindled.

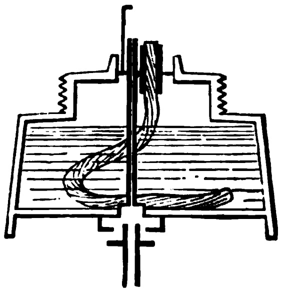


Fig. 578.—Hydrogen-Oil Lamp. (Section.)

In 1892, Professor Clowes designed a portable apparatus for gas-testing with the hydrogen-flame in the mine. This consisted of a small steel cylinder in which was stored sufficient compressed hydrogen-gas to feed the standard flame continuously for 16 hours. This cylinder could be slung from the shoulder and connected at will by means of flexible tubing with an ordinary oil safety-lamp, provided with a suitable jet for burning the gas, see Fig. 578. The tube for the passage of the hydrogen through the oil-vessel of the lamp was of very fine bore; it was arranged with the jet on a level with the top of the wick, and so near to it that the hydrogen-gas when turned on was kindled by the oil-flame without the necessity of opening the lamp.

It was found on actual trial that so large a supply of hydrogen was unnecessary, and the flexible tube was inconvenient in use; consequently, in 1893, the lamp was somewhat modified. The connection with the hydrogen-jet which had been made at the bottom of the lamp was now more conveniently placed at the side, and a smaller and lighter form of cylinder was introduced, which could be carried in the pocket, and attached directly to the lamp itself when the hydrogen-flame was required. Several other improvements have since been introduced, and the lamp has now reached a high state of perfection.

Figs. 579 and 580 show the gas-testing and gas-measuring appliance attached to the lamp. The chief feature of the invention is a cylinder containing compressed hydrogen, which can be instantly attached to the lamp above and below by a quarter-turn, or by a screw-union, after which it forms a rigid handle at the side of the lamp. In this position it is shown in Figs. 579 and 580, the latter being partly a sectional view of the lamp and cylinder. In the alternative method of attachment the cylinder drops into two claws on the upper part of the lamp above, and is then in position for screwing up a collar below (Fig. 579). This method is not so rapid and simple in use as the first described, but has the advantage over it of preserving a gas-tight joint after much wear. The segmental band of metal which projects from the lamp to receive and retain in its place the projection formed near the top of the cylinder, is slotted, and may be moved sufficiently, by means of screws, round the lamp to take up the ordinary wear of the oil-vessel screw. The cylinders are all interchangeable, and can be attached to any lamp.

The gas connection with the lamp is at the side near the bottom of the lamp, and a detachable key is used to regulate the entry of gas into the lamp from the cylinder. A fine copper tube passes from the opening in the side of the lamp, terminating in a jet beside the wick. The bore of this tube is too small to pass flame even under high pressure, and is, therefore, safe in use without being closed. There is no danger then in carrying the lamp into an inflammable atmosphere with the tube open. The hydrogen passes along the tube when a low percentage test is to be made, and is kindled as it impinges against the oil-flame. There is no visible evidence that the kindling has taken place, except that obtained by watching the oil-flame, from which a sharp tongue of flame projects upwards when the stream of hydrogen ignites. The luminous oil-flame quite conceals the hydrogen-flame from view, when both are burning at the same time. The flow of hydrogen is controlled by the key which opens the cylinder valve. This valve is of the simplest construction possible, and is closed by a screw which cannot become deranged. To guard against the possibility of an unduly large flame, and consequent injury to the lamp top, a check-pin has also sometimes been put on the tap, but this is quite unnecessary. The valve is so constructed as to open very gradually, and the key is made sufficiently large to be under perfect control. The careless enlargement of the hydrogen flame until it plays upon the top gauze in no way damages good steel wire gauze.

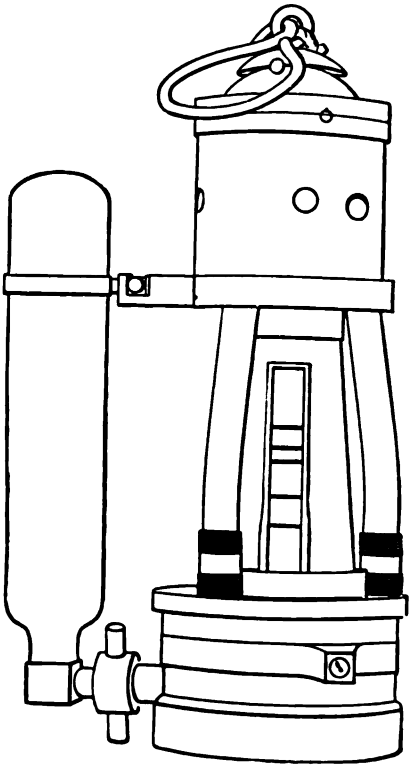


Fig. 579.—Hydrogen-Oil Testing-Lamp.

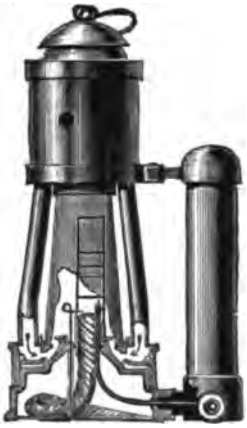


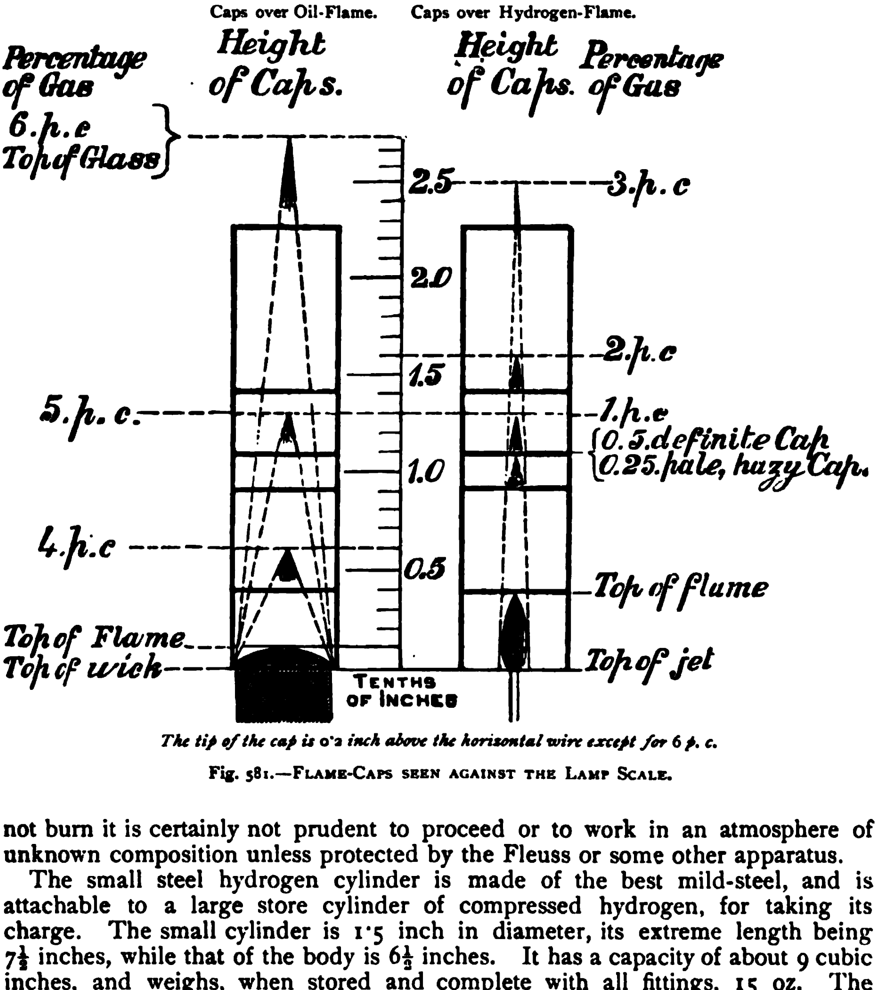
Fig. 580.—Hydrogen-Oil Testing-Lamp.

As soon as the hydrogen-gas is seen to be ignited the oil-flame is extinguished by pulling down the wick into its tube with the pricker, and the hydrogen-flame is then adjusted to the standard height by regulating the cylinder-valve, until the tip of the flame is level with the first bar on the brass scale, which is 10 mm. or 0·4 inch above the hydrogen-jet. This scale, as shown in the drawings, is ladder-like in shape, and may be fixed by a simple screw fastening upon the top of the oil-vessel, and thus kept in an erect position in front of, and to avoid errors, close to, the flame. It can be seen through the lamp-glass, and serves to measure cap heights corresponding to different percentages of fire-damp, from 0·25 to 6, as well as for the adjustment of the hydrogen-flame. Black horizontal lines corresponding in level to those on the brass scale may also be burnt across the clear portion of the lamp-glass so as to form a scale there. If this is used without the brass scale slight errors creep into observations made when the eye is raised above or lowered below the proper level. It is better at first to use both scales. The lamp can then be held so that a particular divisional line on the glass is seen exactly on a level with the corresponding divisional line on the brass scale. Either the position of the lamp or of the eye should be moved to ensure this at any observation. The observer is then certain that his eye is truly level with the line on the brass scale, and that no error is introduced by lack of proper adjustment of the eye.

The tip of the cap in every case rises 5 millimetres or 0·2 inch above the bar of the scale, except for 6 per cent., so as to make the bar visible as an opaque object against the cap as a background. Percentages of fire-damp in the air are ascertained by reference to a small pocket scale-card, similar to that shown in Fig. 581, but it is found that when the lamp is in constant use the observer requires neither the card nor the scale. In experimental trials the hydrogen flame, when set in air free from gas, did not appreciably increase in size until more than 3 per cent. of marsh gas was present in the air. The height of the hydrogen flame may therefore be regulated in the presence of the gas, and it is not necessary, as might have been supposed, to regulate it in pure air. The standard flame is set, and the cap is measured at the same time at the testing place. After the cap-height has been read, the lamp-wick is pushed up again by means of the pricker, and on reaching the hydrogen flame is immediately lighted. The hydrogen is then shut off, the cylinder is detached and pocketed, and the lamp becomes an ordinary safety-lamp with oil-flame. The time occupied in a single test need not exceed 30 seconds.

If the cylinder remains attached to the lighted lamp for a long time it becomes warm, but the rise in temperature is not sufficient to seriously increase the pressure of the hydrogen, and is not greater than that caused by the compression of the gas in charging the cylinder. The cylinder should be used for hydrogen only. If previously charged with another gas, the residue of this gas in the supply from the cylinder affects the nature of the hydrogen flame. When fully charged with hydrogen the cylinders have been severely tested by hammering until distorted in shape, and by being thrown violently down upon hard surfaces, without any leakage or bursting, so that there appears to be no danger from their being roughly used or accidentally dropped.

The great value of the hydrogen flame as a means of testing is due partly to the facts that it is the hottest flame known, and is also the only non-luminous flame. The size of a flame-cap depends on the intensity of the heat of the flame, and the cap is more easily seen as the flame is less luminous. A large and hot colza flame probably produces caps in low percentages of fire-damp, but they are invisible owing to the luminosity of the flame. The hydrogen flame also possesses the advantage that it does not become extinguished in air currents of great velocity which may occur in the mine, or to which the lamp may be exposed by being swung or dropped. These blow out the flame of oil with ease, and still more readily the flame of alcohol. In passing through narrow or awkward places, where there is liability to swing or drop the lamp, the hydrogen flame may be burnt along with the oil flame; if this last be extinguished the hydrogen will survive and re-kindle it. The hydrogen flame will also continue to burn in air containing a proportion of carbonic acid gas nearly four times as great as that which extinguishes flame of oil and of alcohol. Hence, if the hydrogen flame is burnt as well as the oil flame in places where foul air might be encountered, the "loss of the flame" is prevented, since on retreating when the oil flame is extinguished by the foul air, the hydrogen flame survives and re-kindles the oil. Of course it is not generally desirable to use the hydrogen flame in passing through foul air spaces in which the atmosphere may be irrespirable. If an ordinary oil flame will not burn it is certainly not prudent to proceed or to work in an atmosphere of unknown composition unless protected by the Fleuss or some other apparatus.



The tip of the cap is 0·2 inch above the horizontal wire except for 6 p. c.

Fig. 581.—Flame-Caps seen against the Lamp Scale.

The small steel hydrogen cylinder is made of the best mild-steel, and is attachable to a large store cylinder of compressed hydrogen, for taking its charge. The small cylinder is 1·5 inch in diameter, its extreme length being 7½ inches, while that of the body is 6½ inches. It has a capacity of about 9 cubic inches, and weighs, when stored and complete with all fittings, 15 oz. The cylinders are tested to 4,000 lbs. hydraulic pressure per square inch, their bursting pressure being 7,000 lbs. per square inch. The greatest pressure they are subjected to when fully charged is 1,500 lbs. per square inch. The completely charged cylinder will feed the standard hydrogen flame continuously for about two hours. The store of hydrogen in a small cylinder is sufficient for about 200 tests, and will serve for more than one complete inspection of a mine of ordinary size, inasmuch as the hydrogen flame will only be used at such times as a reduced oil flame does not indicate the presence of gas. The cylinders retain their store of gas without leakage or loss for an indefinite length of time. The tests made when the cylinder is nearly discharged are quite as effective as those made when it is just replenished, but in this case the standard flame drops rapidly after it has been set.

The oil-chamber is supplied with a mixture of colza oil and water-white paraffin, the paraffin being introduced to prevent charring and crusting of the wick, while the admixture is safe in use and gives a better light than colza oil alone.

The weight of the brass lamp with oil and all fittings, except the hydrogen cylinder, is nearly 4 lbs.; the simpler form of brass lamp weighs 3½ lbs., and the hydrogen cylinder about 14 ounces. The lamp fully charged weighs 1 lb. 11 oz. when made in aluminium, and the hydrogen cylinder with aluminium fittings weighs 12 ounces.

The lamp is not intended for common use in the mine by the working miner, but only for those whose duty or desire it is to make examinations for gas. It may be most usefully applied to ascertain the state of the return airways, and for the regular examination of the working places before each shift begins work. A few of these lamps at a colliery will probably be found to be sufficient. For detecting percentages of gas above 3 and up to 6 the reduced oil flame is recommended. Higher measurements than 6 per cent. are unnecessary, for from 6 to 7 per cent. of gas becomes explosive and dangerous. For testing and measuring percentages below 3 down to 0·25, or even to 0·1 per cent. when requisite, the hydrogen flame is used by attaching the hydrogen supply and changing from the oil to the hydrogen flame. The hydrogen flame is pale, so that it requires no screening from the eye, and in fact assists the eye in detecting the cap. Even to a colour-blind observer it is quite distinct from the cap, but in order that the latter may be seen more clearly, a dead-black permanent surface has been produced upon the interior of the back of the glass behind the flame. A standard of 10 millimeters (0·4 inch) is used for measurement of gas from 0·25 to 3 per cent.; if higher percentages are to be indicated a standard hydrogen flame of half that height is employed. Gas may be detected and measured when it is present in proportions from 0·1 right up to 6 per cent., by means of the hydrogen flame alone, if the flame is reduced to 5 mm. or 0·2 inch in height for percentages from 3 to 6 per cent.; the indications for the lower percentages may also be increased, if desired, by using a flame 15 mm. or 0·6 inch in height (see Fig. 577); this increase is, however, unnecessary, since the caps over the standard flame, with even 0·1 per cent. of gas and upwards, are clearly seen by even an inexperienced observer.

Since the lamp was first used it has been noticed that the reddish-purple tinge of the hydrogen flame changes to pale blue in the presence of carbonic acid gas in the air. The change of colour is distinctly perceptible when only 2 per cent. of this gas is present, and it becomes more and more pronounced as the proportion increases. The flame is, therefore, a much more delicate indicator of carbonic acid, or " choke-damp " than the ordinary oil flame; the oil flame begins to diminish in size only when 10 per cent. of this gas is present, and is extinguished by 15 per cent.

This lamp, therefore, fitted with the hydrogen gas-testing apparatus, is not only used for ordinary illumination in the mine, but may be employed without any other apparatus to ascertain the presence of low percentages of gas in the air of the mine. The Pieler lamp will measure percentages of gas varying from 0·25 to about 2, but when it is used it is carried in the mine as an extra lamp beside that used for illumination.

The Ashworth-Hepplewhite-Gray safety-lamp flame is scarcely affected in air-currents of great velocity, and the indications made by the hydrogen lamp in high-speed ventilating currents are as trustworthy as those made in still air. No tests whatever are trustworthy if the air is very abundantly charged with fine particles of coal dust, the pale cap then becoming invisible owing to the increased luminosity of the hydrogen flame. But in the ordinary condition of the mine, gas tests can be made, at regular intervals of time, in the main returns, the returns of each district and in other places, since the amount of dust in the air is quite insufficient to affect the flame.

The caps are all smaller than those shown by the Pieler lamp, a comparison of height in the two lamps, given by Professor Clowes as the result of his experiments (with a 10 mm. standard flame in the hydrogen lamp) up to the 2 per cent. indicated by the Pieler, being as follows:—

Percentage of Fire-damp.	Round Hydrogen-flame H.		Round Pieler Alcohol-flame P.	
	Millimetres.	Inches.	Millimetres.	Inches.
¼	17	·67	30	1·18
½	18	·71	55	2·20
1	22	·87	90	3·54
2	31	1·22	140 cap reaches lamp top.	5·51

But the hydrogen flame caps are quite as easily seen and measured as those over the Pieler flame, and have the advantage of being produced and observed in the ordinary lamp.

Although the difference between ·67 and ·71 inch, representing ¼ and ½ per cent. of gas respectively in the hydrogen lamp, is slight and inappreciable, no difficulty arises in distinguishing the caps by their appearance, the ¼ per cent. giving a pale, hazy flame, with ill-defined outline as compared with the ½ per cent. cap. The difference in the length of the caps is less and the rate of increase less in the hydrogen safety-lamp than in the Pieler, but the latter, seen through the metallic gauze, are faint and not so plainly visible as those viewed through the transparent glass against the black background on the screen in the hydrogen-lamp, and all the hydrogen flame caps are clearly seen and easily measured, even by an inexperienced observer. The taller caps of the Pieler flame could not be observed in an ordinary lamp. The caps become more visible and definite as well as larger, in the hydrogen lamp, as the percentage of gas increases, but even in the low percentages of gas ranging from ¼ to 2 the hydrogen lamp compares favourably with the Pieler for delicacy in indicating the presence of fire-damp in the air, and its standard flame is not variable in height as is that of the Pieler lamp. When tried in a return airway passing 145,000 cubic feet of air per minute, the cap over the hydrogen flame indicated the presence of ¼ per cent. of fire-damp. Tests taken side by side in different parts of this airway with the hydrogen safety-lamp, the Pieler lamp, and the Liveing indicator, gave absolutely identical results in percentages of gas ranging from ¼ to 2.

The hydrogen lamp has been selected for use by eminent scientific and practical mining authorities in cases of the greatest importance, and its indications are acknowledged by them to be thoroughly practical and trustworthy, and to proceed with ease down to 0·25 per cent. of gas. It is perfectly easy to detect 0·1 per cent. if necessary.

The Clowes gas-detector is made and supplied by Messrs. W. J. Fraser & Co., 98, Commercial Road East, London, E., and the small cylinders are re-charged by the same firm.

We have now three gas-testing lamps of varying degrees of accuracy and delicacy, but all of them more delicate than the oil-lamp, viz., Ashworth's benzoline-flame, Stokes' alcohol-flame, and Clowes' hydrogen-flame lamps. There is, therefore, no excuse for the slovenliness and inaccuracy permissible in former days, when any proportion of fire-damp under about 3 per cent. present in the workings was unnoticed by the fireman, who reported "no gas present." Better means of gas-testing are now available and ought to be rigidly enforced, for the constant use of a delicate fire-damp detector will surely tend to safety in the mine.

The hydrogen lamp, as used in the coal mine, may be employed to test mixtures of air with any inflammable gas, such as coal-gas or water-gas, and it is also capable of testing for petroleum vapour in the air. Before the lamp is taken into such air, however, it must be known that the proportion of vapour is too small to render the respiration of the air dangerous. The manner of procedure is then the same as in the coal mine. These conditions, however, do not prevail in the oil-tanks of steamers, and Mr. Boverton Redwood has designed an apparatus in which the hydrogen-flame is introduced to test samples of air charged with petroleum vapour previously collected from such sources. The apparatus used for collecting samples and testing them are of special construction (see "The Transport of Petroleum in Bulk," by Boverton Redwood, Proc. Inst. Civil Engineers, Vol. CXVI., Session 1893-4, Part II.). All observations with the hydrogen flame, when testing, must be made in a room from which *day*-light is excluded, or else the light must be shut out by a black cloth, as is done when a camera is used. The presence of light from a gas-flame, or a candle or oil-flame, interferes but slightly with the perceptions of the hydrogen flame-caps.

THE TEST-CHAMBER FOR OBSERVING FLAME-CAPS.—This, as already mentioned, was devised by Professor Clowes in the early stage of his investigations, and has been used by him with remarkable success to mix certain proportions of inflammable gas or vapour with air, the mixtures being retained in the apparatus during the observation and measurement of the caps over the testing-flames. Each mixture can then be rapidly removed and replaced by a fresh one. The test-chamber consists of a gas-tight wooden box of 100 litres capacity, supported on four legs to raise it to a convenient height for observation above a bench or table. It is provided with large openings above and below, which can be closed gas-tight by means of water-sealed covers; and a portion of the front woodwork is removed and replaced by glass for observation of the caps, see Figs. 582, 583, and 584. An inlet is formed near the top of the box for the introduction of the measured volume of the light inflammable gas; a means of escape for the heavier air displaced by this gas is provided at the bottom of the chamber (Fig. 583).

The gas is stored in an ordinary glass gas-holder (Fig. 582), which is connected with the chamber, for the supply of a measured quantity of gas. The precise volume of gas required is obtained by pouring into the top of the gas-holder an equal volume of water. This water flows into the lower part of the holder, and, as it rises, expels the gas into the chamber, there to be thoroughly mingled with the air by swinging to and fro a light wooden flap, shown in dotted lines in Figs. 583 and 584. Motion is given to this flap by means of a handle projecting from the front of the chamber, shown on the drawings. It takes from 2 to 3 minutes thus to prepare a mixture of the exact proportions of air and gas desired. The lamp to be examined is passed up through the lower opening to its position in the chamber, where it is afterwards observed through the window in front. The daylight is excluded from the room while observations are made, and the eye of the observer should not have been exposed to daylight or to any other *powerful* light for at least 15 to 20 minutes prior to the observations if faint flame-caps are to be seen. The light of ordinary flames of gas, lamp, or candle does not seriously interfere with the sensitiveness of the eye, and may be present in the room if their light is excluded by means of an ordinary black camera-cloth while the observations are being made.

The gas used by Professor Clowes was chemically prepared methane, or marsh gas, as this is the inflammable gas actually present in fire-damp. The results obtained were confirmed by repeating the experiments with natural fire-damp of known composition. The indications furnished by coal-gas are nearly the same as those yielded by marsh-gas, but coal-gas is not admissible for use in accurate observations, being itself variable in composition.

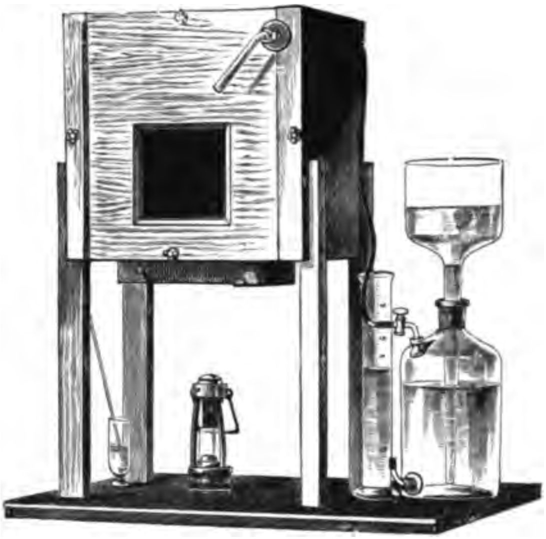


Fig. 582.—The Test Chamber.

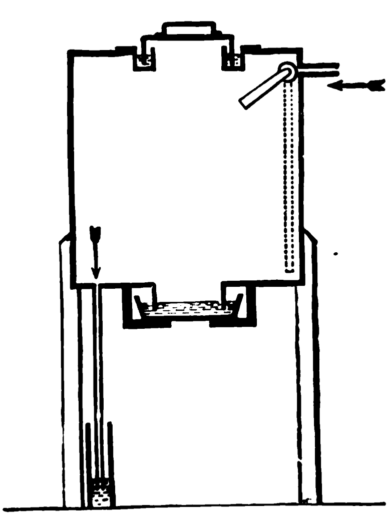


Fig. 583.—Test Chamber (Section—Front).

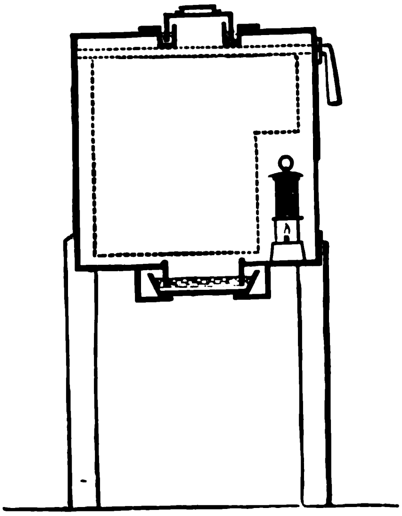


Fig. 584.—Test Chamber (Section—Side).

The precise measurement of flame-caps could not be obtained by placing a scale behind the cap because the light of the pale cap did not illuminate a scale sufficiently to enable a reading to be made. Professor Clowes, therefore, used an ordinary sliding parallel ruler which was pressed upon a strip of paper against the glass window of the chamber. The edges of the rule were primed with luminous paint to make them visible. The rule was opened with one edge level with the top of the test flame and the other extended to the top of the cap. The space included between the two edges of the rule was marked upon the paper strip by means of a sharp pencil and was afterwards measured against a scale graduated into tenths of an inch or into millimetres. Allowance was then made for the distance of the paper strip from the caps and thus the true cap-height was obtained.

After the completion of an observation, the top and bottom apertures in the test-chamber are opened: the gaseous mixture then escapes completely and is replaced by fresh air in 2 minutes, but even this time can be reduced by swinging the flap in the chamber while the apertures are open.

The opening of the lower aperture only for 5 minutes at a time, or occasionally for the insertion and removal of safety-lamps, does not alter, to any perceptible degree, the composition of the gaseous mixture in the chamber. The mixture is also unaffected by the burning in the chamber for at least 20 minutes of the small flame usually employed in gas-testing, but the large alcohol-flame of the Pieler lamp rapidly changes its composition.

The chamber is constructed to enclose exactly 100 litres of air. The preparation of mixtures of air with known percentages of gas is thus a very simple matter. One litre of gas when transferred to the chamber gives a 1 per cent. mixture, two litres 2 per cent., and so on.

The chamber is blackened inside, and is then brushed over inside and out with melted paraffin wax to ensure its being gas-tight. The water-seals to the upper and lower openings secure air-tight closure; the lower tray swings on bars from the bottom of the chamber. A small circular opening is cut in the front of the floor of the chamber, which, when in use, is covered by the safety-lamp over it, and when not in use is closed by a suitable flap. The observer is enabled by this means to adjust the wick, by making use of the pricker, when the reduced oil-flame is being used in the chamber: this is necessary because the flame has a tendency to drop gradually and to become extinguished.

In starting the experiments it is necessary before the connecting tube is secured to the chamber to let a little gas pass from the gas-holder in order to replace the air in the connecting india-rubber tube with gas. When the gas issues from the upper end of this tube the water-pressure in the gas-holder is shut off, so as to leave the gas under atmospheric pressure only. The rubber tube is then at once clamped close to its upper end, which is then connected with the inlet opening at the top of the chamber. Immediately after the passage into the chamber of the necessary volume of gas, that remaining in the holder is again at atmospheric pressure, and the upper part of the rubber tube is again clamped to prevent diffusion occurring between the gas in the tube and the air of the chamber.

Mr. James Grundy has adapted the test-chamber to deliver a portion of its gaseous mixture into other forms of testing apparatus which cannot themselves be introduced into the chamber. A bladder is placed in the chamber with its neck connected to a short pipe which passes air-tight through the side of the chamber, as shown in Fig. 585. This bladder is first emptied of air by suction at the open end of the tube, after which the gas and air are mixed in the chamber. A portion of the mixture is then expelled from the outlet tube in the chamber into a suitable receptacle by blowing down the bladder tube from outside the chamber. When the bladder is fully inflated it occupies a position in the chamber shown by dotted lines in the Fig. By this means Mr. Grundy has introduced a mixture of known proportions of air and gas into a Liveing's indicator.

Professor Clowes gives the following detailed dimensions of the test-flames which he used in his experiments. In the diagram, Fig. 586, the flames are indicated by the initials of the names in the table, P denoting the Pieler flame, H the hydrogen, O the oil, and B the benzoline flame. Where different heights of one kind of flame were used, a number follows the initial letter which denotes the flame. The heights are stated in millimetres and in inches, those of the oil flames, however, being only roughly approximate, since they vary much. The Pieler and Ashworth benzoline flames also cannot be adjusted with absolute certainty owing to their hazy tips. The hydrogen flame heights are absolute and invariable.

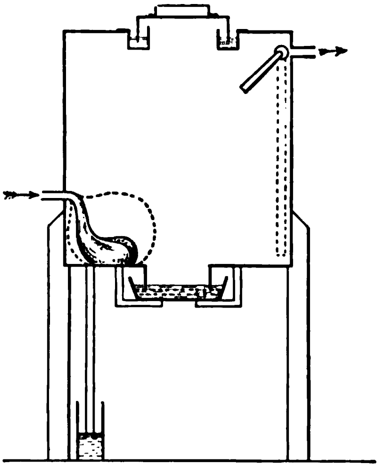


Fig. 585.—Arrangement for Supplying Gaseous Mixture from the Test-chamber.

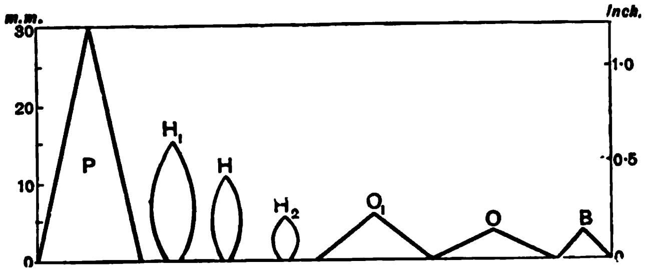


Fig. 586.—Dimensions of Testing-flames. (Actual size.)

Dimensions of Testing-Flames.

	Height.		Diameter at broadest part.	
	mm.	inches.	mm.	inches.
Hydrogen flame (round)	5	0·2	4	0·16
	10	0·4	5	0·20
	15	0·6	6	0·24
Pieler flame (round, conical)	30	1·2	13	0·52
Benzoline flame (round, conical)	3	0·12	7	0·28
Oil flame (flat, slightly conical)	3	0·12	13	0·52
	6	0·24	13	0·52

Professor Clowes also gives the following measurements of cap-heights arranged in tabular and in diagrammatic form, as the result of his observations by means of the test-chamber. The only measurements which, on repetition, were closely accordant were those of the hydrogen flame; in the case of other test-flames, the numbers given are the average results of several more or less differing readings.

Flame-cap Measurements with different Gas-testing Lamps in Air containing Methane.—Those marked with a query (?) are uncertain and cannot be relied upon.

HEIGHTS OF TESTING-FLAMES AND CAP-HEIGHTS, WITH KNOWN PERCENTAGES OF METHANE, OR PURE FIRE-DAMP.

Percentage of Methane Present in the Air.	Hydrogen flame.			Pieler Alcohol flame. 30 mm.	Ashworth's Benzoline. 3 mm.	Colza-petroleum flat flame.	
	Standard 10 mm.	15 mm. in the Gas.	5 mm. in the Gas.			Small blue. 3 mm.	Flame partly luminous. 6 mm.
0·25	17	37	—	30 (?)	—	—	—
0·5	18	42	—	55	7 (?)	—	—
1·0	22	60	—	90	10	—	—
2·0	31	enters top of lamp.	—	140 reaches top of lamp.	14	7·5 (?)	7·5 (?)
3·0	52	—	14·5	—	20	7·5	7·5
4·0	enters top of lamp.	—	22·2	—	25	12·0	24·0
5·0		—	35·0	—	30	29·0	41·0
6·0		—	60·0	—	35	67·0	enters top of lamp.

Multiply millimetres by 0·04 to convert them to inches.

The test-chamber devised by Professor Clowes is the best form of apparatus existing, not only for the purpose to which it was originally put, but as a means of familiarizing colliery officials and others interested in coal mines, with the appearance of the flame-caps, corresponding to different percentages of gas. It is true that they have facilities for observing the effect of gas upon the flame in the coal mine; but these observations are made with unknown percentages and would, therefore, be much more successfully undertaken, and much more valuable in their indications after repeated trials on the surface, with mixtures of known proportions of air and gas. The experience so obtained will be of the greatest value to those who will now probably have to make their examinations for gas with a more delicate and accurate flame-test than they have hitherto been accustomed to, and who will require, therefore, to know the exact appearance and dimensions of the cap which correspond to the different percentages of gas. A test-chamber for educational purposes, possibly provided with unbreakable vessels instead of the glass gas-holder and measuring-glass, might be placed in the colliery lamp-room, or other suitable place. Coal-gas in place of methane might be used for mixing with air in the test-chamber. The flame-caps obtained

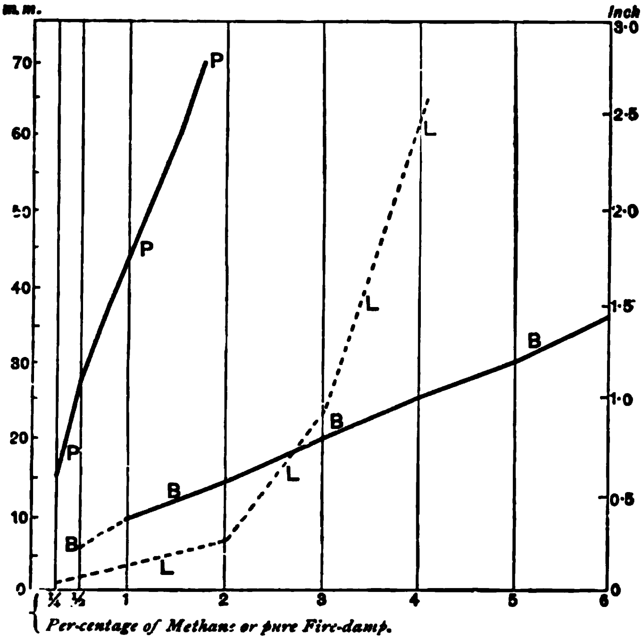


Fig. 587.—Fire-damp Indications to Scale.

P = Caps over Pieler Alcohol flame (one-half height).
B = Caps over Ashworth benzoline flame (full height).
L = Liveing's electrical indicator, relation between light emitted by covered and exposed spirals.

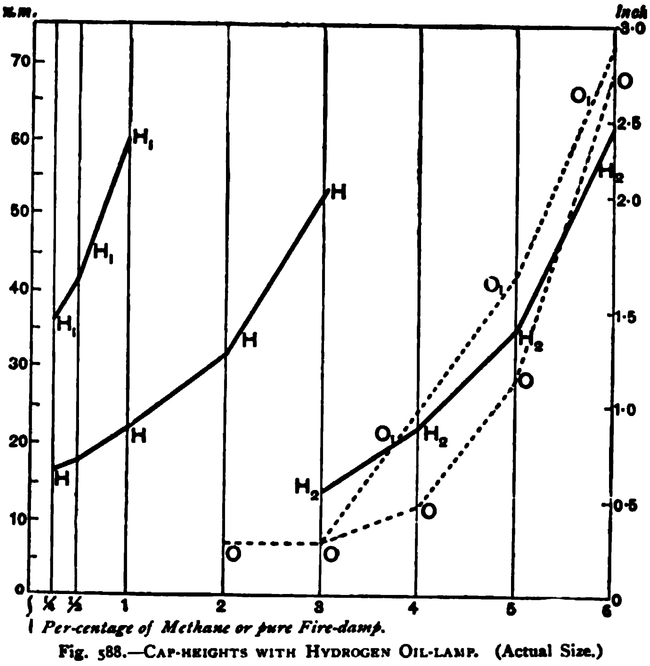


Fig. 588.—Cap-heights with Hydrogen Oil-lamp. (Actual Size.)

H_1 = 15 mm. hydrogen flame.
H = 10 mm. hydrogen flame.
H_2 = 5 mm. hydrogen flame.
O_1 = Maximum oil-flame.
O = Pale blue oil-flame.

by mixtures of coal-gas and air, although not identical with those obtained by methane, only differ slightly in size and appearance from the caps obtained in mixtures of marsh gas and air; they would serve perfectly well for training the eye of the observer. By means of a test-chamber placed at the colliery, young officials could thus prepare themselves for duties in the mine which necessitate a knowledge of the construction and use of delicate gas-detectors, and could familiarize themselves with the indications furnished by these detectors in the presence of known proportions of gas.

CARBONIC ACID GAS DETECTOR.

An apparatus for ascertaining the amount of carbonic acid gas, or carbonic anhydride, in the atmosphere of a mine is shown in Fig. 589.[*] It is the invention of Herr Pieler, and consists of a burette B with glass cock A, the long stem C of the burette being graduated to indicate definite proportions of the contents of B. D is a three-way cock, and F a vessel containing solution of caustic potash; E is an india-rubber pump by means of which the air to be experimented on is drawn into the burette through the cock A. When the burette is full the cock D is closed, and the contents of B are in free communication with the liquid in F. The contents of F and B are then shaken together, causing them to mix as much as possible, when, if there is any carbonic acid gas in the burette B, it will have been absorbed, and its place will be occupied by some of the fluid passing out of F into C, and when F is moved up or down till the top of the fluid in F is on a level with that in C, so that no undue pressure may be exerted to raise the level in C, the graduations on the stem will indicate the percentage of the gas which previously existed in the mixture and was afterwards absorbed by the potash.

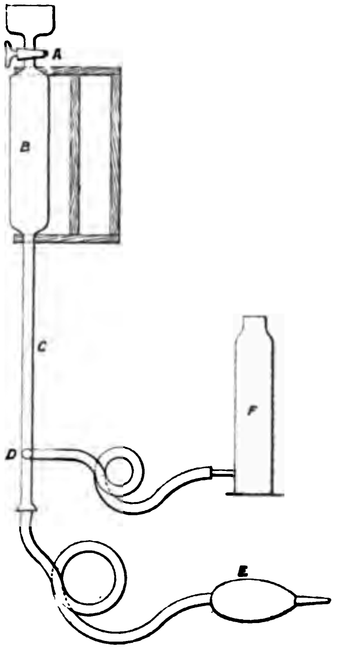


Fig. 589.—Pieler Carbonic Acid Gas Detector.

NAKED LIGHTS.

Candles and small Oil-lamps.—At many collieries, such as those of the Forest of Dean and Radstock, underground operations are carried on with unenclosed lights, the mines being absolutely free from all traces of fire-damp. There being no regular emissions or sudden irruptions of inflammable gas at the working faces or elsewhere, these mines enjoy a happy immunity from a source of danger common to others, and are, therefore, some of the safest to work. The simplicity of the means of illuminating such mines tends to promote the comfort and safety of those engaged in them, as the rays of light from the open candle or lamp are shed in all directions, thus revealing dangerous roofs, sides, or timbering much better than any form of safety-lamp can.

Where candles are used, the lighting is very inexpensive, especially if made, as at Radstock, in the colliery yard. The tallow and candle-cotton are bought and made into candles in the tallow-chandler's shop, from whence they are sent out to each pit, there to be given out to the colliers immediately before they descend the mine. About 20 candles go to the pound, and about 4 suffice for an 8-hour shift, the precise number depending upon the state of the air. The candle is generally held in a candlestick, so constructed that it can be carried through underground galleries in the hand or in the cap, see Figs. A and B, 590, and 591.

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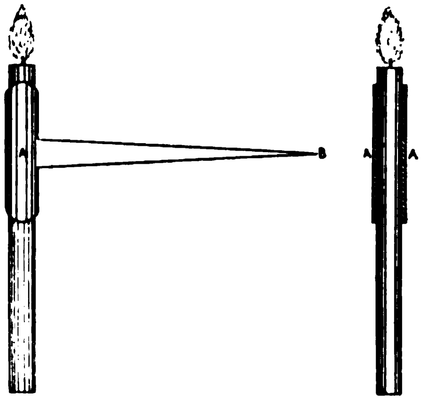
[*] See Transactions, North of England Institute of Mining Engineers, vol. xxxiv., p. 287.

The candlestick is made of iron or copper, and consists of two similar halves, Fig. 590. The socket A is roughly curved to press against the surface of the enclosed candle, while the shank is flat and made to diminish in size, until it ends at the point B. The two similar portions are soldered or brazed together near the point B, but are free to open at the socket, the two halves of which may be separated sufficiently to allow of a candle being passed in between them, after which they close over upon the candle, and hold it firm. When work is begun underground the candle may be placed in a loop sewn in the cap (see Fig. 591), or the pointed shank may be stuck into the timber, roadside, or floor, where it is shielded from the current of air. In travelling through small airways or around the faces of workings in a seam 1 foot thick, there is a great advantage in carrying the candle in the cap, so as to have both hands at liberty to assist in crawling. In strong air-currents, however, the candle-flame is blown from an upright position, and in burning rapidly melts the tallow on one side. This causes great inconvenience, as the hot grease streams down the candlestick to the cap, sometimes reaching the face, and if the candlestick becomes highly heated, as it will if the user neglects to raise the top of the constantly diminishing candle above the socket when required, the candle itself drops right out of the socket to the floor. Where a candlestick is not used, the candle is carried by means of a ball of clay, which may be moistened from time to time as it becomes dry, and thus will adhere to the timber, road-side, or floor, when it becomes necessary to free the hands. In the metalliferous mines of Cornwall, where the air-currents are not strong, the miners frequently use a ball of moistened clay to secure a tallow-candle firmly in their hats.



A. B.

Fig. 590.—Candlestick used in Somersetshire.

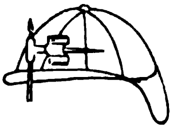


Fig. 591.—Candle carried in Leather Cap.

In the North of England boy-drivers are frequently employed on the main trolley-ways or horse-roads. These main roads are usually intake air-courses with strong currents of air flowing along them, and if a driver uses a candle the flame is often protected from the wind by what is called a "mistress." This is a rough wooden box-shaped case, consisting of a back shade of thin board about 8 inches × 4 inches, and having side and end pieces fixed to it. The space in front is larger than the board forming the back, so that the sides and ends are not placed quite at right-angles to the back. The depth of the mistress is about 4 inches, and the length is such as to admit of a whole candle being placed inside; this is kept in its place by a lump of moistened clay in which the candle is firmly imbedded. A long iron hook is fixed to the back by means of which the "mistress" is either carried or attached to the front tub of a set. The light so enclosed is not so readily extinguished in strong currents as if fully exposed.

With a proper supply of pure air a well-made tallow candle burns without unpleasant smoke emission, and in places where there is a certain proportion of carbonic acid gas in the air which would surely cause its extinction if left undisturbed, it may be kept alight by spreading out the wick so as to leave spaces between the threads of cotton, and then inclining the candle for the tallow to feed the flame faster. About 14 per cent. of carbonic acid gas in the air is usually sufficient to extinguish the flame of a candle in an upright position, and probably if the percentage is increased only a little, attempts to preserve the light by spreading out the wick or tilting the candle will be quite ineffectual, for 15 per cent. at once extinguishes the flame of candle, oil, or alcohol, and is quite unfit to support life.

Fig. 592 shows a form of oil-lamp much used in workings won by adit levels in the hills of South Wales. The cover over these workings is not great, and the surface-water penetrates the underlying strata into them, so that they are often wet and free from fire-damp. The vessel B for the oil is globular, and is made of brass or iron. The bottom of the lamp has a circular projection soldered or welded on, and this has a screw-thread cut on it to receive the holder C when the lamp is in use as shown in the Fig. It can then be carried in the hand or placed in the cap if a suitable loop be formed for the reception of the holder C. The top of the lamp A has a circular opening cut in it, and a similar projecting piece to that underneath. The holder C can thus be unscrewed from the bottom of the lamp, and screwed on above at A, where it forms a cover to protect the wick from injury and dirt, when the lamp is not in use. A screw-thread is also cut on the inside of the metal at A, and into this the round wick tube is screwed. The lamp may be trimmed by means of a pricker or long bodkin carried in the pocket. The lamps are made of various sizes, the largest holding enough oil for a 10-hour shift. The smaller ones used are re-filled during the day from a reservoir in the mine. In other districts different shapes of oil-lamp are in use, but they are generally made with a shank or fixed to the side of a large nail for carrying in the cap, a hook for hanging or a handle.

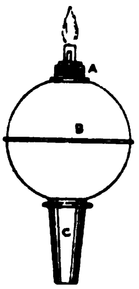


Fig. 592.—Open Oil Lamp used in South Wales.

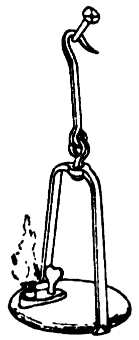


Fig. 593.—French Hanging Lamp.

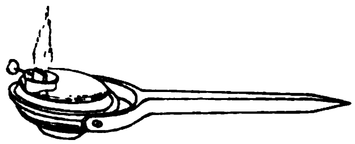


Fig. 594.—French Oil Lamp.

Figs. 593 and 594 are common forms of miner's oil-lamps used in the metalliferous mines and open light collieries in France. They are made of iron; the hanging lamp, Fig. 593, may be carried in the hand or suspended by the carrying-hook. The oval-shaped oil vessel is attached to the two legs of the striding support by pivots, so that it is free to swing. The two limbs of the support are inclined towards each other below, and curved above to join. The carrying-hook is attached above by means of a swivel-joint, which allows the lamp to be turned round in any direction. A round wick is used. The oil vessel of the lamp, shown in Fig. 594, has a flat wick, and is free to swing in the fork carrying it. The fork terminates in a pointed piece of iron, by means of which the lamp is carried in the hat, or pressed into the timbers or floor. If the iron point be stuck vertically in the floor the oil vessel can be turned so as to maintain the flame in an upright position.

Many other forms of light miners' oil-lamps are made in other countries, some of them being of elegant design.

In Saxony the oil-lamp used is enclosed in a square wooden box, which sometimes has a glass front, and is then virtually a lantern. Without the glass front the wooden box protects the lamp flame in the same way as the " mistress " does the tallow-candle. The miner wears a leathern strap round the waist, or suspended from the neck, and this is passed through the hook at the back of the box, so that the lamp is carried without encumbering the hands. The lamps are usually fed with animal or vegetable oils, but occasionally a mineral oil is mixed with the other. A large oil flame emits smoke which, where the air-supply is restricted, is very unpleasant, and in such places candles are much less objectionable. In some parts of Scotland tallow is used in suitable lamps.

Large Oil Lamps.—At many collieries the pit bank and main roads from the pit bottom are lighted by electricity or by gas manufactured in the colliery yard, but where this is not so, larger and more powerful lights than those used at the face are necessary in certain positions. A torch-lamp, such as is shown in Fig. 595, is often usefully applied at the bottom or other porches of the downcast shaft, where the large volume of air and high velocity do not admit of small lights burning. The torch lamp has usually a light wire handle, as shown in the illustration, with a hook at the top, if it is intended for hanging. Sometimes a covering of wood is placed over the horizontal part of the wire handle. The oil receptacle is in the form of a kettle, closed by a tightly fitting lid to which, or to the side of the lamp, is attached a chain carrying the pricker, for use on the burning wick when required. The oil is poured in at the top when the lid is lifted, and a large wick is used which must be long enough to reach down the spout well into the oil vessel. A common size holds about 3 pints of oil, large ones holding up to about 6 pints, and often having two wicks. Smaller torch lamps are also made of different shapes and used for special purposes. Torch lamps should be strongly made and almost unbreakable, so as to stand rough usage. They should give a fairly steady light, even when subjected to the strong currents of air commonly found at different points in the shaft undergoing repairs, &c. Further, they should be convenient for carrying from place to place, and should diffuse the light where it is wanted. The form of a torch lamp must, therefore, be modified to suit the special circumstances in which it is to be used.

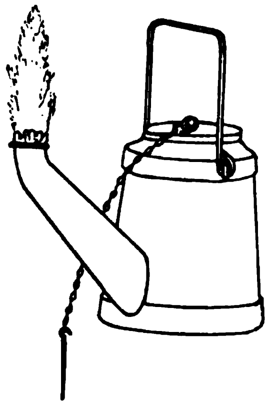


Fig. 595.—Kettle Torch-Lamp.

The danger of fire being communicated by large flames to the timber, &c., in the underground workings, and the contamination of the atmosphere caused by smoke from large oil lamps, render their employment at the working face quite out of the question, and even on main roads their use should be restricted as much as possible.

Oil lamps for fixed positions, such as engine houses and pit banks and pit porches, usually have the flames enclosed, so as to steady the light. These lamps are often furnished with glass chimneys, but sometimes metal cylinders with mica panes are substituted, as being less liable to breakage than glass. Various sizes of enclosed lamps are made, and their illuminating power ranges from about 20 to 100 candle power. Different shaped reflectors are used with them according to the direction in which the light is to be thrown.

The Sinclair Comet Lamp.—In this lamp, which is supplied by Messrs. Sinclair & Co., 19, Eldon-street, Finsbury, London, E.C., a very large and powerful flame is obtained by a special contrivance for vapourising the oil before it is burnt.

Figs. 596 and 597 show different sizes of the lamp. The oil tank is filled with common paraffin, petroleum, kerosene, lustre or Russian oil, to within 4 inches of the top. The flash point of the oil used should not be under 73° F. or higher than 100° F., the best results being obtained with oil that flashes at 86° F. Having regard to the safe use of oil that flashing at 100° F. is best. When partly filled with oil the filling hole is closed by a plug which is screwed down air-tight. The air-pump handle shown in Fig. 596 is now worked, and the air forced into the top of the tank until it reaches a pressure of 20 lbs., as shown on the pressure-gauge. One valve for air and another for oil are shown in Fig. 596, and may be turned, when required, to admit air or oil into the vertical pipe above.

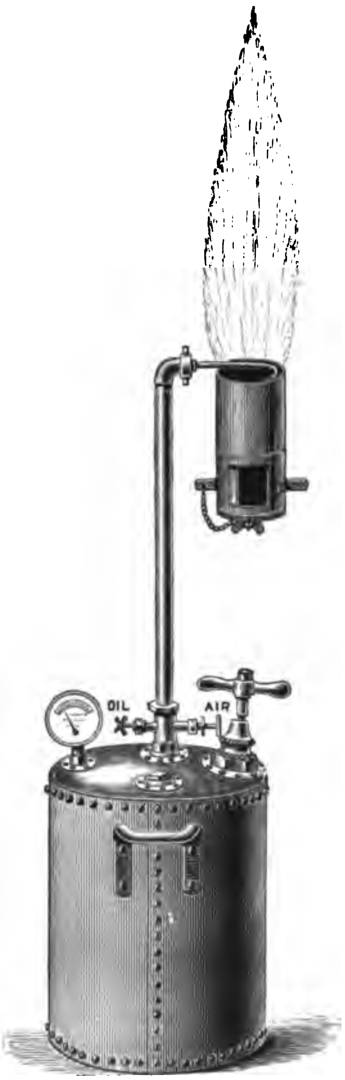


Fig. 596.

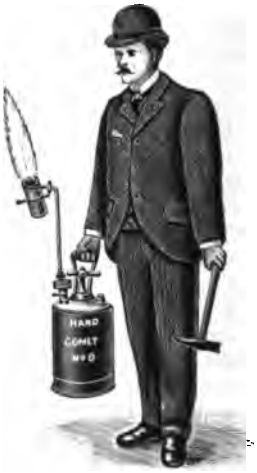


Fig. 597.

THE COMET LAMP.

At the top of this pipe the encased vapourising coil and burner are attached by a swivel joint, so that the flame can be turned at any angle. The case has a door or disc which may be partly closed to regulate the entry of air at atmospheric pressure on to the burner. This regulating door is shown more clearly in Figs. 598 and 599A, while the vapourising coil, out of the case, is shown in Fig. 599B. To start the burner, a piece of cotton waste is placed in the cup of the burner and a little oil is poured into the bottom of the case. The waste is then lighted, and the air disc is closed to within about ½ an inch. The tank air regulator is now turned to admit a little air from there, and in three or four minutes the burning waste will raise the coil to the right temperature. The tank air regulator is then closed and the oil valve opened. The air pressure must not be allowed to fall below 10 lbs., to prevent which the air pump must be worked only while heating. The air pressure forces the oil up the pipe until it comes within the influence of the heated coil which produces the vaporisation. The issuing vapour is ignited at the burner, and the powerful flame passing through the coil causes a continuous vaporisation of the ascending oil. The air must be regulated at the disc in the case, to prevent the flame from jumping or smoking, and also to secure the proper heat of the coil.

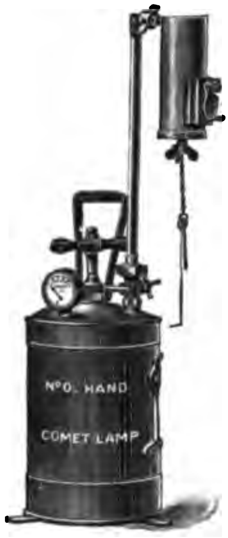


Fig. 598.

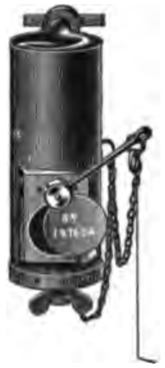


Fig. 599A.

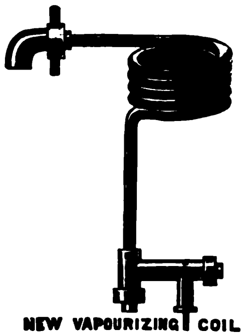


Fig. 599B.

Vapourising Coil and Burner in Case.

To put out the light air is pumped into the tank until it reaches a pressure of 25 lbs. The oil is then turned off and the air valve opened so as to allow all the air in the tank to escape by the burner. This thoroughly cleanses the coil and prevents the formation of a deposit of carbon. A steel pricker, the size of the hole, is used to keep the burner clean. It is necessary to use this sometimes when the lamp is burning, as in spite of great care foreign matter will get into the oil and choke the burner. When pouring oil into the tank a funnel lined with gauze should be used to prevent the entry of impurities. If through inattention the coil gets choked, it should be heated to a dull redness and allowed to cool after removing the bottom plug or nut. The coil is then tapped smartly all round with a hammer, and air from the tank is blown through it. This will remove the deposit. When it becomes necessary to re-charge the lamp with oil, the light must be put out. All lamps are tested before leaving the factory. To set the head at any angle, the oil valve is closed and the swivel joint on the standard is loosened. The head may then be turned to the angle required and secured firmly there by means of a screw.

The lamps are strongly made of steel or copper to give an illumination varying from 100 to 3,000 candle power, the following being some of the sizes and capacities:—

No.	Size.		Weight when charged.	Oil tank capacity in gallons.	Duration of supply in hours (approximately).	Consumption per hour (approximately).	Light in candle power (approximately).
	Height in inches.	Diameter in inches.					
000	12	6	20 lbs.	1½	4	2 pints.	200
00	13	7	28 ,,	2	5	2¼ ,,	360
0	14	9	36 ,,	3	5	2½ ,,	500
0A	17	12	56 ,,	6	4½	7 ,,	2,000
1	16	15	100 ,,	10	17	4 ,,	1,000
2	20	17	150 ,,	15	15	6 ,,	1,500
3	22	17	200 ,,	20	15	8 ,,	2,500

The lamps are not required underground, even if they could be safely used there, but they may be applied to light large areas in safe positions free from inflammable material on the surface, where the roaring noise with which they burn is not objectionable. The largest can be mounted on wheels and shifted from place to place with ease. The heat given by the No. 0 hand lamp is sufficient to melt lead, iron or copper rod; it may, therefore, be utilized for boiling water. A stand for a lead-pot is supplied, suitable for the plumber's use. It is employed in many positions, where petroleum oil is procurable, both for lighting and heating purposes. A good light can thus be secured where there is no fixed appliance, and there is a great advantage in the lamp being self-cleansing. The cost of lighting by comet lamps is from ½*d.* to 4½*d.* per hour, according to candle power.

At collieries where it has become necessary to enforce the use of locked safety-lamps in the place of open lights, the change is often viewed with aversion by the workmen, especially if the collieries have been working for a long period, and it has been the general practice throughout the district to work with open lights.

If collieries could be always worked by large shafts and large airways which could be constantly maintained without change, it might be possible to keep the workings of such as are only subject to slight and regular discharges of fire-damp in a state in which naked lights may be used, that is, practically free from fire-damp. In that case the open lights would be of great value in enabling the workmen to see and avoid other dangers in the mine which are not apparent with a feeble light. But, however well a colliery is ventilated, and however harmless the fire-damp may be rendered under the usual conditions of the mine, there is no such thing as absolutely preventing falls and blocks in the airways or controlling sudden irruptions of gas, and therefore the best possible system of ventilation does not prevent the necessity for the use of good locked safety-lamps. Even these, as we have shown, do not afford absolute immunity from danger of explosion in fiery mines, for a perfect safety-lamp suited to all conceivable risks has not even yet been invented. A careful and intelligent use of the best forms, however, will reduce the risk of explosion to a minimum.

CHAPTER XVI.

SUNDRY AND INCIDENTAL OPERATIONS AND APPLIANCES.

Coal-Dust—Watering the Underground Roadways—Explosives and Blasting Operations—Gunpowder—Gun-cotton—Tonite—Nitro-glycerine—Dynamite—Useful Work performed by Explosives—Blown-out Shots—Johnson's Tamping Plug—Charging, Stemming, and Firing Shots—Shot-firing Safety-lamps—The Lauer Detonator—Experiments with Wooden Plugs for Tamping—The Water Cartridge and Accessories—Sand and other means of Protecting Cartridges—Tamping with Wet Moss—Roburite—Bellite—Carbonite—Securite—Ammonite—Ardeer Powder—Westphalite—Lime Cartridges—Wedges for Coal-getting—Macdermott's Rock and Coal Perforators—Ingersoll Hand-power Rock Drill—Ingersoll Machine-power Rock Drill—Gillott and Copley Rotary Coal-cutting Machine—Bower, Blackburn, and Mori Electrical Coal-cutting Machine—Stanley's Coal Heading Machine—Caging Appliances and Drop Staples—Pit Horses, their Food and Work—Fleuss Apparatus for Breathing in Noxious Gases—Fleuss Lamp—Exploring for Water—Underground Dams—Water-blasts—Underground and Surface Fires—Testing the Roof—Driving through Faults—Watt's Steam Indicator—Richards's Indicator—Use of Indicator Diagrams—Continuous Diagrams—The Thompson Indicator—Schäffer and Budenberg's Double Indicator—Bourdon's Pressure Gauge—Schäffer and Budenberg Bourdon Gauges—Steel Tube Gauges for Very High Pressures—Duplex Gauges—Graduating Ordinary Pressure Gauges—Graduating Steel Tube Pressure Gauges—Bourdon Vacuum Gauges—Schäffer Diaphragm Gauge—Testing Vacuum Gauges—Lightning descending Shafts—Dunford and Emen's Patent Automatic Tub-greaser—Self-lubricating Pedestals for Colliery Tubs.

COAL-DUST.

Closely connected with the lighting and ventilation of collieries is the effect of coal-dust in promoting or increasing the disastrous effect of explosions. It is only of late years that attention has been to any extent directed to this question, but the experiments by Mr. W. Galloway (whose name will always be honourably associated with this matter), and those of the Royal Commission and others, prove beyond doubt that the existence of fine coal-dust in the underground roadways and workings of fiery mines is a dangerous element, and that it has played an important part in colliery explosions. Coal-dust consists of minute particles of coal, and when these are mixed with the air in the workings they appear to be liable to combustion. It is doubtful whether explosions of coal-dust occur on any large scale where the mine yields no fire-damp, although a blown-out shot may fire the coal-dust. It is stated, however, that if a small quantity of fire-damp be present and fired in this way, its effects are much intensified and the explosion is extended along the roads containing coal-dust, which thus feeds the flames of the explosion. Although there may only be a small quantity of gas, the flames, when thus started over a dusty road, are carried to a point far beyond that due to the simple explosion of the inflammable mixture of fire-damp and air. These facts are now fully recognised, and in dry fiery mines, where coal-dust is largely produced, it is becoming a general practice to remove the dust, and to moisten the roof, floor, and sides of the roads. Mr. Galloway says that 1 per cent. of fire-damp mixed with coal-dust and air forms an explosive mixture. Sir Frederick Abel states that from 2 to 2½ per cent. of fire-damp in the mixture is necessary.

Since the publication of the official report on the circumstances attending the Camerton Colliery explosion, which occurred on the 13th Nov., 1893, it seems well established that coal-dust as found in the mine is itself explosive in the entire absence of inflammable gas, and under such conditions as may ordinarily be present in the mine. The Radstock series of seams are worked at Camerton, and no fire-damp has ever been found in them either at Camerton or elsewhere. After the explosion, tests were made with Prof. Clowes' hydrogen lamp, and with Mr. Stokes' alcohol lamp, the former of which is capable of detecting $\frac{1}{4}$ per cent. and the latter $\frac{1}{2}$ per cent. of fire-damp in the air, without their revealing the presence of either of these low percentages in the workings even after these had been sealed for a time and then re-opened. The explosion originated on a self-acting incline where the roof was being ripped, and it caused the death of a man and a boy, who were the only persons employed in the district at the time. The explosive they were using was powder, and there was evidence that the deceased had fired a shot in the roof and had taken refuge in a manhole. When found, both bodies were more or less burned, and appearances pointed to their having endeavoured to escape when overcome. Coked dust was found on some of the timbers in the horse-road outside the bottom of the incline, as well as on a few in the incline itself above where the shot was fired, and at portions of the horse-road which were dusty, the dust when examined under the microscope appeared to have been in a state of incandescence. There were indications of force both inwards and outwards which began at a distance of 80 or 100 yards on each side of the shot, and practically ceased at points where the supply of dust ended. About 1,100 yards of roadway were wrecked by the explosion. The shot had done a part of its work, and it seems almost certain that being overcharged it disturbed the dust on the floor under it, which on rising in a fine cloud was ignited by the flame from the powder and thus initiated the flame and force of the explosion, which was afterwards fed by dust alone.

Beside the evidence obtained from the investigation of the Camerton explosion, further proof that dry coal-dust alone, under such conditions as are to be found in the mine, may be fired by a blown-out gunpowder shot and cause an explosion, is given by the interesting experiments made by Mr. H. Hall, H. M. Inspector of Mines, and reported on by him to the Secretary of State on Jan. 23, 1893. In Mr. Hall's experiments a charge of blasting-powder was fired by electricity from a cannon suspended in a wet shaft, the air of which was proved by careful chemical analysis to be absolutely free from any trace of inflammable gas. In some of the experiments, even in the absence of dry dust in suspension in the shaft, the dust lodged on the timbers was exploded by the firing of a charge of $1\frac{1}{2}$ lbs. of gunpowder from the cannon.

From the remarks made on Ventilation, it is easy to see that any ventilating current after descending the downcast shaft will have its temperature raised as it passes round the workings; its capacity for absorbing moisture being thus increased, in comparatively dry mines the natural result of such absorption is to leave large quantities of dust. Nearly all deep mines are dry, and if fiery as well, the usual practice now is to water them. A simple form of watering the roadways is by a water tram having a perforated pipe through which the water spurts out much in the manner of town water carts. This water tram may be attached to the sets of tubs on engine planes, or be hauled about by a horse. But the distribution of the water by this means is not very perfect, and latterly the watering of mines has been done by means of pipes and jets. In Mr. Blakemore's patent, a pipe is carried along the roadway to be watered at a convenient height above the floor. A parallel pipe, connected at intervals with the main pipe, has continuous small perforations, the latter being in constant operation discharging jets or sprays of water. A cistern with an ordinary ball tap is placed at a suitable height in the shaft to get the required pressure, and this cistern supplies the pipes leading into the workings. The air current is thus kept moist as well as all the surfaces, and no dry dust can be raised.

In recent experiments made for laying the dust, efforts have been directed to the most suitable means for perfectly saturating the intake air, and from these it appears that this may be done by the use of specially made "spray-producers." From these the water issues in the form of spray or mist of such exceeding fineness as to be absorbed by the intake air to the point of saturation. This condition of the air is maintained by placing the spray-producers at suitable intervals along the roadway. A vertical branch pipe having an internal diameter of ½ an inch is laid from the main road pipe to the roof and carried to the centre of the roadway. The pressure of the water may be from 100 to 150 lbs. per square inch, and issues downwards through the spray-producers, which are placed at intervals of about 50 yards along the roadway. Each branch pipe is provided with a regulating cock.

At collieries where compressed air is used, a better result in the system of damping the intake air will be obtained by the use of compressed air in conjunction with water conveyed in the pipes as just described. A main pipe is necessary to convey the compressed air along the main intake, but probably that is already laid to supply some machinery in the workings. At each spray-producer a ½-inch branch pipe leads out of the main compressed air pipe and may be laid parallel to the water pipe branch. The compressed air pipe is connected to the water pipe by a nozzle in the interior of an ordinary T-pipe, which forms the junction of the air and water. The water is driven out by the air through an adjustable spray-producer, which is regulated by means of a nut and screw. Spherical valves are placed in both the air and water pipes to prevent the water from passing into the air pipes and also the air from escaping into the water pipes should any accident occur to the water main. The air and water issuing at the spray-producer become mechanically mixed, in infinitesimal globules, and as the pressure of the air gives considerable impetus to these, the moisture is carried further than with the globules of water unmixed with air. Consequently, where compressed air is used, the spray-producers may be placed at greater intervals apart, probably from 100 to 400 yards, to suit the circumstances of each colliery. A further advantage of the air-and-water system arises from the fact that a much lower pressure of water can be used. Indeed, it is only necessary for the water to have sufficient pressure, when throttled down, to find its way into the chamber where the compressed air meets it and drives it out. In shallow pits, where no head of water can be obtained, this may be a better means of laying dust than adopting some expensive plan to get sufficient head of water.

The working faces may be damped as well as the main roads by either the water, or the air-and-water system. The pipes are laid to the face and the spray-producers fixed there. In compliance with the Act of Parliament, no shot is to be fired in the presence of dust, but if the dust is destroyed by damping, for a radius of 20 yards from any shot-hole and no fire-damp be present, the competent man appointed for the purpose may fire the shot.

EXPLOSIVES AND BLASTING OPERATIONS.

Probably no mining subject has had more attention during the past few years than that of blasting. By the Mines Act, 1887, various new restrictions are imposed on the use of explosives, both as to the actual process of blasting, and the places where it is permitted.

The use of explosives is intimately connected with the lighting of mines, and their freedom, or otherwise, from gas and dust. Thus, wherever safety-lamps are used, shots can only be fired by or under the directions of a competent person appointed for the purpose. Similarly, wherever a mine is dry and dusty the same restriction is imposed. Further, where safety-lamps are used, the competent person may not fire a shot unless "he has examined the place itself, and all contiguous accessible places of the same seam within a radius of twenty yards, and has found such place safe for firing; " and if gas has been reported in the ventilating district at any of the four inspections recorded last before a shot is to be fired he must also examine the place or places, and see "that such gas has been cleared away, and that there is not at or near such place sufficient gas issuing or accumulated to render it unsafe to fire the shot," or if this is not the condition of the place as regards gas the shot can only be fired if it is "so used with water or other contrivance as to prevent it from inflaming gas, or is of such a nature that it cannot inflame gas."

Then, if the place is dry and dusty (without reference to gas), such place and all contiguous accessible places within a radius of twenty yards, must be thoroughly watered, or have treatment equivalent to watering, but if watering would injure the roof or floor, then the explosive may be used with water or other contrivance so as to prevent it from inflaming gas or dust, otherwise the explosive must be "of such a nature that it cannot inflame gas or dust."

A further restriction is applicable to any part of a main haulage road, or place contiguous thereto, showing dust adhering to the roof and sides. Not only must the place of firing, and within a radius of twenty yards, be watered, or have treatment equivalent to watering, but the explosive must be used with water, or other contrivance, so as to prevent it from inflaming gas or dust, unless it is of such a nature that it cannot inflame gas or dust. An alternative to this is, that one of the conditions mentioned must be observed, and all workmen—except those engaged in firing the shot, and others (not exceeding ten) employed in attending furnaces, boilers, engines, signals, or horses, or inspecting the mine—removed from the seam or seams on the same level.

Thus it will be seen that, with one exception, greater stringency is now required in the matter of blasting than formerly. Under the Act of 1872, where gas had been found issuing so as to show a "blue cap" on the flame of a safety-lamp, shots could only be fired in coal-work when persons ordinarily employed were out of that part (which was decided to mean the "ventilating district") of the mine. Now, if certain safeguards are employed, shots may be fired with all the men in the mine.

Where an explosive is required in the actual getting of coal, as in those cases where the reduced percentage of large coal and its enhanced cost with hand labour alone do not result in a fair profit, *Blasting powder* is still used, notwithstanding many objections. Its composition is given in Chapter XI. of this work.

The explosive force of gunpowder depends on the sudden formation of gases, chiefly nitrogen and carbonic acid, which, at the high temperature at which they are evolved, amount to about 2,000 times the volume of the powder employed. The granular form of the gunpowder increases the rapidity of its combustion, as the flame is better able to penetrate it, and thus kindle every grain almost at the same time. For this reason, in mining, where it is desirable that the combustion should be comparatively slow, the powder is coarse-grained. The temperature at which it explodes is about 600° F. The products of its combustion are given elsewhere. The combustion being comparatively slow, the pressure resulting from the expansion of the gases has more time to act on the mass, and, in coal-getting, to rend it without much shattering effect, thus admitting of larger coals being obtained than by the aid of a quicker explosive.

A great drawback to the use of gunpowder in mining is the large amount of flame emitted from the shot on ignition. This emission of flame is a thoroughly recognised element of danger in mines producing fire-damp, especially where they are dry and dusty; and even without fire-damp many mining engineers, managers and others, are firmly convinced that there is considerable danger in using powder where a place is dry and dusty. Without actually prohibiting its use, the Legislature has endeavoured to minimise the risk in both cases by the stringent regulations already referred to.

Cartridges made of compressed powder have been used during the last few years, the object being to obtain within a minimum volume a given store of power. These cartridges are very convenient, being purchased ready for use, and obviate the danger attending the making of cartridges from loose powder, which operation is often left for the miners to do at their homes. Where the shot-hole can be bored truly circular there is no practical objection to the use of compressed-powder cartridges. Numerous accidents have, however, happened with them, probably caused by the cartridges having exploded owing to the heat generated by excessive friction while being forced into holes either too small or not truly circular.

The Mines Act, 1887, prohibits explosives being taken into a mine except in cartridges, unless such mine is exempted from the clause by order of the Secretary of State. Although not expressed, probably it is the intention of the Act that the explosive shall not only be taken into the mine, but also be used in cartridges —unless the mine is exempted. The grounds for claiming the exemption are not stated in the Act, but apparently this provision is intended to meet those cases where it is more practicable to use powder in the loose state, and because of a difficulty in boring holes round enough for cartridges, which difficulty undoubtedly exists in certain kinds of rock, and even in fireclay containing nodules of ironstone.

Many modern explosives are nitro-compounds, and these may be reduced to two, viz., gun-cotton and nitro-glycerine. All the nitro-compounds have low points of ignition, ranging from a few degrees below 200° C.

In the manufacture of *Gun-Cotton*, cotton waste is steeped in a mixture of sulphuric acid and nitric acid. During this process three equivalents of hydrogen are removed by the oxidising action of the nitric acid, and replaced by three equivalents of nitric peroxide. After steeping, it is washed in water, and cleansed from acid, following which it may be stored, or converted into other compounds. Although its constitution is different from what it was when placed in the bath, its appearance remains the same.

Gun-cotton explodes at 400° F., and sometimes at a much lower temperature, and as gunpowder explodes at 600° F., gun-cotton may be fired on gunpowder without igniting it. It has double the explosive force of gunpowder, and has the advantage of yielding no smoke. Time, moisture, and exposure do not alter its qualities. It is exploded by detonation. Cotton-powder is gun-cotton reduced to a fine state of division.

Tonite is cotton-powder, with the admixture of a nitrate or similar body.

In the manufacture of *Nitro-Glycerine*, a similar chemical change takes place to that obtained in the manufacture of gun-cotton. In this case the nitrification of a liquid is effected, viz., glycerine. Glycerine is mixed with nitric acid, and the mixture allowed to fall or drop in a narrow stream into water, when the nitro-glycerine immediately separates.

The action of the nitric acid removes three equivalents of hydrogen, and replaces them by equivalents of nitric peroxide, and without apparent change in the material.

Nitro-glycerine in its liquid state is inconvenient to use. Moreover, it is highly dangerous while in this condition.

It is susceptible to ignition in two ways. If burned at the wick of an ordinary spirit-lamp, and the experiment be conducted by a skilful operator, it burns quietly and harmlessly. This experiment is too dangerous to be entrusted to any but skilled hands. Similarly, gun-cotton may be comparatively safely lighted by applying a flame to it, when it will burn rapidly without explosion.

When nitro-glycerine or gun-cotton is ignited by means of a detonator, extreme rapidity of decomposition and combustion follow, and cause a violent explosion.

Dynamite is manufactured by impregnating a spongy kind of clay, called kieselguhr, with nitro-glycerine, in the proportion of 25 parts of the former to 75 parts of the latter, so as to form an explosive of a less objectionable nature. When in a proper condition, dynamite is plastic, safely handled, and very convenient for use as an explosive. Irregularly shaped shot-holes may be charged with it. It does not explode by spark or flame, but by detonation.

When the cartridges are below 32° F. in temperature, they will not explode except with difficulty; indeed, they are not in a safe condition when under 40° F., at which temperature the nitro-glycerine in them freezes. When in this condition they require to be thawed, or "tempered," before use, and this operation should only be attempted by the use of a "warming-pan," such as the makers of dynamite supply, and the instructions issued by the makers with each packet of cartridges should be strictly adhered to, in order to prevent accident.

The explosion of pure nitro-glycerine being so much more destructive than that of an equal weight of dynamite, efforts have been and are still being directed to substitute for kieselguhr (which is an inert substance) some absorbent body which will also assist the explosion. Thus, Litho-fracteur has a proportion of clay, with saltpetre and sulphur. Other combinations have been tried, all of which act very similarly.

Shots charged with the nitro-compounds, or high explosives, as they are sometimes called, are violent in their action, and shatter the material in which they are exploded. Although unsuitable for coal-getting, they are most usefully applied in extra strong rippings, in stone drifts, in sinking shafts, or where the presence of water interferes with the easy use of powder.

The actual work performed by any explosive used in blasting operations is limited by incomplete combustion, compression, and other changes in the material operated on, energy given out in cracking and heating material not displaced, and escape of gases through the shot-hole and fissures caused by the explosion. The useful work produced by various explosives has been estimated at from 14 to 33 per cent.

BLOWN-OUT SHOTS.

When the explosion of a shot results only in the blowing out of the stemming, or in doing little or no work, it is said to be a "blown-out shot." This result may be caused either by the shot being badly stemmed, or by its being improperly planted with regard to the work which it is intended to do. In the blasting of coal, if the shot-hole extend even a short distance beyond the holing, so as to make a "fast-end," there is liability to a blown-out shot; in stone-work especially, unless skill is exercised in so planting the hole that it shall not have too much resistance, there is still greater liability for this to happen.

For years this subject has had considerable attention. It is a very important one, for undoubtedly many serious explosions have arisen through the flame from a blown-out shot igniting gas, or gas and dust, and, in some instances, dust alone.

Experiments have shown that in air containing 6 per cent. of inflammable gas the flame from a blown-out gunpowder shot extends to 15 yards, but under such circumstances no shot should on any account be fired. Even with no gas the flame will reach a distance of 13 feet, with clay used for stemming.

Johnson's tamping plug is said to be a reliable remedy for blown-out shots. It saves time in stemming, and its use ensures getting the greatest possible amount of work from the explosive charge. It is said that it effectually prevents the shot from blowing out even in fast holes. A special drill has been designed to bore the hole, which is circular in shape. The plug is made of phosphor-bronze 14 inches long, and accurately fits the shot-hole. A small machine for making plastic pellets from the underclay of the coal-seam accompanies the plug and drill. After the explosive charge is placed in the shot-hole, to prevent the gases from entering between the plug and the sides of the hole, one or more of the plastic pellets is interposed between the charge and the plug. One or more of the pellets is placed over the plug outside and the tamping is completed. The pressure of the gases given off from the explosion must go on increasing till the material in which the hole is drilled, yields, or the whole of the charge is burned. Even with flameless explosives, the use of the plug is to be recommended, as it ensures the greatest possible effect from the charge used, and the plug may be used over and over again. To facilitate its recovery after use underground it may be painted white.

CHARGING, STEMMING, AND FIRING SHOTS.

The comparatively minor operations of charging, stemming, and firing shots are of sufficient importance to require careful study, for although they are left to ordinary workmen as a rule, they nevertheless demand skill and care on their part as well as direction and guidance on the part of those acting in the management. The firing of shots has always been a source of danger in mines, and with the view of the further diminution of accidents the subject has, under the Act of 1887, more restrictions imposed on it than formerly. Thus, it is now made clear that all iron or steel tools are prohibited in the charging and stemming of shots, the object being to guard against accidents from sparks kindling the charge prematurely. Coal or coal-dust must not be used in stemming, thus reducing the chance of a large flame issuing from the shot. There can be no doubt that the use of small coal for stemming lengthened the flame from a gunpowder charge very considerably. A further restriction is, that no explosive shall be forcibly pressed into a hole of insufficient size, which, again, is intended to minimise the risk of premature explosion.

The unramming of a shot which had missed fire was formerly an illegal act, and it remains so. Besides this, where a shot has missed fire, another hole for a charge shall not be bored at a distance less than six inches from the former.

There can be no question as to the necessity for these restrictions in the use of explosives, nor any doubt that the proper observance of the whole of the regulations relating thereto will tend to increased safety.

In the stemming of shots, probably no better material can be used than clay, made up in pellets. The mode of procedure is as follows:—After the charge has been carefully pushed to the end of the hole, a pellet of dry clay is first put in and pressed firmly against the charge, a copper or wooden rammer being used for the purpose. Other pellets are inserted, one at a time, and pressed in by the rammer, which may then be struck lightly by a hammer, and the operation repeated until the hole is filled. Where no suitable clay is available care should be taken to use a soft material, free from grit, but which is not of a coaly nature.

Shots are fired by the aid of either straws, paper squibs (each filled with fine powder), safety fuze, or by an electric shot-firer.

Where a straw or squib is used, after the explosive charge has been placed in the hole, a copper pricker or needle is inserted, the pointed end being pushed gently into the charge, whether consisting of a cartridge or loose powder. The handle end of the needle projects from the hole a few inches. The rammer has a groove formed at its side, so that it may slide on the needle during the operation of stemming. When this is completed the needle is carefully withdrawn by giving it a twisting motion while doing so. The straw or squib is then inserted in the needle-hole, and the space between the stemming and the straw or squib closed at the mouth with a piece of soft clay. A "match," consisting of a piece of touch-paper, or a piece of lamp-cotton twisted straight and oiled, is fixed on the end of the straw or squib, and the shot is then ready for lighting.

An objection to both straws and paper squibs is that they may not always allow sufficient time for the shot-firer to be well out of the way before the explosion takes place.

In firing by means of safety-fuze a needle is unnecessary. The fuze is put in the shot-hole in contact with the explosive charge and the hole stemmed, the coating of the fuze preventing injury thereto during the process. The best fuze should be used. It burns at the rate of about 32 inches per minute, thus allowing more time for the shot-firer to withdraw to a place of safety. To ignite the fuze the shot-firer opens about an inch of the end into two or three parts, and fixes in it a piece of tinder or touch-paper. Where safety-lamps are in use, the touch is kindled by a fine wire passed through a hole in the gauze of a safety-lamp, the ends of the wire remaining in contact with the flame and the touch-paper respectively until the wire becomes red-hot and the touch burns, unless a special shot-firing safety-lamp or Bickford or other chemical lighter is used.

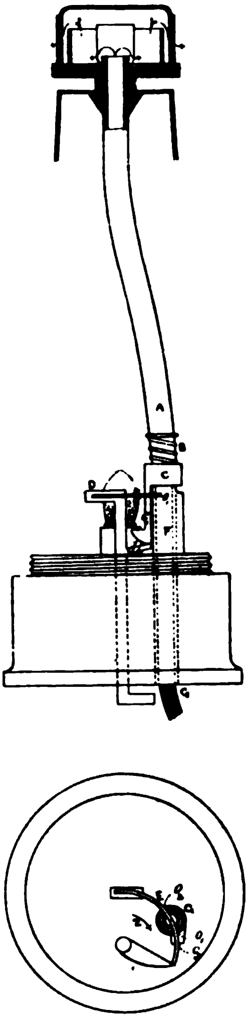


Fig. 600. — PLAN AND ELEVATION OF THE HEATH AND FROST SHOT-FIRING LAMP.

Some forms of shot-firing lamps have been devised, which are thought to be more convenient in use than an ordinary safety-lamp and fine wire. Almost any type of safety-lamp may be used for the purpose, and have fitted to it the shot-firing appliance. Fig. 600 shows the Heath and Frost shot-firing lamp. Within the lamp is a tube, A, passing through the oil-reservoir, and extending upwards. The upper part carries an inverted truncated cone, which forms a support for the lamp-gauze. Its upper extremity is covered with a double gauze cap, against which the sparks given off in ignition are thrown. At the height of the wick the tube is pierced with two little holes, o_1, o_2, placed on the same arc of a circle. The fuse, G, to be kindled, is passed up the tube from beneath, the products of combustion escaping upwards.

The tube, A, is surrounded with two cylindrical cases. The upper case, C, is movable, and is provided with an arm, c_1, ending in a catch which is moved by a similar catch, *d*, attached to the working lever, D. When moved, it compresses a spring, B. The lower fixed case, F, carries a small, square guide, pierced by a hole, o_3, at the same height as those in the tube.

The lever, D, is moved by a small handle placed below the oil-vessel, and carries, soldered to its end, an iron wire, E, ·04 inch diameter, curved to form an arc of a circle, and kept in its course by the guide on the fixed case, F.

To fire a shot with this lamp a slight rotary movement is given to the handle, upon which the lower cam lifts the catch, c_1, of the upper movable case. The spring, B, is compressed, and the openings, o_1, o_2, allow the passage of the curved wire which is to be made red in the lamp-flame. The end of the fuse projecting from the shot-hole is introduced into the tube, and pushed upwards till it just touches the curved wire.

After the wire has become red-hot, if the handle is turned in the direction of the arrow, 2, or opposite to the direction last turned, the red-hot wire passes over the fuse, and ignites the powder. The gases from combustion escape up the tube; the spring, B, causes the return of the movable case, C, over the lower one, F, and the openings, o_1, o_2, communicating directly between the chamber of combustion and the external air are closed.

The fuse should not be withdrawn from the lamp until 3 or 4 inches of the fuse are burned, as a protection from the sparks at first given off. It should be of good quality, and free from defects likely to cause lateral projections of flame.

Fig. 601 shows the Roberts shot-firing lamp. C is a brass tube, $\frac{5}{16}$ inch in diameter, extending from the oil-vessel, J, through which it passes, and surmounted by a brass box, A, carefully fitted to it. The brass box, A, has a protecting cover of gauze, B, fitted to it, against which the sparks given off by the fuse strike. The brass tube has a hole, E, $\frac{3}{16}$ inch in diameter, cut in it at a point opposite the flame, which, when not in use, is covered by a short cylinder, D, fitting over the pipe, and operated by the pricker, H. By this means it is pulled down against the spiral spring, G, and held securely there during the exposure of the hole, E. On releasing the pricker, the cylinder, D, returns by the reaction of the spring, G, till it covers the orifice, E. An additional means of safety is provided by a second cover, M, worked by a spring, P, the position of which is outside and underneath the oil-vessel, J. F is a blow-pipe passing through the oil chamber, J, by means of which the flame of the lamp can be blown to one side, and so be brought to bear upon the opening, E, in the tube, C. A slotted cover, K, closed automatically by the spring, N, protects the outer end of the blow-pipe tube.

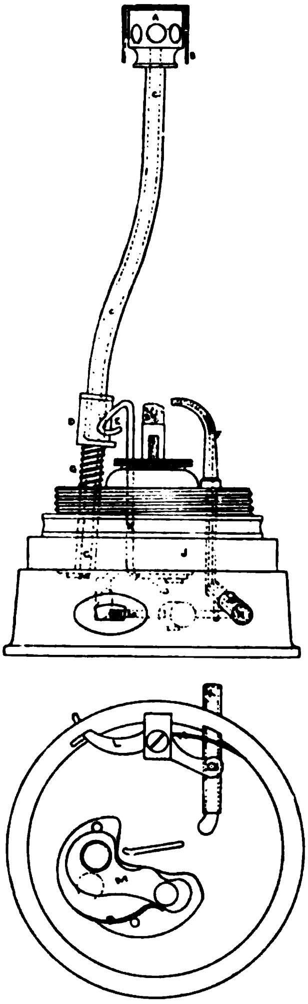


Fig. 601. — Plan and Elevation of the Roberts Shot-firing Lamp.

To fire a shot with the lamp which has been tamped in the ordinary way, the fuse outside the shot-hole is left of the right length for the purpose, and is then inserted underneath the oil-chamber, and pushed up the brass tube till the end is level with the flame, and in that position it may be exposed at the opening, E. The cylinder, D, is drawn down by means of the pricker, H, the mouth applied to the blow-pipe, K, its cover removed by pressing the lever, L, after which the flame is blown on to the end of the fuse at E, and causes its ignition. The pricker, H, is released, immediately upon which the reaction of the spiral spring, G, causes the return of the cover, D. Care should be taken by the shot-firer not to withdraw the fuse from the lamp until certain that sparks will not be thrown from the end of the fuse on its removal from the lamp. An emission of sparks only ceases after a Bickford fuse has burned for a length of from 2 to 3 inches.

Messrs. Bickford, Smith & Co., Limited, of Tuckingmill, Cornwall, are the manufacturers of Bickford's safety fuses, which they make in numerous qualities adapted to every requirement of blasting. The safety fuse is employed to fire the charge in every kind of mining, quarrying, and subaqueous blasting, whether with powder or any modern explosive. It is usually supplied in coils of 24 feet, and must be stored in a dry place, and free from contact with oil or grease, which substances are prejudicial to its composition.

To those engaged in coal-mines, the *patent colliery fuse* will be of the greatest interest. This fuse is one of Messrs. Bickford's more recent specialties; it is designed to obviate the dangers attending the use of safety fuses in fiery or gaseous collieries, as recommended in the Report of the Royal Commissioners on Accidents in Mines (1886), and to meet the requirements of the Coal Mines Regulation Act, 1887. The chemical and other means employed in the manufacture of this fuse, render it practically smokeless, a desideratum in close places, but its speciality is that it does not emit spark or flame or highly heated gases laterally during combustion, and it may therefore be safely used with gelatinous or water cartridges in fiery mines. Closely allied with this patent is another recent invention of Messrs. Bickford, namely, their *patent safety lighter*, introduced to remove the difficulty of passing wires into safety-lamps, or of using special shot-firing lamps. This little instrument, shown in Fig. 602, effectually prevents the emission of spark or flame when the fuse is fired. The lighter consists of a tin tube having one end closed and the other open for the reception of the fuse. A small glass tube containing sulphuric acid is placed within the tin tube, the glass being in contact with a little chlorate of potash and sugar. One end of the fuse is inserted into the open end of the tin tube, and the mouth of the lighter is then closed firmly, but not too tightly, around it by pressing its open end with the Bickford nippers (see Fig. 603), the slots in the handles being made for this purpose. The fuse is then lighted by squeezing the centre of the tube with the nippers (see illustration). The grip of the nippers breaks the glass tube and releases the sulphuric acid which then ignites the chlorate mixture, and this in turn ignites the end of the fuse. The chemical action which takes place inside the tube produces no objectionable external effect, as all sparks are kept within the tube.

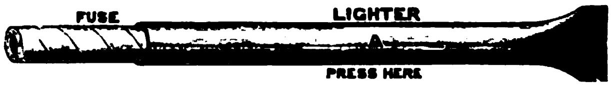


Fig. 602.—The Bickford Patent Safety Lighter.

Messrs. Bickford also make an instrument which may, under certain conditions, be found a useful alternative to the patent colliery lighters, called the *Pistol shot-firer*. To use this, one end of the fuse is fitted with a specially-adapted percussion cap, after which it is placed in the longitudinally divided barrel. The barrel being closed, the fuse may be fired by a pull of the trigger. When ignited, the fuse is withdrawn with the cap on it by opening the barrel. The combustion which then goes on in the fuse is completely cut off from the surrounding atmosphere.

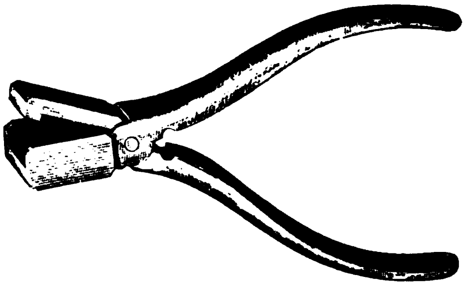


Fig. 603.—THE BICKFORD NIPPERS.

Until recently, the simultaneous ignition of a number of charges was only to be obtained by means of electric machines and appliances, but Messrs. Bickford have now patented a simple and cheap *volley-firer and instantaneous fuse*, by which the same result may be produced. If used in a fiery mine, the volley-firer must, of course, be placed in the care of the competent man appointed to fire the shots.

The invention consists of an appliance (see Fig. 604) called the volley-firer, one end of which contains an ordinary safety-fuse, and the other a set of red instantaneous fuses, which may be of any number from two to sixteen or more in accordance with the number of holes to be blasted. The instantaneous fuses may be of equal or of different lengths. Between the end of the safety-fuse and the ends of the instantaneous fuses within the tin tube is an explosive disc, which, when fired by the safety-fuse, causes immediate ignition to the whole of the instantaneous fuses. The instantaneous fuse burns at the rate of 150 feet per second, and all the quick fuses explode at the same moment. The mouth of the tin tube is protected by some waterproof substance.

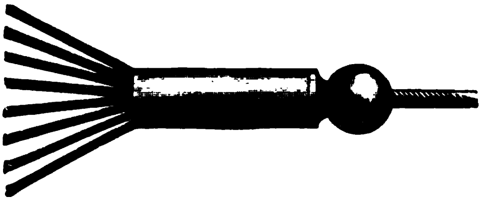


Fig. 604.—BICKFORD'S PATENT VOLLEY-FIRER.

In driving cross-measure drifts, or in sinking shafts, it is often of the utmost importance to fire a number of charges together.

In the workings it is not desirable to fire shots simultaneously, unless this be done at a time when only the shot-firers are in the mine and a long interval be allowed to elapse between the firing and a visit to the positions of the shots. Where a number of shots are fired simultaneously the sound of the explosions is as one, and it is impossible to know whether or not there has been a miss-fire. An immediate return to the site of the shots would therefore be extremely dangerous. If it is required to fire a number of shots in the workings from a particular spot by means of ordinary fuses, these may be of different lengths, so as to allow the shot-firer to count the number of explosions and thus ascertain if there has been a miss-fire.

Fig. 605 shows the volley-firer in the act of firing a number of charges simultaneously.

The patent volley-firers are supplied in various forms and sizes, as required for mining and tunnelling, the two ordinary forms being that shown in Fig. 604 for use in the mine, and a T pattern more especially adapted for quarry or surface work. When required, the inventors supply the volley-firers with looped fuses, so that if the whole length of the fuse so looped is, say, 10 feet, the miner can cut it into lengths of 3 feet and 7 feet, 4 feet and 6 feet, or any proportions of 10 feet without detaching them from the volley-firers, or affecting the simultaneousness of the explosion. Slight differences in the lengths of the special fuse do not disturb this simultaneousness because of the extreme rapidity of burning.

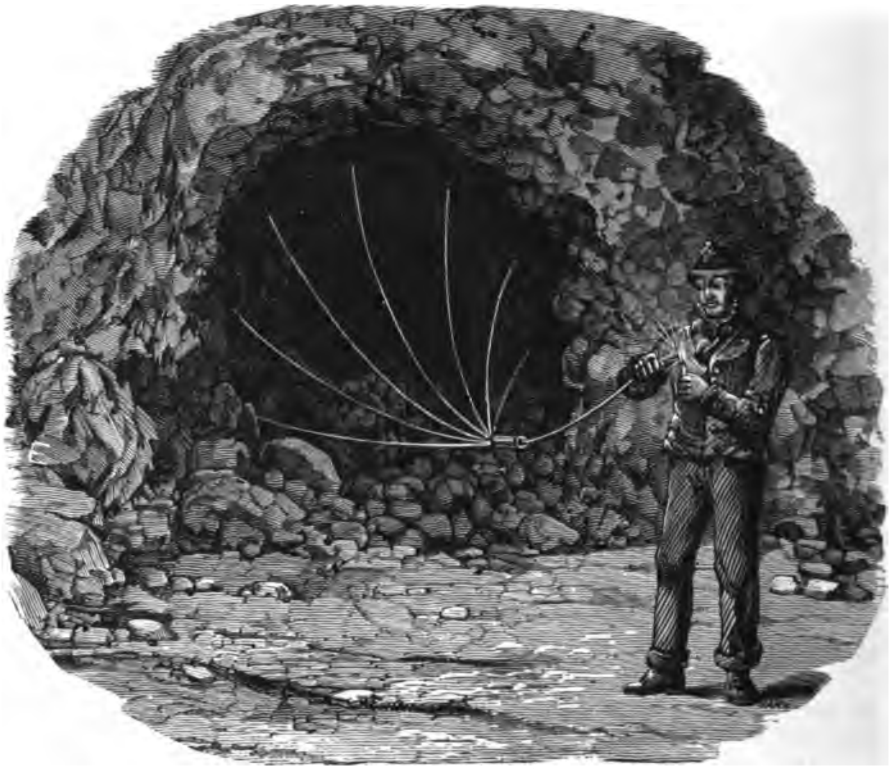


Fig. 605.—The Bickford Volley-Firer in Use.

Explosives are sometimes fired by means of friction matches, but it is doubtful if these are more free from danger than the best fuse.

The friction detonator invented by Lieutenant-Colonel Lauer, of Austria, is thus described in the Report of the French Commission on the use of explosives, 1891 :—

" These detonators consist essentially of a friction match attached to a plugged detonator containing 11·57 grains of fulminate of mercury.

"The friction match consists of a hollow copper cylinder *a c b b c a*, contracted at *c c* (see Fig. 606, double size). In this cylinder is placed another cylinder, *e d d e*, split lengthways, and to which an iron wire *f g h* is fixed. The extremity of the iron wire is flattened and bent for about ·3 inch, between the two cylinders; the part *f g* is also flattened and notched. Below the contracted part *c c* is found the fusee material M M, and below it a little disc of paper *n n* is placed. Beyond this, paper is rolled round the wire to maintain the diameter of the tube, and the whole is covered with a very strong cardboard cylinder at least ·04 inch thick. At the end, the iron wire passes through soft putty; it is bent as shown in the sketch, and is maintained in that position by a paper cylinder *i j j i*. The working is very simple: when the iron wire is pulled the flattened and notched part *f g* is drawn through the fusee material, which takes fire, and the flame, having no outlet behind, is thrown forward and causes the detonation of the fulminate in the detonator.

"The Lauer detonator is attached to the cartridge in the same manner as the fuse, fitted with an ordinary detonator. The stemming is done in the usual way. The end of a very supple cord from 30 to 50 yards long, which has previously been unrolled in the drift is attached to the ring of the iron wire without tearing the paper cylinder *i j j i*. Once attached, the cord is pulled, the iron wire is straightened by tearing the paper cylinder, then the roughened part penetrates the fusee material, ignites it, and the explosion is produced. The manufacturers recommend that the paper cylinder be torn and the wire straightened before attaching the cord, but this practice appeared more dangerous than the preceding. If the drift is crooked, a little guide-pulley is fixed, so that the direction of the pull does not vary too much from that of the shot-hole.

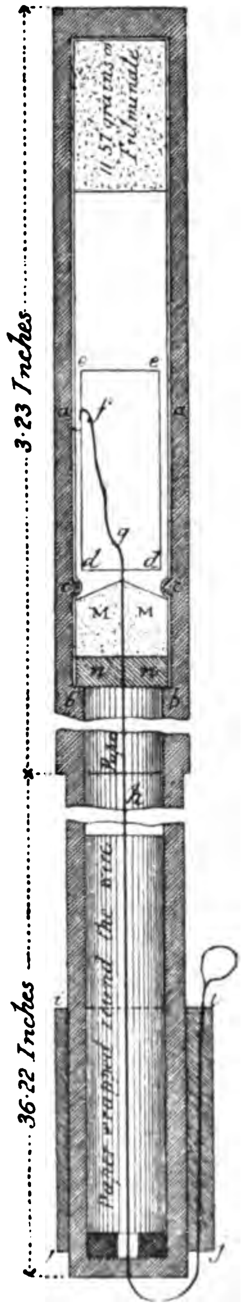


Fig. 606.—The Lauer Detonator.

* * * * * *

"The mode of ignition with the Lauer detonator costs a little more than the safety-fuse method, but on the other hand it costs much less than the electrical methods hitherto used. The Lauer detonator avoids the use of safety fuse, and consequently the danger of ignition of fire-damp by the flame which the fuse projects during the combustion of the portion outside the stemming.

"By combining the use of the Lauer detonator with the new explosives, it appears probable that shots might be fired in fire-damp mixtures without danger. Nevertheless, it is always prudent to suspend firing shots in the presence of gas.

"The Lauer detonator, like electrical firing, permits an immediate return after a missed shot.

"The principal imperfections of this process are as follows:—

"1. A great number of shots cannot be fired so nearly simultaneously as by electricity, for this system is scarcely practicable for firing more than two shots at once.

"2. The Lauer detonators are 40 inches long; they are cumbersome and can only be carried with difficulty in thin seams.

"3. In straight drifts, screens must be erected about 40 yards from the face to avoid projected stones.

"4. It is feared that the manufacture of these detonators will not always be satisfactory. Several collieries in Austrian Silesia, where this system had been adopted throughout, have abandoned it, in consequence of defective supplies (this information is given by the director of the Karwin collieries).

"5. This system possesses certain dangers to which miners are unaccustomed. An unskilful workman may cause the explosion of a shot before he reaches the shelter."

The detonator is sunk into the cartridge, and the stemming is rammed round the tube enclosing the wire until it is embedded in the stemming.

Firing shots by means of electricity, although adding slightly to the cost, is doubtless the safest method that can be adopted. An electric fuze or "exploder" is required. It consists of some explosive compound which is capable of being acted on by an electric current so as to produce an explosion. One method of firing is by inserting in the exploder a fine platinum, iron, or alloyed metal wire, and connecting this with other wires in the circuit of a powerful voltaic battery. The fine wire not having sufficient conducting power offers sufficient resistance to the electric current to heat the wire to redness and thus cause an explosion of the compound in contact with it. Another method is to make a sudden discharge of static electricity take place between the terminals of two wires embedded in the charge. The passage of the sparks between the terminals causes the explosion of the fulminate, which then fires the shot.

An electrical machine for exciting and accumulating electricity, and conducting wires, are also necessary. If it is desired, a large number of holes may be fired simultaneously. The usual practice is for the whole apparatus to be placed in the hands of a competent man, and an assistant, who travel from place to place, and charge and fire the shots if, after the prescribed examination of the place and its vicinity has been made, it is found safe to do so. Some electric shot-firers test the caps before firing them.

In firing by electricity it is of the utmost importance that the wires or "cables" be left unconnected with the battery until everything else is in readiness and the persons present have removed from the position of the shot to a place of safety.

In charging a hole with cartridges of dynamite or similar compounds, one or more being used according to the charge required, each cartridge should be pushed quietly home with a wooden rammer. If safety-fuze is used, a suitable length is taken, and one end,—cleanly cut,—is inserted in a detonator-cap down to the fulminate, and the cap well closed over the fuze by pressing with circular nippers. In wet places, or if water is used in tamping, the junction of the cap and fuze should be well greased. A small cartridge, called a "primer," is then opened for the reception of the detonator-cap. The cap, with the fuze attached, is inserted in the primer to a depth of about three-fourths of its length, and the paper covering securely tied round the fuze with string. The primer and fuze are then ready for being placed on the charge in the hole, after which some loose stemming is put in and the hole stemmed as may be necessary.

Where gunpowder is now used for blasting, it is usually in cartridges, and in damp places the cartridge-paper should be waterproof. If there be water entering the shot-hole through fissures, it should be kept back by clay well rammed into the hole.

Experiments have been made in ordinary blasting to ascertain how far wooden plugs pierced with a hole for the fuze, and 6 or 8 inches long, could be substituted for tamping, but they were not attended with much success. It was found that, in most cases, the gases escaped between the plug and the sides of the shot-hole before ignition was completed, and that the plug was blown out like a projectile.

THE PREVENTION OF FLAME FROM SHOTS.

Where blasting operations are unavoidable in mines which are either fiery or dry and dusty, it has become an absolute necessity to use means whereby flame shall not issue from the shot on its explosion, if the explosive used be one which produces flame.

Before the passing of the Act of 1887, however, efforts had been made to attain this object, amongst the first of which was the method of surrounding the charge by water before it was placed in the hole. This "water-cartridge," as it is called, is the invention of Mr. Miles Settle, and is used in conjunction with Nobel's gelatine-dynamite. When properly manipulated it is an effectual preventive of flame.

The *water-cartridge* consists of a cylindrical case of specially prepared water-tight paper, 18 inches long by 2 inches in diameter. In the centre of this case the explosive is placed and kept there by thin metal webs or diaphragms, the last cartridge or primer having inserted into it the detonator with wires for electric firing. The space between the charge and the paper cartridge is then filled with water, and the outer end firmly tied round the projecting wires. The charge is then ready for being placed in the shot-hole.

The *Gelatine-Dynamite* or *Gelignite* is a nitro-compound containing 60 per cent. of blasting gelatine (consisting of nitro-glycerine and nitrated cotton), 32 per cent. of sodium nitrate, and 8 per cent. of wood ground into fine powder and freed from all resinous matter.

The following information and instructions on the subject have been supplied by Nobel's Explosives Co., Limited, Glasgow:—

"The frequent recurrence of colliery disasters directly due to the use of blasting powder has drawn public attention, more especially in recent years, to the necessity for adopting some other method of coal-getting, and the efforts of many scientists and mining engineers have long been directed towards the attainment of so desirable an object. On the Continent, as well as in this country, Government inquiries have been instituted, and the study of this subject has led to a wonderful development in mechanical means of coal-getting, and in improved systems of blasting, accompanied by the introduction of new explosives, &c.; but, unfortunately, the majority of the inventions brought before the public have tended much more to increase the cost of obtaining coal than to advance the object in view. Among methods other than those which depended solely upon the use of mechanical appliances may be mentioned the hydrogen cartridge of Dr. Kosman, the lime cartridge, the use of water-tamping and moss-tamping with explosives. Other systems, from their obvious impracticability, need not be referred to here, and of those mentioned above, none appear to have made any appreciable headway.

"To Sir Frederick A. Abel, C.B., F.R.S., the celebrated War Department Chemist, the principle of the water-cartridge, in its original form, undoubtedly owes its origin. So far back as 1880 this distinguished scientist conducted experiments in the Garswood Hall Collieries, near Wigan, which clearly demonstrated that dynamite, in conjunction with waterproof envelopes, might be effectively utilised in coal-blasting. These experiments were further pursued under the direction of the Royal Commission on Accidents in Mines, on which Sir F. A. Abel occupied a prominent position; and the results obtained showed that the adoption of a water-cartridge would eventually solve the difficult problem of safety-blasting in fiery mines and in the presence of coal-dust.

"In the meantime, however, two events happened which were destined to have a most important bearing upon the question of safety-blasting. These were the patenting by Mr. Settle of his improved water-cartridge, and the introduction into Great Britain of the gelatine explosives of Mr. Nobel. Mr. Miles Settle, the managing director of the Madeley Coal and Iron Co., Madeley, Staffordshire, and of the Darcy Lever Collieries, Bolton, himself a practical coal worker of wide experience, conceived a method by which the safe use of the water-cartridge might be ensured, and his invention was patented in October, 1882. Meanwhile the gelatine compounds manufactured under the patents of Mr. Alfred Nobel were placed on the market; but it was not until after 1885, when these explosives had been brought into more general use by the introduction of two moderately-priced compounds, known as gelatine-dynamite and gelignite, that the advantages of combining their use with the water-cartridge became strikingly apparent. Among other facts which were brought into prominence by experiments conducted on behalf of the two patentees was the important discovery that the gelatine explosives invented by Mr. Alfred Nobel *possess in themselves* qualities which render these compounds specially suitable for safety-blasting. When used for blasting in the ordinary way they give off flame and sparks to only a very limited extent, and in this respect recommend themselves, even when employed without the water-cartridge, as yielding a degree of safety in coal-blasting to which no other explosive has yet attained. To this valuable recommendation they add the important distinction of being practically impervious to water, which is an advantage that is shared by no other explosive.

"The improved water-cartridge of Mr. Settle consists of a simple arrangement of discs or tin supports (see Figs. 607—610), whereby a most essential advantage is obtained, namely, the fixing of the explosive charge in the water-cartridge in such a manner that it is maintained in a central position so as to be *entirely surrounded by water*, at whatever angle the borehole may be placed. This invention of Mr. Settle obviates the danger which had hitherto existed of the explosive charge occupying a position of close proximity to the outer envelope, and in consequence exposing an unprotected side or end to a "blower" of gas, and thus facilitating the escape of flame and sparks, which cannot be prevented by a mere film of water, if even such were assumed to exist. In proof of the absolute necessity for fixing the explosive charge in a central position, it may be stated that, in numerous carefully-conducted experiments, *flame was observed when the explosive was left unsupported* and allowed to rest on the side or end of the envelope.

"Before acquiring from Mr. Settle his valuable patent, Nobel's Explosives Co., Limited, added still further proof of the reliability of the water-cartridge—if such were needed—by means of an elaborate series of experiments conducted by experienced mining experts in their employment, who submitted the patent gelatine-water-cartridge to crucial tests far exceeding what would ever be required of it in actual use, and the results were in each case absolutely satisfactory. For instance, the experiment of firing the water-cartridge in a barrel containing a mixture of fine gunpowder has been frequently performed by the Company's experts without igniting the powder.

"It will be seen from what we have said that safety and the other advantages which we are justified in claiming for the patent gelatine-water-cartridge do not only rest upon the invention of Mr. Miles Settle, but upon the employment of the gelatine explosives of Mr. Alfred Nobel in conjunction therewith. The joint use of the two inventions is indeed a most fortunate combination.

"The adoption of the improved water-cartridge and the gelatine compounds by Mr. Settle in his Madeley and Darcy Lever collieries, which are undoubtedly among the most fiery in Great Britain, is a sufficient proof of the safety of this method of blasting. In the collieries referred to, many thousands of water-cartridge shots have been fired with absolute success during the past three years. In the Fair Lady Pit, at Leycett, Madeley, where, on 6th January, 1880, a disastrous explosion occurred, caused by a blown-out gunpowder shot, and resulting in the loss of sixty-two lives, no blasting of any kind was allowed until Mr. Settle introduced his water-cartridge. Since the adoption of this system, blasting has been regularly carried on for upwards of fifteen months with perfect safety, and to the entire satisfaction of H.M. Inspectors and of the workmen employed in the pit.

"It may be interesting to add that, between the 1st May, 1886, and the 31st August, 1887, nearly 300,000 shots have been successfully fired with the patent gelatine-water-cartridge.

"As proved by the experiments conducted under the direction of the Royal Commission on Accidents in Mines, all nitro-glycerine compounds do not possess in an equal degree the safety or advantages of gelatine-dynamite and of gelignite, which latter is another form of the same explosive. Dynamite, although it has been employed with the water-cartridge in several experiments with comparative safety, has in other instances shown sparks and flame, and, moreover, possesses the disadvantage, that unless each charge is protected by a waterproof covering it is liable to dangerous exudation of nitro-glycerine owing to the action of water.

"In view of the danger attending the use of dynamite, gun-cotton, tonite, gunpowder, and other explosives with the water-cartridge, Nobel's Explosives Company, Limited, consider they are justified in prohibiting the employment of the Settle water-cartridge in conjunction with any explosive compounds other than Nobel's gelatine-dynamite and gelignite."

The following description of the requisite appliances has also been issued by Nobel's Explosives Company:—

"ACCESSORIES.—*Water-Cartridge Cases* (Fig. 608).—These are manufactured from specially-prepared waterproof material. The water-cartridge case or bag in general use measures 18 inches long by 2 inches in diameter; but several other sizes may, under certain circumstances, be found to be more suitable, and these can be procured on application.

"*Gelatine-Dynamite or Gelignite Cartridges.*—These cartridges constitute the explosive charge, and are supplied of varying lengths, according to the strength of the shot required. Gelatine-dynamite and gelignite, when used in connection with the water-cartridge, may be considered to be about four hundred per cent. stronger than ordinary blasting powder. About 4 oz. of gelatine-dynamite, or about 4 oz. of gelignite, are therefore equal to 1 lb. of compressed blasting powder; but, as the results obtained in water-cartridge blasting are superior in many respects to those obtained by ordinary powder blasting, the regulation of the charge can most readily be determined by experience. No other explosive than these two gelatinous compounds should be employed with the water-cartridge. One cartridge alone should be used, and in Fig. 607 B is shown the method of fixing the electric detonator fuse therein.

"In exceptional cases, where it is absolutely necessary to employ more than one cartridge, they should be connected by means of a wooden skewer (Fig. 611), and squeezed tightly together. Figs. 612, 613, 614, show cartridges so connected. This precaution prevents a film of water lodging between the cartridges, and thus tending to cause a miss-fire.

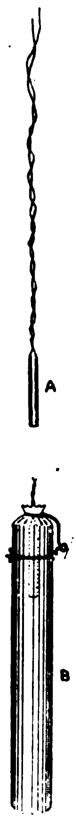


Fig. 607.

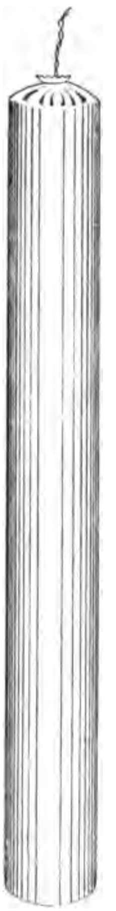


Fig. 608.

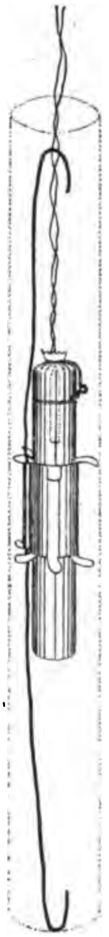


Fig. 609.

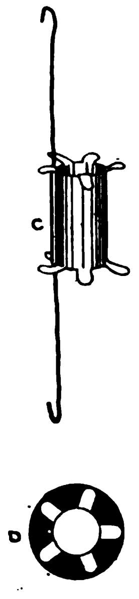


Fig. 610.

SETTLE'S PATENT WATER-CARTRIDGE.

Fig. 607.—A, NOBEL'S ELECTRIC-DETONATOR FUSE. B, CARTRIDGE OF GELATINE-DYNAMITE, OR GELIGNITE, WITH ELECTRIC-DETONATOR FUSE, SHOWING DETONATOR INSERTED OVERHEAD AND FIXED READY FOR PLACING IN TIN SUPPORT C.

Fig. 608.—WATER-CARTRIDGE-CASE, PREPARED FOR INSERTION IN BORE-HOLE.

Fig. 609.—SKELETON WATER-CARTRIDGE-CASE, WITH CHARGE IN POSITION.

Fig. 610.—C and D, TIN SUPPORT.

" The sizes of the cartridges usually employed in water-cartridge blasting are as follow, viz. :—

⅞ inch diameter × 3½ inches long.
1 ,, ,, × 5 ,, ,,
1⅛ ,, ,, × 6 ,, ,,

"*Nobel's Electric Detonator Fuses* (Fig. 607 A).—These are thoroughly reliable, and differ from the fuses of other manufacturers in several respects. They are usually prepared with wires 54 inches long, but, if desired, they can be furnished with wires of various lengths attached. Fig. 607 B shows the electric detonator fuse inserted into the cartridge, the detonator being pressed in overhead and the cartridge-paper tied over it, the twine being then twisted round the cartridge, to keep the whole in position.

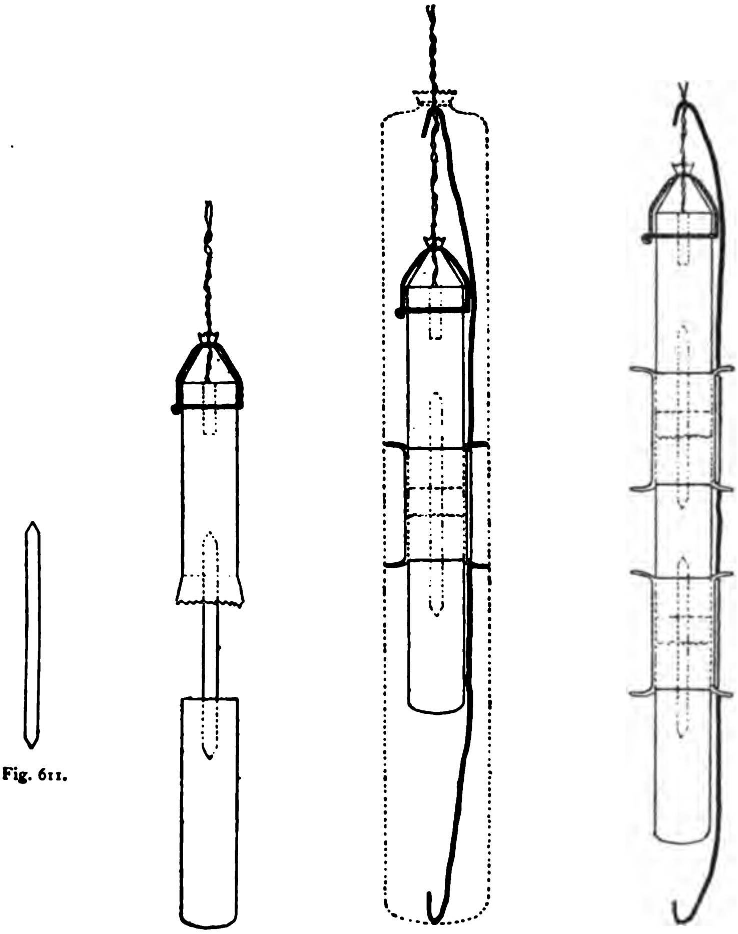


Fig. 611. Fig. 612. Fig. 613. Fig. 614.

SETTLE'S PATENT WATER-CARTRIDGE.

Fig. 611.—WOODEN SKEWER FOR CONNECTING CARTRIDGES TOGETHER.

Figs. 612, 613, 614.—CARTRIDGES CONNECTED TOGETHER TO FORM ONE CHARGE.

"*Supports.*—These are made of tin, and are so formed as to suit any size of cartridge. Fig. 610 C, D show the tin support, and from Fig. 609 the method of fixing the supports on the cartridge of gelatine-dynamite or gelignite will be readily understood."

The following are the permanent appliances which (in addition to the "accessories") are required when Settle's water-cartridges are in use :—

Battery, or Magneto-Exploder (Fig. 617).

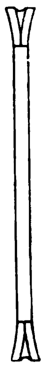


Fig. 615.

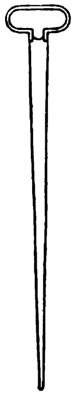


Fig. 616

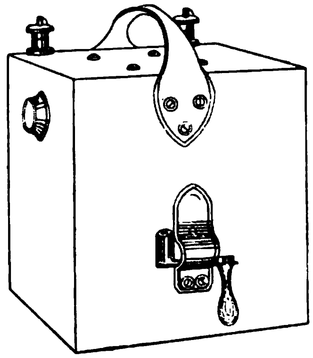


Fig. 617.

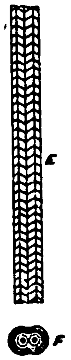


Fig. 618.

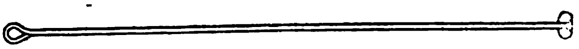


Fig. 619.

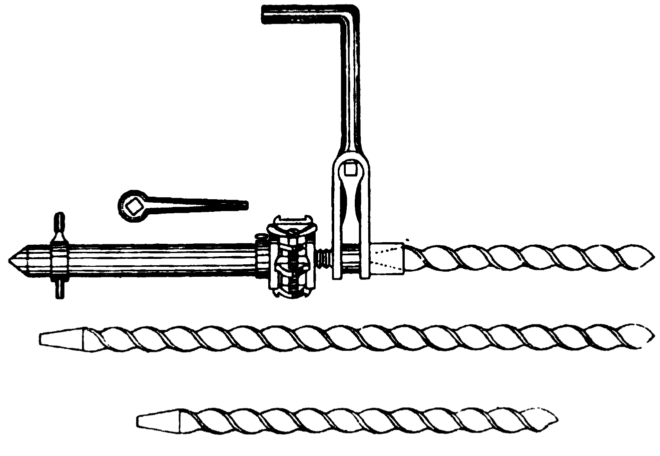


Fig. 620.

SETTLE'S PATENT WATER-CARTRIDGE APPLIANCES.

Fig. 615.—WOODEN TAMPING ROD.
Fig. 616.—FUSE PROTECTOR.
Fig. 617.—ELECTRIC BATTERY.
Fig. 618.—E and F, INSULATED CABLE.
Fig. 619.—SCRAPER.
Fig. 620.—MECHANICAL DRILL.

Insulated Cable (Fig. 618, E F).—Supplied with box and reel if desired.

Fuse Protector (Fig. 616).—A tapered hollow rod of Muntz metal for preserving the fuse wires from abrasion during the process of tamping.

Mechanical Drill (Fig. 620).—This machine is very strongly constructed, and is specially adapted for preparing bore-holes for the water-cartridge. It is most useful for coal, soft rock, or in moderately hard ground.

The feed of the mechanical drill is automatic, and so arranged that the screw may be instantly withdrawn to change drills. The standard is so constructed that it can be extended from its shortest length to within about one foot of double that length. All the parts of this machine which are required to withstand strain are made of steel, thus ensuring the greatest possible strength combined with lightness.

For *very hard rock* the Elliott drill should be used, and it will be found to bore any material capable of being penetrated by a rotary drill.

Tamping Rod (Fig. 615).—These are made of strong, tough wood, with ends pointed of Muntz metal.

Scraper (Fig. 619) is a rod of Muntz metal, having a flat disc of the same metal at one end, which is found of much use in clearing loose stone, &c., out of the bore-hole prior to inserting the water-cartridge.

The possibility of the water getting out of the cartridge before a shot could be fired suggested a means whereby this might be overcome, while the benefit of the water was still retained, and this was attained by adding a proportion of soap or other gelatinous substance. In this way, the cartridges are surrounded by a liquid which shortly "sets," and there is no danger of its getting out.

Another method is to use common sand moistened with water, the cartridges forming the charge being placed in the centre of the paper covering, which should be large enough to admit of a thickness of half an inch of sand all round. The paper cartridge for holding the wet sand should be made water-proof by soaking in oil or paraffin, and then dried.

Tamping the charge with wet moss is another means used for preventing the issue of flame, and appears to be fairly effective when used with gelatine-dynamite as the explosive.

SAFETY EXPLOSIVES.

Seven of the new explosives are said to be flameless, and to provide in their composition all the advantages sought to be secured by water-cartridges, viz., roburite, carbonite and securite. The first, *Roburite*, has been tried and stood the test of exhaustive experiment in this country. It is the invention of Dr. Carl Roth, of Berlin. In appearance, it is like brown sugar, and if placed on a bright coal fire it burns away, but not vigorously. It is about three times as powerful as gunpowder, weight for weight, but about three times the cost. Roburite is a mixture of two compounds, which are kept separate until the employment of the explosive is required. They consist of 7 parts of nitrate of ammonium and 1 part of chlorinated dinitro-benzol. Each of these compounds is harmless in itself, is inexplosive, and does not easily catch fire. The mixture of the compounds is done at central depôts, from which consumers are supplied. As showing how safe it is to handle after mixing, a light, if applied to it, does not produce explosion, but causes it to burn with moderate intensity. Neither friction nor percussion causes it to explode, and it is only when primed by a specially strong detonator that it *can* be exploded.

Cases are reported in which workmen have been injuriously affected by the fumes, or by handling the roburite. It is not, however, clearly proved that any injury to health has been caused by the fumes, and the precise composition of these, or the gases evolved consequent on the explosion of roburite, is a point which requires clearing up. With regard to the handling, this difficulty may be met by not allowing the roburite to come into contact with any part of the person, to prevent which the cartridges may be made up into any suitable size before being sent to the pit. The Roburite Company issue full instructions, and, if these be strictly followed, injury to the workmen from handling will not arise. A smaller quantity of roburite is required for a given work than of some other explosives, a shot-hole 1½ inches in diameter doing the same work as a 2½-inches water cartridge, and thus effecting a saving in expense. With roburite the flame is quenched by the gases resulting from the explosion, the gases not supporting combustion.

In the beginning of the year 1889 two medical practitioners and the professor of chemistry in Owens College were appointed to investigate the effects of roburite on the health of the workmen in the Park Lane Colliery, Lancashire. As a result of the investigation, the committee came to the conclusion that, with care, the roburite need neither be spilt nor come into contact with the hands of the operator, but in some mines proper precautions are not observed, and cases of undoubted nitro-benzine poisoning have been brought to the notice of the committee, due to improper manipulation of the cartridges. The committee also found that roburite, when properly confined, undergoes complete combustion, leaving no trace of nitro-benzine derivatives unburned, but that there is a chance of incomplete combustion occurring, owing to the explosive not meeting with sufficient resistance.

The following are the general conclusions and recommendations of the committee:—"That although roburite itself is a strong poison, and undoubted cases of poisoning have arisen from the use of it in coal-mines, yet, if stringent care is exercised on the part of the managers, shot-firers, and colliers, the use of roburite will not add to the harmful conditions under which the miner works. As indicating the directions in which additional precautions are in our opinion most necessary, we venture to add the following recommendations to our report:—(1) That the entire manipulation of the cartridges should be entrusted to special shot-firers, who should be instructed in their use. (2) That the effective tamping of the cartridges should be insisted on. Experiments would show by what means the complete combustion of the roburite could be invariably secured. (3) That every care should be taken to ensure the removal of the fumes from the working-faces before the return of the miners, *e.g.*, by continually bringing the brattice cloth up to the working-face. (4) That the products of explosion should be rapidly mixed with a large volume of air. We lay stress on the necessity for the carbonic oxide being diluted with a large quantity of air before it can be breathed with impunity by those who enter the mine."

Bellite is a new explosive, and is said to be as explosive as No. 1 dynamite, although it is not a nitro-glycerine compound. It is the invention of Mr. Carl Lamm, and is composed of four parts by weight of ammonium nitrate, and one part of dinitro-benzol. The powder has a yellowish colour, is almost dry to the touch, and, as it is said to act like the best slow powders, is very useful for coal-mining.

Carbonite is said to resemble gunpowder in its action more than any of the new explosives, except that it is flameless when exploded. It is a nitro-glycerine compound, containing about 25 per cent. of nitro-glycerine absorbed in a mixture of wood meal, and saltpetre, with a little carbonate of sodium, and is made up into cartridges of ⅞ inch or upwards. It is of a brownish colour, and being of a plastic nature it may be pressed into irregularly-shaped bore-holes so as to completely fill them. The cartridges may be stored for a considerable time without losing their efficacy. When struck with a hammer or stone it will not explode, and will only do so by a detonator. The power of the explosive, which is about twice that of gunpowder, is not reduced when placed in wet shot-holes. Many trials have been made with carbonite, and they prove that it is free from flame, even when exploded in coal-dust, under circumstances where the explosion of gunpowder and some nitro-compounds resulted in great flame and violent interior explosion.

Securite is a granulated powder of light yellow colour, with an odour of bitter almonds, and is the invention of Mr. Schoenewez, a German. It is made out of the bye-products of coke-ovens and gas-works, is said to be efficient, and 35 per cent. cheaper than dynamite. It is a mixture of about 17 parts of meta-dynitro benzol, 80 parts of nitrate of ammonium, and 3 parts of oxalate of ammonium.

The products of combustion act as diluents of the oxygen in the air, but, it is stated, are otherwise possessed of no deleterious properties, so that no discomfort arises to miners using the explosive.

It is said to be about four times as forcible as blasting-powder. In wet holes it must be used in waterproof cartridges. It is claimed for it that it cannot be exploded by ordinary concussions or blows, nor by a burning or a glowing body. It is exploded by detonation. It is said to break down the coal in a similar manner to ordinary powder, and to destroy all flame and sparks within the range of the products of its combustion.

Ammonite is a mixture of about 88 parts of nitrate of ammonium and 12 parts of dinitro-naphthalene.

Ardeer powder is composed of about 34 parts of nitro-glycerine, 11 parts of charred kieselguhr, and 55 parts of sulphate of magnesium and nitrate of potassium.

Of these six explosives, carbonite and ardeer powder are nitro-glycerine compounds, the others being nitrate of ammonium compounds. They were experimented with by a committee of the North of England Institute of Mining Engineers and were found to be less liable than blasting powder to ignite inflammable mixtures of air and fire-damp. They do not, however, ensure absolute safety when used at places containing inflammable gaseous mixtures, for all the high explosives produce flame on detonation.

Westphalite is the latest safety explosive which has been introduced to this country, and from the few experiments which have been made with it it has given satisfactory results.

Reports of scientific men in Germany and France, who were appointed to examine into the question of explosives, state that these substances fired in a dangerous mixture of air and fire-damp ignite the mixture only when their temperature of explosion exceeds about 2,200 degrees Centigrade. The temperature of explosion of ordinary gunpowder is about 2,231° C., of nitro-glycerine about 3,170° C., of dynamite 2,940° C., and of cotton 2,636° C. Therefore it is possible for all these substances to produce an explosion of fire-damp. But if other substances can be added to these explosives so as to reduce their temperature of explosion to 2,000° C., or less, their use will be attended with perfect safety. Experiments appear to prove that an equal weight of either the carbonate or the sulphate of soda to dynamite renders the latter incapable of exploding a fiery atmosphere. Similar results follow from mixing a considerable quantity of finely powdered coal-dust with the dynamite. By experimenting further it is probable that suitable substances may be found, which, when added to any compound, will render its use safe.

The use of *Lime cartridges* as a substitute for blasting has been tried within the last few years. The system is patented by Messrs. Sebastian Smith & Moore.

Nearly pure carbonate of lime is used in the preparation of these cartridges. Being ground to a fine powder, it is by means of a specially arranged hydraulic-press subjected to a pressure of 40 tons simultaneously at both ends, by which means its density is nearly doubled. In their preparation a groove is formed on the side of the cartridges about half-an-inch in diameter. In the shot-hole an iron tube, half-an-inch in diameter and having a small groove on the upper side and provided with perforations, is inserted along the whole length of the bore-hole. This tube is enclosed in a bag of calico, which covers the perforations and one end, and has a tap fitted on the other. The cartridges are then inserted and lightly rammed so as to ensure their filling the bore-hole. Tamping is then used as with gunpowder, and a small force-pump being connected with the tap at the end of the tube by means of a short flexible pipe, water, equal in bulk to the quantity of lime used, is forced in. The water is driven to the far end of the shot-hole through the tube, escaping as it passes along at the groove and through the perforations and the calico, flowing towards the tamping into the lime, there saturating the whole of the charge and driving out the air before it. The tap is then closed, so as to prevent the escape of the steam generated by the action of the water on the lime, and the flexible pipe removed. The action of the steam first takes place, cracking the coal away from the roof, and this is followed by the expansive force of the lime.

In a seam of coal, having a good, smooth parting, this means of bringing down the coal after it is holed operates fairly well. If the coal be very strong it is not so successful, neither is it so where the coal is very open or porous, for the steam escapes through the pores and a great part of the force is thus lost. Probably for these reasons its use has not become at all general.

None of the many explosives now manufactured are absolutely trustworthy under all conditions. Those called flameless are certainly a great advance on gunpowder since they produce less flame. Some of them are rarely seen to emit any flame from the shot-holes; when they do so it is only from a large charge. Still the fact remains that it is possible for flame to flash out in firing a shot, and although firedamp is believed to be practically absent at the time, there is an undefined dread that firing may prove disastrous. Continued efforts may end in perfecting some one or more of these already improved explosives, or better ones may yet be introduced. In the meantime those we have should be used with the greatest care and as sparingly as possible, and at a time when only those persons who are absolutely necessary are in the mine.

In December, 1896, an Order was made by the Home Secretary, under Section 6 of the 1896 Amendment to the Coal Mines Regulation Act, 1887, with reference to the use of explosives in coal mines. This Order was revoked by another, dated 4th June, 1897, which modified the restrictions previously imposed. A third Order was issued on the 20th December, 1897, and a fourth Order on the 4th February, 1898, to supersede the third. The third and fourth Orders are the same, but in the latter the list of permitted explosives has been amended by the addition of six explosives. The text of this Order is given in Appendix II., to which the reader is referred. Reference to the official testing of explosives is also there made.

WEDGES FOR COAL GETTING.

Considerable attention has at different times been directed to mechanical wedges of various designs for breaking down coal, and thus to a large extent obviating the necessity for explosives in coal-mines. No doubt these mechanical wedges in their best form have the advantage of an increased percentage of large coal in the output (and therefore a gain in the average selling price), as the use of the best explosives tends to shatter the cohesion of the coal. A great obstacle, however, to their more general use lies in the fact that they are not so easily handled by the workmen as gunpowder, and the yield of coal from them per day is less to the collier.

A form of this machine, known as Burnett's Patent Roller Mining Wedge, is shown in Figs. 621–623. The action of breaking down the coal is by drawing the wedge *e* between the rollers *f*, working against two feathers, *g g*, which have been previously inserted in a bore-hole in the coal. The hole is about four inches in diameter. The wedge *e* is a prolongation of the bar *h*, which terminates in a screw, *k*, of such a length as to project a short distance clear of the bore-hole. The rods, *l l*, form a continuation of the feathers, *g g*, to the outside of the bore-hole. The necessary force to draw the wedge *e* forwards, is obtained by the backward and forward movement of a lever in communication with a ratchet-wheel, to which is attached a nut acting on the screw *k*, not shown in the drawings.

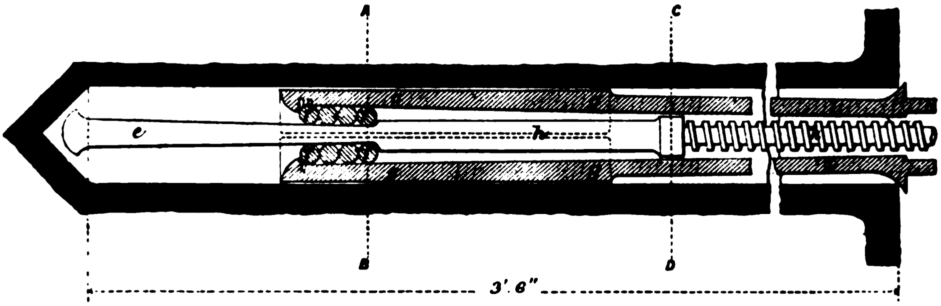


Fig. 621.—Scale $\frac{1}{8}$ full size.

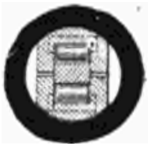


Fig. 622.—Section through A B.

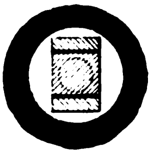


Fig. 623.—Section through C D.

BURNETT'S PATENT ROLLER MINING WEDGE.

The wedge requires a bore-hole to be drilled 4¼ inches in diameter, for a depth of 3 feet 6 inches. The wedge-bar, *e*, projects 8 inches beyond the full diameter of the machine, and only a depth of 2 feet 10 inches of coal is forced down at one operation. The other 8 inches is utilised when preparing the next bore-hole. It is very seldom necessary to draw the wedge quite home, and the process should take place slowly and intermittently to give the coal time to work.

Hydraulic wedges are used much in the same way as mechanical wedges, their rending action resulting from the water pressure brought to bear on the wedge.

ROCK PERFORATORS AND HAND-POWER ROCK DRILLS.

For the most part, drilling the bore-holes for shot-firing or wedging is done by manual labour at the coal-faces in carrying on mining operations. In some cases, the operation is more easily performed by means of a coal perforator than with the usual drill and hammer.

A drawing of Macdermott's Patent Rock and Coal Perforator is shown in Fig. 624. It can be worked by one man, and will drill a hole at the top, bottom, or any intermediate point of the working, and at any required angle. It is very portable, and easily fixed in its working position. By means of the telescopic standard it may be fixed in roadways of varying height.

The standard may be fixed between the floor and the roof, or between any two bearing surfaces. At the upper extremity of the standard is a claw working on a rocking joint, the lower end has a nut fixed inside, on which a screw works. The screw terminates in a point, so that one or two turns of the screw suffice to render it absolutely fixed and rigid by driving the point into the floor and the claws at the other end to a solid bearing, whatever may be the inequality of the surface on which they rest. This constitutes the fixed portion of the implement. The working portion consists of a metal clip collar, hidden from view in the drawing, which slides along the whole length of the standard. The collar also revolves around the standard and can be clamped at any required point. To this is attached by a very strong conical joint the malleable iron box shown on the drawing, enclosing the mechanism for giving an automatic feed to the screw. By this means the screw and its attached tool are adapted to the varying hardness of the material to be pierced. Through the box the main or driving screw passes and is furnished at one end with a socket to receive the steel auger for boring the hole and at the other with a ratchet handle for turning the screw.

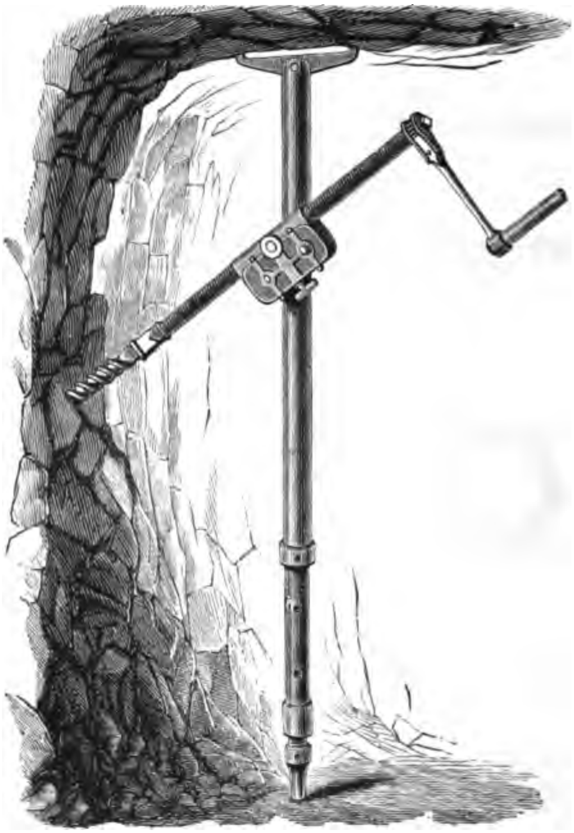


Fig. 624.—Macdermott's Patent Rock and Coal Perforator.

After the standard is fixed in position, the sliding collar is unclamped, an moved up and down to the required position where it is clamped; then, by making the box revolve on the conical joint the auger is adjusted to the angle and point required. A backward and forward movement of the ratchet handle causes the auger to revolve and penetrate the material in contact with it.

Several types of Macdermott's Perforators are made by Messrs. Glover & Hobson, Engineers, Albert Iron Works, St. James's Road, Old Kent Road, London, S.E. No. 1A being specially adapted for boring holes of great depth, such as those made in the neighbourhood of old workings, in advance of the face to test the presence of water. No. 2A is designed to bore holes immediately close to the roof of the mine, or contiguous to the floor. It is arranged so that a man may work the handle at the proper height for his arm whilst the machine bores a hole within one or two inches of the roof or floor.

Messrs. Glover & Hobson thus describe Macdermott's perforators:—

"The Patent Rock and Coal Perforators which M. Macdermott was the first to introduce belong to the class of machine-tools which have done so much of late years to increase and cheapen production in every Art and Industry. They perform, with great economy of labour, one of the most necessary operations in Mining. They will drill a hole, whether at the top, bottom, or any intermediate point of the working, and at any required angle, and are therefore invaluable for ripping down roof or forcing up floor (whether by powder or wedge), boring in advance for water, sinking shafts and making plug-holes in same, driving stone or metal drifts, &c. &c. The uniformity of the hole made by these implements renders it well adapted for cartridges, and gives the maximum effect of the explosive. No blown-out shots need be apprehended. The saving in the quantity of powder where these Perforators are used amounts to about 33 per cent. They are easily carried about, easily adjusted, and easily worked.

"The Patent Rock and Coal Perforators possess advantages which are peculiar to themselves. Firstly, the automatic feed by means of which the same machine can be adapted without alteration to varying hardness of material. Secondly, the adjustment which enables the drill to *be withdrawn instantaneously*, and without any danger of clogging in the hole. Further, they will drill in any required direction equally well; the hole made is uniform throughout its length; any unskilled labourer can use them; one man can carry about, fix, remove after fixing, and work them; and last, though not least of their advantages, they will enable one man to do the work of four—and frequently the work of six—drilling with jumper or hammer and chisel in the ordinary way."

Messrs. Legros, Mayne, Leaver & Co., of 60, Queen Victoria Street, London, E.C., supply the Patent Ingersoll Hand-power Rock Drill, which is specially designed to be worked by hand in places where steam or compressed air is not available, and shown in Fig. 625. It is simple in construction and as nearly as possible resembles the Ingersoll machine-power drill, having a cylinder in which is placed a volute spring (as used for railway buffers) of 200 lbs. power; this is compressed by a bar which is lifted by cams on the crank-wheel shaft, and its release gives the blow; rotation and automatic feed being at the same time provided for. The 200-lb. spring, weight of drill rod, and the momentum produce a very efficient blow, while the form of the spring and other parts render renewals rarely necessary. All working parts are made of steel and the wearing parts hardened. Three sizes of drill are made, one being for a single man-power, another being a 2 man-power and a third suitable for working with 4 men. With a 2-man power machine about 250 blows per minute may be struck, drilling in granite from 1½ to 2 inches per minute.

The subjoined directions are given for the use of the drill:—

"First fix the drill on the tripod (or column) by the centre-bolt on the frame of drill, then set the legs of the tripod with the drill vertical or angled, as may be required. Insert a short drill-bit into the end of working rod, and secure it by tightening the screw of the clip. Loosen the nut on the top of feed-screw, and wind the drill down until the drill bit presses on the stone, then tighten the nut on top of feed-screw and commence working, going on until the bit is deep enough, or as far down as possible, then loosen the nut on top of feed-screw and wind back the drill to the highest point, when the drill bit can be taken out of its socket and a longer one inserted for drilling deeper holes, this being repeated until the necessary depth is bored.

"The automatic-feed is worked by the conical part on top of the working rod, and this cone should be under the top edge of cylinder cover when the drill is down on the rock. (Note.—The nut on the top of the feed-screw must be screwed down tight before the drill will feed automatically.)

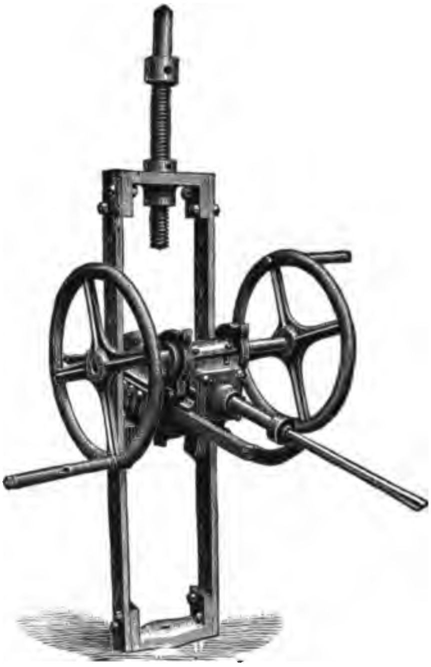


Fig. 625.—Ingersoll Hand-power Rock Drill.

The shaft and cams may be removed by taking the four screws out of the cap in front of shaft. The working rod and actuating spring may be removed by taking the nuts off the bottom cover of cylinder. The wearing parts of crossbar may be renewed by turning the crossbar round back to front, and again by turning crossbar upside down (to do this take off the spring and unscrew collar nut, turning crossbar top to bottom to replace), and again by turning crossbar back to front, thus utilising four different wearing surfaces. The working rod and the cams should be kept well lubricated.

MACHINE ROCK DRILLS.

The necessity for driving headings, or for sinking shafts rapidly, calls into requisition machine rock drills as a substitute for hand labour in boring holes for shot-firing. In various forms these operate on the rock with a percussive action, as in the ordinary hand drill.

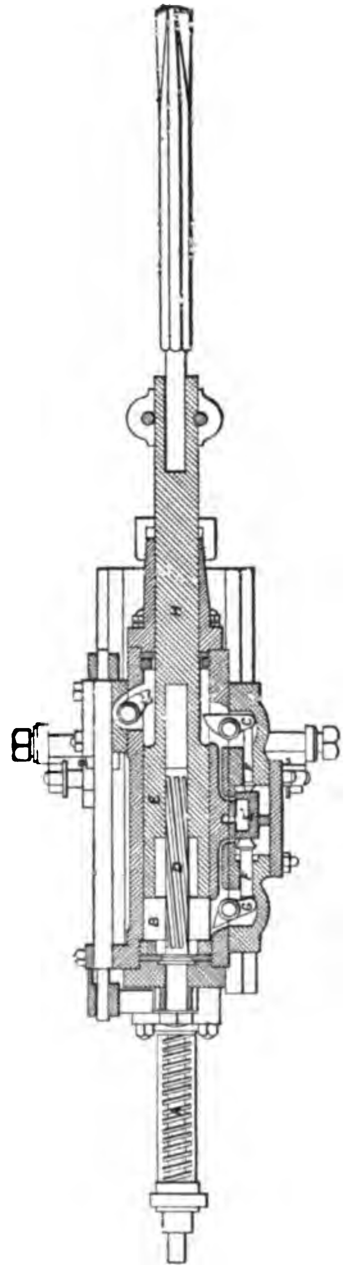


Fig. 626.

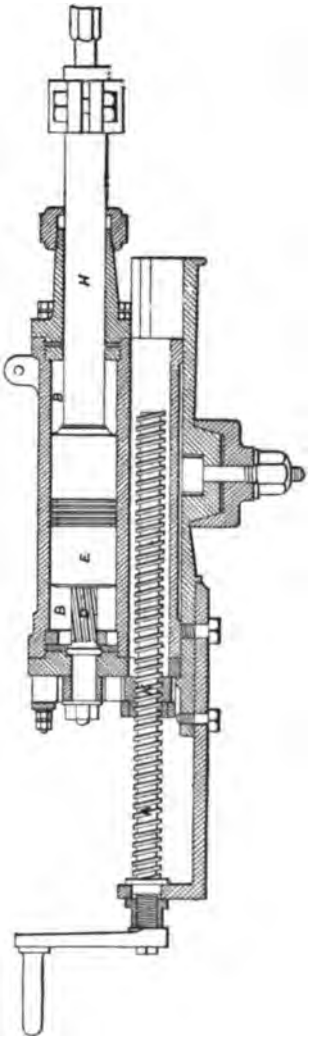


Fig. 627.

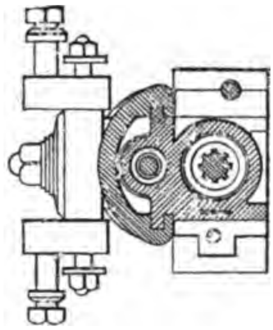
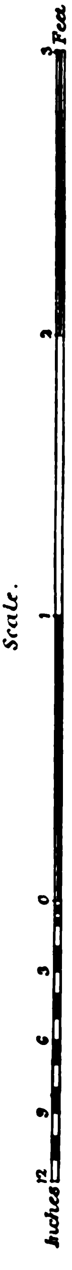


Fig. 628.



Figs. 626-628.—The Ingersoll Machine-power Rock Drill as formerly made.

One of the best is the Ingersoll Machine-power Rock Drill, shown in the drawings, Figs. 626–628. It consists of a cylinder B, in which works the piston E, H being the piston rod. The valve-gear is actuated by the piston through the medium of tappets. It consists of a slide valve E′, two valve spindles F, F, and the two tappet levers C, C.

The piston, as it moves backwards and forwards, alternately encounters and depresses the limbs of the tappets which project into the cylinder, as shown on the drawings; and as these limbs move simultaneously, the motion thus communicated is transmitted by the valve spindles to the slide valve. The tappets are so adjusted relatively to the stroke of the piston as to admit of a cushion (of steam or air) being formed at either end of the cylinder; this cushion prevents injury to the cylinder covers and preserves the drill from excessive vibration.

A spirally grooved bar D, recessed into the back end of the piston, gives a rotary motion to the drill. The piston has a cap screwed into it fitted with studs to run in the grooves of the bar.

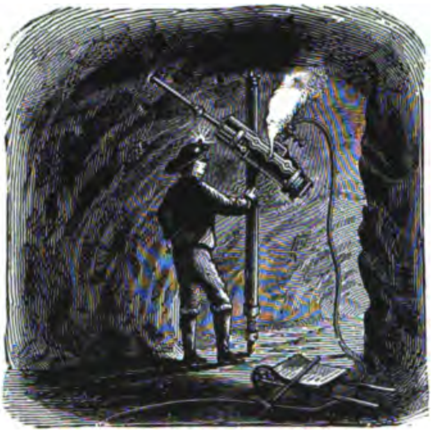


Fig. 629.—The Ingersoll Rock-Drill in Work.

A ratchet-wheel is fixed to the end of the bar, and into its teeth a pawl is held by a spring. By means of the pawl, the piston is compelled to turn during the back stroke, whilst allowing the spiral bar to rotate during the forward stroke. Besides the automatic rotary motion of the drill now described the feed motion in the Ingersoll rock drill is produced automatically, so that as the borehole advances the cylinder equally moves forward and there is no diminution in the force with which the drill strikes the rock. As the drill penetrates the rock, the piston approaches the forward end of the cylinder, and strikes against a tappet lever C′ which partly rotates the rod on which it is carried. By means of pawls and ratchet teeth, this rod turns a nut upon the back end of the cylinder. The feed screw A passes through this nut, which in the act of rotating upon the fixed feed screw causes the cylinder to advance.

The drills used are of different sizes, according to the work to be done. A 2½-inch cylinder drill is capable of boring a hole from 1 inch to 1¾ inches in diameter 8 feet deep. The motive power may be either steam or compressed air, the usual working pressure being 45 lbs. per square inch.

For shaft-sinking the drill may be carried on a tripod, or on a special shaft-sinking frame. For driving headings it may be supported on a stretcher-bar, as shown in Fig. 629. This is a wrought-iron hollow tube, provided at one end with a claw, and at the other a strong adjusting screw lock and nut for fixing it in position. For more important tunnel work the apparatus is mounted on a tunnel-car, specially designed for the purpose. This carries four drills.

The rate of progress by rock-boring machinery is from 3 to 12 times greater than by hand-labour, according to the hardness of the rock.

The only objection we have heard against the above described form of the Ingersoll rock drill is that when heavily driven, the tappets are apt to break, followed almost certainly by injury to the cylinder through the broken pieces jambing the piston against the walls of the cylinder.

Since the foregoing description and remarks were written, a new type of Ingersoll machine-power drill has been devised. In it the use of tappets is dispensed with and also the stems which acted in conjunction with them to move the valve across its face. The valve is now formed by a segment of a circle, and is brought in direct contact with the piston, where it acts in the place of a pair of tappets by receiving a gentle push from the inclines on the piston, instead of the blow formerly given by the piston to the tappets, and thus admits the motive-fluid fore and aft alternately into the cylinder, while at the same time the fluid keeps it firmly up against the face.

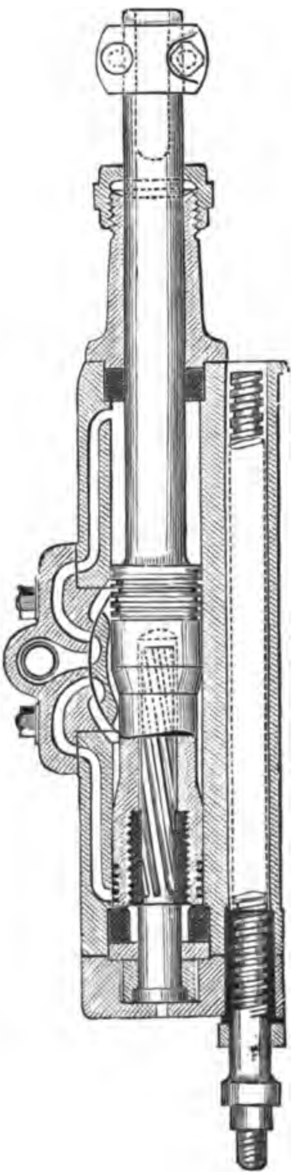


Fig. 630.

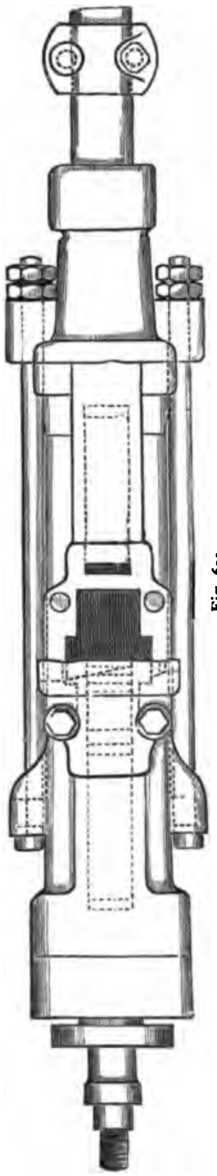


Fig. 631.

The Ingersoll Machine-power Rock Drill as at present made (1890).

The following particulars are supplied by Messrs. Legros, Mayne, Leaver & Co.:—

" Fig. 630 shows a section, and Fig. 631 a plan of this new patent drill, which has been designed specially with a view to reduction of cost, and simplicity, durability, and economy in working, combined with increased efficiency in operation.

"This new patent is of the direct-acting percussion class of drill, and has the following distinguishing features and advantages:—

"1. Its Valve Motion.—This is entirely unique, and while retaining a positive motion of the valve (a most important feature), all moving parts between the piston and the valve, such as tappets, valve-stems, levers, &c., are entirely done away with, the valve itself being the only moving part remaining, as will be understood from the illustration and the following description.

"The piston has two fitting portions, divided by a recessed, or smaller portion, having gentle inclined planes at either end; this space round the centre portion of the piston forms the chamber into which the steam or air is first admitted. The valve is over this central portion, and is in form a segment of a circle, about (in the 3-inch drill) 4 inches long and 2 inches wide, fitting accurately against a curved face in the valve-chest. The upper side of the valve is provided with recesses properly proportioned, to admit the motive power past the valve to the ends of the piston.

"The steam or air being admitted into the central portion of the cylinder presses the valve closely to its curved face, and the piston being at one end of the stroke as shown, one end of the valve has been raised by the inclined plane, and moved around a portion of the curved face, sufficient to open the passages for the admission of the motive-fluid to the other end of the piston, forcing it along the cylinder, when, as it arrives at the proper position, the valve is again moved round its curve by the other inclined plane, and the steam (or air) admitted to the other end, reversing the motion of the piston and completing the stroke. By this it will be seen that the only moving pieces are the piston and the valve, and the actual results of this simple arrangement are *less wear, less liability to derangement, and the striking of a harder blow with less consumption of motive-power,* the valve being kept firmly up to its seat at any angle or at any speed.

"2. Its Rotation.—The rotation is effected, as in the original Ingersoll, by means of a rifled bar, and ratchet wheel having double pawls, this being the most reliable and positive motion obtainable; but in the new patent drill, the mechanism, instead of being exposed, is entirely enclosed, thus avoiding all chance of damage through dirt, grit, or rough usage.

"3. The New Patent Drill, like the original Ingersoll, can be used, and is equally efficient, in any position, either vertically for drilling upwards or downwards, horizontally, or at any angle, this peculiar arrangement and form of valve keeping in its place, and working well under any conditions.

"4. Provision is specially made in all sizes for taking up the wear in the saddle or slide.

"5. Durability.—In the construction of this drill none but the very best materials are used, and the workmanship is of the highest class, thus ensuring the maximum of durability in a machine subjected to the hard work and rough usage of a rock drill.

"The improved drill, as shown in Figs. 630 and 631, is adapted for hand-feeding, and is suitable for an operator who, if not attending to the feed, would have nothing to do but look on. The same machines, however, are supplied with the automatic feed in an improved form, especially where the drills are used in groups, the number of operators being thereby decreased. The present arrangement for obtaining the automatic-feed is simpler and more efficient than the old one. In it the spindle is still retained, having a tappet keyed to the bottom with a sloping side, which projects into the cylinder in such a way that when the piston slides past it a turn is given to the spindle, which thus disengages a pawl at its top-end from the feed ratchet, and a coiled spring in a box then acts on the pawl, and pushes it into the ratchet again, sending it down the feed-screw. The action is the reverse of the old style, in which the tappet causes the pawl to be thrust sharply into the ratchet, and the spring disengages it, whereas, in the present form, the effect is ensured by a more gentle motion, and with fewer parts, the knuckle-joints, separate tappet, and feed-regulator being dispensed with. It is now intended to extend the side bolts so as to take the back as well as the front cylinder-cover, and in the automatic-feed drills these have to be set slightly on one to allow the spindle to occupy its proper position.

"In the hand-drill shown in Fig. 625 the automatic-feed is obtained by the cone at top of the working rod acting on the feed ratchet by means of a lever, spring, and pawl, the cone in effect taking the place of the tappet in the power-drill."

COAL-CUTTING MACHINERY.

The labour of coal-getting is arduous even under the most favourable circumstances, and in thin seams it is especially severe and trying, owing to the constrained and unnatural attitude of the collier during the time he is at work, and more particularly at the work of "holing."

It is considered allowable, with good and skilful hewing, to take a height not exceeding 9 inches for the holing when carried to a depth of 3 feet.

Where the holing is altogether in the coal, and the seam thin and hard, hand-hewing is placed at a great disadvantage, because the coal obtained as a result of the holing is much less than that obtained from the same depth of holing on thick seams, the labour of holing in which is much less. Coal-cutting machines have been designed, and are in use to some extent, for undercutting the coal. Where circumstances admit of their use they save the collier the heaviest part of his toil. There is, however, great difficulty in the general adoption of coal-cutting machinery, on account of the undulating character of the seams, and the resulting irregularities of roof and thill over very limited areas. These irregularities form great obstacles to the smooth working of coal-cutting machinery, besides which, some districts are so much intersected by faults breaking the continuity of the coal-seams that machinery for getting the coal is quite out of the question. Other coalfields are more favourably situated, and on economical grounds it may be judicious to adopt coal-cutting machines.

The following description of the Gillott & Copley Rotary Coal-Cutting Machine is furnished by Messrs. John Gillott & Son, of Barnsley:—

"The machine described below has now been over 13 years in practical operation, under a variety of circumstances. It is designed for holing or undercutting, and is more especially adapted for collieries worked on the 'Longwall,' or some similar system, where a considerable length of face can be operated upon.

"It is driven by compressed air, and works at the low pressure of from 20 to 30 lbs. per square inch. This is of great importance, as when this pressure is exceeded it is attended with considerable difficulty and a largely increased cost.

"It can be made to cut level with the floor, in a parting between two coals three feet or more above the trams, or at any other height; and is applicable for any seam of coal where a height of not less than 20 inches can be afforded for it to travel along the coal face.

"It will cut in fire clay seating, *hard or soft coal*, or take out a pricking between two coals. It is self-propelling, is made to suit the gauge of any colliery tramway, and travels on the rails as ordinarily laid by the colliers. No fixing is required to keep the machine up to its work.

"The machine is made principally of steel, thus combining the greatest strength with the least weight in the smallest space. It is made in three sizes; in the largest machines the frame, which is of crucible steel, is about 5 feet 6 inches long by 2 feet 4 inches wide, and on this are fixed two cylinders 9 inches in diameter, with a 9-inch stroke, working on to a forged steel crank-shaft, which by a simple arrangement drives the pinion which gears into the slots of the cutter wheel. This wheel, which is of best crucible steel, is carried by a steel bracket, projecting horizontally from the side of the machine. It makes about six revolutions per minute, and on its outer edge or periphery are fixed from twenty to thirty steel cutters, thus giving from 120 to 180 strokes per minute; it is 4 feet 1 inch in diameter, and makes a clean cut of 3 feet 6 inches deep by 3 inches wide, from which it effectually sweeps out the whole of the cuttings as it revolves. The cutter-wheel is easily removed by slackening four bolts, and then the machine can be run to any part of the workings where it may be required. The cutters, which are made of the best tool steel, require less keeping in repair than the ordinary picks, and are about 4 inches long. In the smaller machines the cylinders are

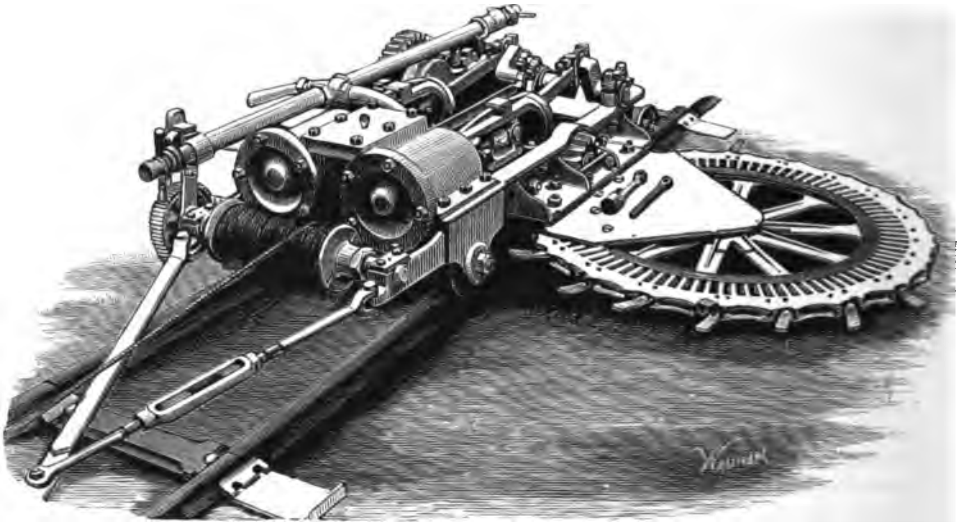


Fig. 632.

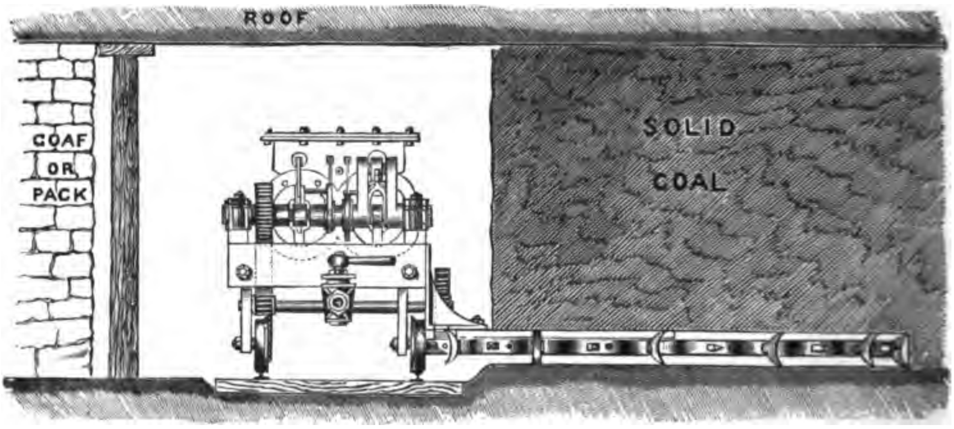


Fig. 633.

THE GILLOTT AND COPLEY ROTARY COAL-CUTTING MACHINE.

7½ inches in diameter, with a 9-inch stroke, and cutter wheel 3 feet 10 inches in diameter, cutting 3 feet 4 inches deep.

"The machine is propelled by a small wire rope fixed to a bridle, passing round a snatch-block at the far end of the 'face,' and on to a drum at the front end of the machine, which is actuated by a ratchet wheel and lever worked by an adjustable crank, so that the feed or pace of the machine can be regulated according to the nature of the material it has to cut. The whole is covered with a moveable sheet-iron casing to protect it from anything falling from the roof. One man only is required to be in attendance on the machine, and another should follow to sprag the coal as it is cut.

"*Working power.*—With a pressure of 27 lbs. per square inch of compressed air, the machine has holed in a hard tough fire clay seating 25½ yards in 40 minutes, and 24 yards 1 foot of strong solid coal in 55 minutes with only 20 lbs. pressure. A fair average rate of work with 27 lbs. pressure may be stated at 20 yards per hour, 3 feet 4 inches under and 3 inches thick, either in a seating or moderately hard coal. The rate of holing by manual labour in the fire clay seating above named is about 7 yards for a day's work—this gives about nine men working a whole day to do what the machine does better in two hours. The men have only to wedge or shoot the coal down and clear it away while the machine is taken to another bank to do its work there.

"Several important improvements have recently been made which enable the machine when used to cut in either direction, and thus work backwards and forwards across the face, obviating the necessity of removing from one end to the other.

"*Compressed air* is now often used for underground hauling and pumping, and when this is the case, the coal cutter may be applied at a trifling cost. In other cases estimates may be obtained for the complete air-compressing plant."

Messrs. Bower, Blackburn and Mori have patented an electrical coal-cutting machine. It consists of a frame travelling on wheels carrying the electric motor, driving a shaft which carries the cutter attached by bolts to a coupling. The motor and the shaft carrying the cutter-bar, can together be rotated in a horizontal plane, so as to bring the cutter in and out of the coal. The motor drives the cutter about 600 revolutions per minute, and develops from 6 to 9 actual horse-power according to the difficulty of cutting the seam.

The machine makes a cutting 3 feet 6 inches deep in the coal, only removing 4 inches in the operation.

A winch is used for drawing the machine along the face, and the under or over cutting is thus carried along the face of the uniform depth stated.

The electric current is produced by a dynamo placed on the surface, and transmitted to the motor in the workings by means of two small cables. Lead-sheathed cables are used in passing through wet places or those subject to foul air. They may be carried on insulators along the roadway or fixed on a wood casing to the roof or floor. At the face it is more convenient to use a double insulated cable, both wires being in one covering. By means of screw connectors this can be attached at any point to the main cable. The position of the main cable can be readily altered.

Including all stoppages, to put up props, &c., 45 yards per hour have been cut with the machine, and in actual cutting time 30 yards have been holed in 36 minutes in hard dirt with pyrites and ironstone lumps. It is claimed for this machine that the form of cutter (which is of special construction) obviates difficulties met with in some other machines, viz., the jambing of the cutter by the falling coal and consequent stoppage.

In a fiery mine the emission of sparks at the electric-motor or elsewhere may be a fatal objection to its use there, unless an efficient covering or protection be devised.

STANLEY'S COAL-HEADING MACHINE.

In carrying on the many operations incidental to coal-mining, circumstances may arise which necessitate long headings or narrow places in the coal being rapidly driven. The use of a machine, such as that designed by Messrs. Stanley Brothers, enables a much greater rate of progress to be attained than is possible by hand-labour. The machine runs on wheels in the heading. Attached to the inner end of a central shaft are a radial and two horizontal arms. Cutters are fixed on the horizontal arms, which, as the central shaft is rotated, cut an annular groove in the coal and form a core within it, which is removed from time to time as the work proceeds. There are two sets of gearing for working the machine. That in front causes the shaft and arms to revolve. This is effected by admitting compressed air to the two cylinders carried on the framework which give motion to a shaft connected with the central shaft by means of cog-wheels, through which the motion is transmitted. The back set of gearing provides for the forward motion of the cutters in operation. The central shaft is threaded so that the back gearing, which consists of a cog-wheel with threaded gun-metal bush fitted into its boss, works on it. The back cog-wheel is secured to the frame, and may be driven by a sliding cog-wheel working on the crank-shaft.

On setting the back gear into motion, the frame advances on the central shaft; when the machine has been moved forward, the back gearing is thrown out, the front gearing thrown in, and the central shaft and arms advance in the frame, at the same time that they are revolving. The machine is provided with two telescopic screw-pins by means of which the frame is held securely in position while the cutters are at work. The usual size of machine weighs two tons, and the cutters form a heading five feet in diameter, at the rate of about three feet in an hour. The arms can only advance between three and four feet; and when out their full length the engines are stopped; the back gear thrown in, and the frame advanced and secured in position again for a further cutting. Different types of the machine are made to suit the varying qualities and classes of coal seams and their thickness.

CAGING APPLIANCES AND DROP STAPLES.

A serious loss of time occurs at collieries in changing tubs where the shafts are deep, if cages having two or more decks are used, and the ropes are lapped on drums in the ordinary way, unless some caging appliance be used. The inconvenience and loss are enhanced if there are intermediate loading stages in the shaft between the surface and the bottom.

In order to avoid having more than one loading stage in the shaft, drop staples are often used for lowering the coal from an upper to a lower seam, or the same object is attained by means of a self-acting inclined-plane driven across the measures between seams of coal.

Drop staples are fitted with guides or conductors precisely the same as the winding shaft. Two single-decked cages are used in the staple by means of which the full tubs are lowered, while the empty tubs are raised. The weight of the full tub or tubs is sufficient to effect the change. A single rope is generally used, which is attached to the upper cage in the usual way, passed over a clip-pulley, and the lower end attached to the cage at the bottom. The clip-pulley is held securely in its place by framework a few feet above the seam. Rails or flat-sheets are laid at the loading and unloading stages to facilitate the entry or discharge of tubs at the cages. The clip-pulley is provided with a brake-ring, by means of a brake acting on which, through levers, the velocity of the cages is controlled and regulated by the upper-stage attendant, and an up and down signal are provided. Both at top and bottom shaft gates are placed for the protection of workmen and others moving about, those at bottom usually being hinged, while those above are lifted by the ascending cage and dropped into place again, when it is lowered in the same way as shaft gates. Where the distance between any loading stage and the bottom of the shaft is great, the expense and loss of power involved will be too great to allow of the expedient of a drop staple, and it is necessary to use the shaft loading stage. In such a case, if the quantity of coal coming from the workings to the shaft bottom is about equal to the quantity coming from the loading stage above, and is likely to remain so, it is usual to make the drums of the winding engine of unequal diameters, so that one cage is raised and lowered on one side of the shaft while the other is lowered and raised on the other side. Thus, one cage is used to raise the coal from the loading stage in the shaft and the other from the bottom. Where the quantity of coal reaching the shaft at one stage is greater than at the other, and in all cases where there are two or more intermediate loading stages in the shaft, great inconvenience in the caging must follow. Loading stages in the shafts, therefore, should be avoided as far as possible.

In the case of a shaft in which coals are only wound from the bottom in a single-decked cage with one or more tubs in it, no inconvenience arises in the change of tubs, which takes place in the most expeditious manner possible, being effected at the top and bottom simultaneously without moving the engine.

With a double-decked cage the tubs in the decks cannot be changed at the same moment in the ordinary arrangement. If the ropes are arranged so that the top deck of the cage at the bottom is exactly level with the flat-sheets or rails there when the bottom deck of the other cage is on a level with the rails at bank, after the change is effected in those decks, on moving the winding engine there will be an unequal travel of the cages owing to the greater length of rope unwound for the surface cage compared with what is wound on to the drum of the other cage. This inequality will be greater or less in accordance with the depth of the shaft, the thickness of rope used, and the kind of drum. When the top deck is lowered into position for changing at the surface the lower deck of the other cage is not raised sufficiently to allow of the change taking place at the pit bottom, and consequently it has to remain stationary, in a position out of the onsetter's reach, while the surface change takes place, after the completion of which it is raised and another stoppage occurs at the engine in order to change the lower deck of the cage at the bottom.

With a treble-decked cage the inconvenience in the change of tubs is increased, as all decks except one must be changed on the surface and underground at different times. Each stoppage of the engine means an additional shock to the ropes, which are thus subjected to greater strains, and so wear out quicker.

Fowler's patent apparatus for loading and unloading pit cages has been designed to overcome these difficulties. In it with a three-decked cage, the lowest tier rests on a level with the bank rails, and two three-decked platforms, one on either side of the cages, are arranged so that the change of the three tubs or rows of tubs takes place at the same time. One of the platforms contains three rows of empty tubs, which are on a level with the decks of the cage when the change takes place. If a single tub is placed in each deck the lowest tub is changed by hand in the usual manner. At the same moment, however, two hydraulic rams push the two upper empty tubs against the full ones in the cage, which are displaced and give way to the empties. The full ones pass into the platform provided for them. The empty tubs are kept in position in the cage by means of catches operated by a single rod.

After the cage has left the surface, the two platforms are lowered by means of hoists, deck by deck in succession, on to the bank level of rails; the full tubs are taken to the screens, and empties placed in the platform arranged for them. Counterbalance-weights raise the full platform after it has been relieved of the full tubs, and the other is also raised with the empties in it ready for the next change.

A similar arrangement is placed at the shaft bottom by which the tubs there are changed at the same time as those at the surface, so that the cages make trips in the shaft as expeditiously as single-decked cages.

Where there are only two decks to the cage, a much simpler appliance is frequently used to facilitate the change of tubs. It consists in placing a platform on the surface above the bank level which receives the full tub or tubs from the upper deck of the cage. After the descent of the cage, the platform is lowered to bank level by a counter-balanced cage which lifts the empty tub or tubs into position.

At the pit bottom the change is effected by means of a balanced platform. It is made large enough for both cages, and is suspended in the sump by means of two ropes, each of which is attached to a drum provided with counterbalance-weights and a brake. After the lower deck has been changed, the onsetter lowers the cage (there being sufficient slack rope for the purpose) till another deck is level with the pit bottom rails, using the brake to do so. After the ascent of the cage the counterbalance-weights raise the platform to the landing, and so place it in position for the next cage. Two or three decked cages may be unloaded and re-charged in this way very advantageously.

PIT HORSES, THEIR FOOD AND WORK.

Questions relating to the care of pit horses, although not directly engineering matters, are yet of great importance to the manager in the economical working of a colliery. Even with an efficient system of mechanical haulage, horses are still required in the remote workings. The aim should be to reduce their number as far as possible and to take the utmost care of those which are necessary. In South Wales, even where all the haulage on main roads is done by mechanical means, there are as many as 100 horses employed at a single colliery.

Horses are usually sent into the pit when four years old; ponies when three; their length of service there depends upon a variety of things, but the average may be taken at about eight years. If heavily worked and badly treated they may not last as many months.

It is hardly necessary to say that much judgment and knowledge are requisite in purchasing horses. The shape and appearance of a horse at rest and in motion afford some indication of his character, to those trained in the study; while the incisor teeth bear marks by means of which the age is ascertained, but unless a manager is versed in the anatomy and the important points of a horse, he had better not trust to his own judgment, but be guided by a qualified veterinary surgeon. In any case, it will be wise for him to buy from a trustworthy dealer, who regularly supplies the district, and will guarantee age, health, soundness of wind and limb, and freedom from vice.

The water given to the horse should be soft and pure, and of moderate temperature, and care must be exercised in timing the supply. When in a heated condition, the horse should not be allowed to drink too freely, nor should he remain too long without water. The diet must be regulated to suit the labour and the conditions under which it is performed. A high temperature, such as that often existing in underground roadways, tends to maintain the heat of its body, and a horse working therein will not require so much nourishment as when doing the same amount of work in a lower temperature. The stomach of the horse is smaller in proportion to its body than that of any other herbivorous animal, and it therefore requires a little food frequently rather than a larger supply at longer intervals. Where at all heavily worked it should be well fed.

The food consists of grain and hay, the daily allowance varying with the work done. As an average it may be taken at about 16 lbs. of grain and 10 lbs. of hay per horse. The grain is for the most part Indian corn, but beans, oats, and peas are given in smaller proportions. The oats should be sound, sweet, a year old, and their natural weight not less than 40 lbs. per bushel. Musty or kiln-dried oats are injurious to horses, and cause colic amongst them, the chief symptoms of which are a tight, dry skin, loss of appetite and debility. The husk of the grain is nearly indigestible, and there is a greater percentage of husk in oats than in the other grain given to horses. The beans and peas should be sound and sweet; they have a heating and binding effect upon the system, and are therefore used in small quantities, or combined with some other article of food which has a counteracting effect, such as bran and maize. Colic in horses is caused by constipation or is accompanied by it; maize has a slightly laxative action on the bowels, and its use reduces the risk of colic to a minimum.

Hay varies in value according to the grasses it contains. If stacked wet or too green, it will ferment. Horses prefer the best seed hay to the best old land hay, and eat more of it, but the latter is more nutritious, and the smaller quantity eaten keeps them in good condition. They do not thrive so well on foreign as on English seed hay, as the seeds are usually thrashed out of the former.

At some collieries it is usual to give green food to the horses for a few weeks in the summer, and it is found to be very beneficial as an alterative and preserver of health. It must be given cautiously, especially for the first time in the season, and should not be allowed at all on the horse-roads or pass-byes. It consists of clover, seeds, and green tares well podded, sent into the pit only on dry days. It is more economical than dry food.

Of late years it has become the practice to use "ensilage," which is fodder stored in a green state. After being cut in the field in the usual way, instead of being dried it is chopped up small and put into pits or "silos," which are bricked and lined with cement. The green food placed in the pit is pressed down by stone flags on the top so as to exclude the air and consequently prevent fermentation. The upper layers are depreciated, but those below remain much the same as when cut. Experiments show that dry food is more economical, beside being more convenient to handle, than ensilage.

As horses age, their grinder-teeth become worn, so that they are unable to properly crush such substances as Indian corn and beans; the grain therefore is partly crushed or bruised, the hay chaffed, and the whole of the ingredients thoroughly well mixed together before the horse is fed with them. The object of chaffing the hay is to prevent waste. When sent down the pit in bundles, a certain amount is damaged or lost in transit, and some is pulled from the racks and trampled under foot. When chaffed it is mixed with the other ingredients forming the provender, placed in bags and sent into the pit, where it is used without waste. Occasionally bran forms a part of the horse-food, and as it acts as a laxative may be most beneficially given at times. In food consumption three 14-hand ponies are usually considered equal to two horses, and one horse costs about ten shillings a week to keep, more or less according to the price of hay, &c.

Some disadvantages of underground horse labour may be mentioned. The number of working days at a colliery seldom exceeds 300 per annum, and is usually rather less. The cost of horses for food and attendance on idle days, such as Sundays, holidays, and those resulting from depression in trade, strikes, accidents, and diseases, adds materially to the cost of production. Horses are subject to epidemics, and it may happen that in times of the greatest prosperity some or all of the horses may be unable to leave the stables. Delays are occasioned by full tubs leaving the rails, when the drivers have to traverse long distances for assistance in order to lift them on again. Other delays arise from horses dropping shoes while at work, when the shoeing-smith must be sent for; if the workings are far from the shaft, a considerable time is taken up before the shoer reaches the horse, during which, and also while the operation of shoeing proceeds, both horse and driver are standing idle. Horses receive injury from falling in the workings, or running away and wedging themselves into roads too low for them, and many are injured or killed by falls of roof, run-away trams, explosions, &c. Besides which there are the injuries inflicted by thoughtless, stupid, or brutal drivers—boys or men.

As to the work performed by horses, much depends upon circumstances. On a favourable road having a slight descent for the full tubs a 14-hand pony may be able to draw 10 tons at a time. But if the inclination is very favourable to the out-going trains, it must necessarily be unfavourable to the in-going. The best result is obtained from horse-labour when the resistance to the trains is equal both ways. If the road is muddy, or the tubs inefficiently greased, or the rails badly laid or worn, a large portion of the energy exerted by a horse is lost in overcoming extra friction, so caused. Again, where a horse works up a steep road. a considerable jerk and strain are necessary to start the load, especially if the back wheels near the face are beyond the rails and rest on a miry or uneven floor. Such strains are very injurious to the animals. Other things diminish the value of a horse's energy, such as a roadway with loose metal, or irregular floor, or one insufficiently ventilated.

The tractive force of a horse working on the surface is usually taken at 125 lbs. when moving at the rate of three miles an hour. If the force exerted by a horse is taken at 33,000 lbs. raised one foot high per minute. maintained throughout a day of eight hours, it amounts to 150 lbs. conveyed a distance of twenty miles, at a speed of two-and-a-half miles an hour. Mr. Nicholas Wood calculated the tractive force of a horse at 120 lbs. when travelling at the rate of from two to three miles an hour, and estimated that it could continue this for ten hours. Allowing for stoppages a speed of from two to three miles an hour = 200 ft. per minute. $120 \times 200 = 24{,}000$ foot-lbs., or say ten tons as the work per minute. Ten tons for twenty miles = 200 tons one mile per day. Mr. Wood takes two-thirds of this as the useful performance, thus $200 \times \frac{2}{3} = 133$ tons conveyed 1 mile a day. This is taken as the maximum effect that a horse can produce, when working on the surface. From experiments made at different collieries it appears that only about one-fourth of this useful effect is obtained from horses working underground. These may be taken to exert a tractive force of about 40 lbs. when moving at the rate of three miles an hour, which may be continued over a period of ten hours inclusive of all stoppages, say twenty miles a day, when the resistance is equal in both directions. If it is necessary to exert a greater force than 40 lbs., the distance travelled must be correspondingly less. If the force be 100 lbs., then $\frac{100}{40} = 2\frac{1}{2}$ times increase; therefore the distance travelled will be $\frac{3}{2\frac{1}{2}} = 1{\cdot}2$ mile an hour. Or we may say the useful performance of a horse underground is to convey about 45 tons one mile per day. Some horses travel from twenty to twenty-five miles in a shift underground, others do not exceed six. The heavier the gradient and the more irregular the roadway, the less will be the distance travelled.

No rules or calculations of this sort can apply generally; they must be taken as mere approximations, for the results obtained will vary in different collieries and districts. It may be useful to compare the actual performances of horses from time to time, and to keep a record for future guidance. Suppose a horse works in a road leading from a shaft the rise of which is 1 in 130. The empty tubs weigh 5 cwt. each and would hold 15 cwt. of coal; the train consists of four tubs, and the friction is $\frac{1}{70}$th. The total weight of the full train is four tons. The assistance obtained from the outward fall is $\frac{4 \times 2{,}240}{130} =$ say 69 lbs. The resistance due to friction is $\frac{4 \times 2{,}240}{70} = 128$ lbs., so that $128 - 69 = 59$ lbs. as the effective resistance to be overcome in moving the full train. The total weight of the empty train is one ton. The resistance resulting from the inclination is $\frac{1 \times 2{,}240}{130}$ = say 17 lbs.; that from friction $\frac{1 \times 2{,}240}{70}$ = 32 lbs.; and the total, therefore, is $32 + 17 = 49$ lbs. The mean resistance of the two trains is $\frac{59+49}{2}$ = 54 lbs. If the length of the horse-road is say 400 yards, and thirty-three journeys are made by the horse in a ten-hour day, then the inward and outward journey will be 800 yards, and $\frac{800 \times 33}{1{,}760} = 15$ miles a day travelled; this gives an average speed for the horse of $\frac{15}{10} = 1{\cdot}5$ mile an hour inclusive of stoppages. $\frac{1{\cdot}5 \times 1{,}760 \times 3}{60} = 132$ feet per minute, and $54 \times 132 = 7{,}128$ foot-lbs., or 3·182 tons for fifteen miles = $3{\cdot}182 \times 15$, or 47·73 tons conveyed one mile a day.

The kind and the size of horse purchased for colliery working will of course depend upon the specific work for which it is required, the nature of the roadways, size of tubs, &c., &c., and in order to prevent the risk of importing infectious diseases into the underground stud, and to ensure new horses being well up to their work, they should be employed on the surface for three or four weeks before being sent into the pit.

Horses are lowered down the shaft by the cages if these are large enough; but if not the animals are securely fastened by a net and suspended below the cage.

At some collieries in South Wales no horses are stabled underground. Where the number employed in a colliery is small, and the cages large, it is found advantageous to lower and raise the pit horses daily, the time occupied in so doing being but trifling. After the day's work is finished each haulier takes his horse up the shaft and away to the stables on the surface. Before commencing work each haulier takes his horse to the pit and down the shaft again.

Generally, however, horses are stabled underground, and many that are taken down never see the daylight again. Usually, the most advantageous place for the stables is near the upcast shaft, and they may be made all together or in a series, according to the number of horses and the state of the strata. The proper ventilation of the stalls is of great importance, the current of foul air passing direct to the upcast. Where the workings are at a considerable distance from the shaft, stables are often made in the interior of the mine so as to avoid unnecessary travelling. An objection to this arrangement, however, is that splits of fresh air must be taken off the main current for their ventilation, thus reducing the quantity for the workings generally. The floors of the stables should be laid with blocks or planks of wood; a branch channel formed in each stall; the floor sloped slightly towards the channel from each side, and the main and branch channels given a definite inclination, so that all liquids will run to a given point. Sawdust is often spread over the floor for the horses to lie on, straw being an expensive litter. In South Wales ferns grow in abundance on the hills close to the collieries; these are cut and when dried make a very good litter much used in the district. It is well by such means to encourage horses to lie down, for it has been ascertained that on brick floors only 10 per cent. of the horses lie on them, 20 per cent. on cemented floors, while the proportion rises to 70 per cent. on cemented floors if thickly covered with sawdust, and 70 per cent. on a wooden floor. Horses which never lie down are, in a few years, seized with numbness of the limbs, and consequently fall down when at their work.

Mangers should be not less than 2 feet by 1½ feet, and 15 inches deep. A strip of wood two or three inches wide may be placed at the top projecting inwards so as to prevent the fodder from being thrown out. A strip of hoop-iron may be fixed to the top of the manger so as to prevent injury from the horses' teeth. Iron

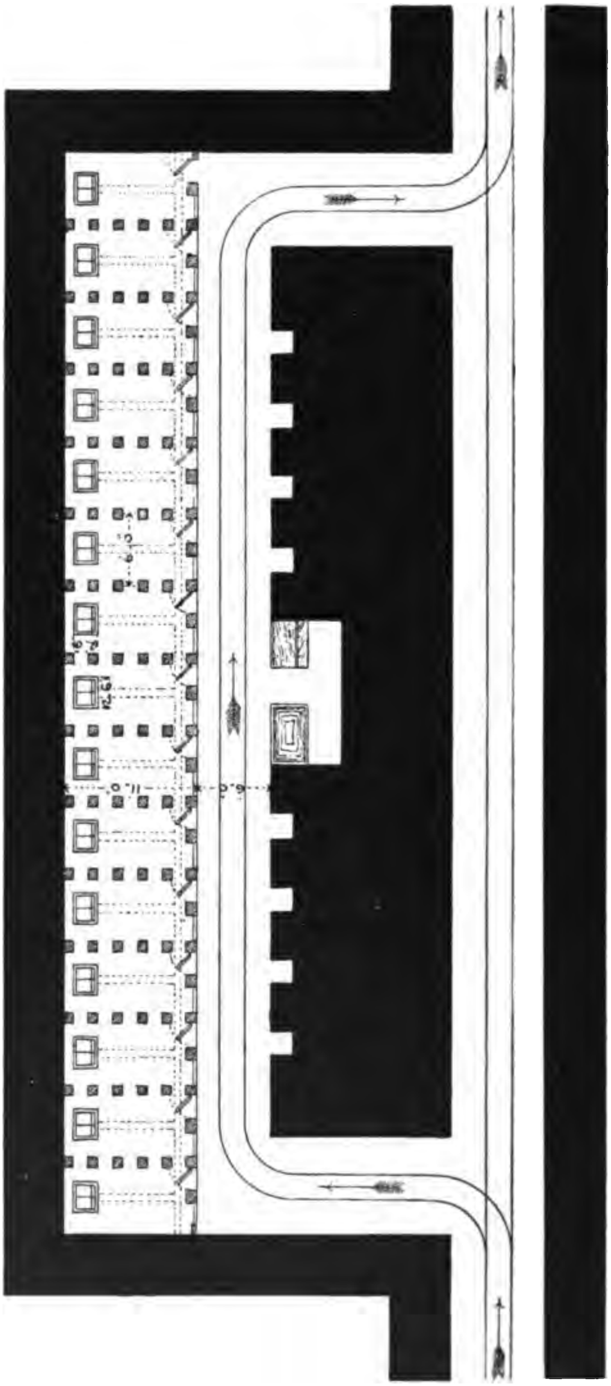


Fig. 634.—Plan of Underground Stables.

Scale 16 feet to 1 inch.

mangers are sometimes adopted, and iron props for the roof and for stall divisions; and occasionally brick mangers resting on stone bases 18 inches high and lined inside with cement. These are more generally built in stables near the shaft, while the iron ones are more applicable to those in-bye.

In the matter of cost, while avoiding extravagance on the one hand and meagreness or faulty design on the other, comfortable stables may be made at about £2 per stall. The site selected should have a good fall for drainage and have an opening not less than 17 feet wide. This will allow for a 6-foot roadway in which to lay a tramway along the whole length of the stables, leaving 11 feet from the gates to the head of the stalls. Stalls should be 6 feet wide, separated by props which secure the roof and form a sufficient partition without being boarded up.

Fig. 634 shows a plan of underground stables made in this way which requires no further explanation.

Stables have, unfortunately, at times been made with an utter disregard to their proper ventilation. The high temperature arising from a number of horses in a confined stable soon produces an almost intolerable stench which is highly detrimental to the health of the horses. Stables of this sort soon become also a harbour for beetles, the ova being brought down with the corn. The congenial warmth causes so rapid a multiplication of the insects that they speedily cover the ground and the mangers, and are consumed with the fodder. Rats also become a nuisance even in well-ventilated stables, where their number is allowed to increase without check. The increase may be prevented in a variety of ways.

Great attention must be paid to the shoeing of horses, the shoe being shaped to fit the horse's foot, and not *vice versâ*. Improperly shod horses worked on hard ground soon suffer from bad feet. The harness must fit easily and comfortably on the horse or he will receive injury from its chafing. Tight or rough collars rub the skin till a wound is formed. By the Mines Act, 1887, every travelling road on which a horse is used shall be of sufficient dimensions for the animal to pass without rubbing against the roof or timbering. All projecting stones and angular corners must be removed from the sides, so as to prevent injury to the animals.

The number of horses under one ostler or keeper should not exceed ten, and of ponies fifteen. Each keeper grooms and feeds his horses, and cleans out the stables. Besides receiving the attention of keepers, a pit's horses are generally under the periodical supervision of a farrier. There should be sufficient horses at a colliery to supply the daily requirements without having to work them double shifts, for systematic overworking is not only inhuman but also (happily) very costly.

In some districts it is customary to work the horses in traces: in others limbers are used, which give considerable holding-back power not to be had from traces. There can be no objection to traces on a level or nearly level roadway, and the harness is of course lighter without limbers. Where trace-horses take full tubs downhill, sprags are often placed in the wheels so as to act as a drag. Over very steep portions of roadway a sprag in each wheel may not be sufficient to prevent a speed which would be dangerous to the horse in front, and where this is so, besides the sprags, additional means are used so as to reduce the speed. In Somersetshire, *e.g.*, this is secured by passing a chain round a road-post at the top of the steep; the roadway being straight allows of the chain working between the rails. The lower end of the chain is at the foot of the steep when the upper end is attached to the full tub. Friction is caused by the chain dragging along the floor and in passing round the road-post as the tub is brought down the steep.

We may add that both mules and donkeys have been used underground instead of horses, especially where the roadways are level and low, and the tubs small. They are both, however, difficult to manage, especially donkeys, owing to their proverbial obstinacy, and are little used, if at all, in this country; but in the mines of the United States mules are extensively employed.

FLEUSS APPARATUS FOR BREATHING IN NOXIOUS GASES.

The Fleuss apparatus for breathing in noxious gases is a most useful appliance, which enables persons to remain under water or in vitiated air for limited periods of time.[*] It is shown in Figs. 635–652. Figs. 635 and 636 show in front and side view respectively the mask portion of the apparatus, by which the nose, mouth and ears of the person wearing it can be shut off from the surrounding air or gases. The mask is fitted with two pipes, A and B, one for the inlet of purified air to the interior of the mask, the other forming a passage for conducting the exhaled vitiated air to the purifying apparatus.

To secure the mask on the face, straps are used at the back of the head and also a bandage which is passed over the mask from C to D, and fastened under the wearer's chin (Fig. 636). In masks of more recent make, instead of the bandage, an india-rubber air-pipe is used, which, when in position round the mask, adapts itself to all the irregularities of the face and effectually makes a sufficiently tight joint. It is better that the person wearing the mask be without a beard.

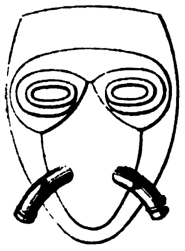


Fig. 635.

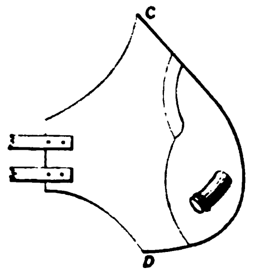


Fig. 636.

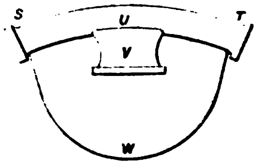


Fig. 637.

FLEUSS APPARATUS.

Even to a beardless person the mask is not a comfortable arrangement, and instead of it, the "goggles" used by Denayrouse to protect the eyes may be substituted together with the apparatus shown in Fig. 637. It consists of a tube U, kept in position on the wearer by the band W. The two ends of the tube, S, T, serve the purpose of the tubes A and B in the mask. The whole face-piece is held by grasping the flat projection V by the lips and teeth. When using the mask the wearer is inconvenienced by perspiration and water which flood the mask after using it some time, and when using the more simple breathing appliance it is accompanied by an emission of saliva which is difficult for the wearer to get rid of.

The nose is in communication with the inlet tube and the mouth with the other, so that respiration proceeds by inhaling through the nose and exhaling through the mouth.

The apparatus for purifying the air is arranged in the form of a knapsack, and can be adjusted to the wearer in five seconds. Figs. 638, 639, 640, 641, 642, 643 and 644 show this. At the bottom is a strong metallic vessel E, about 6 inches in diameter and 16 inches long. This vessel is charged with compressed oxygen at a pressure of 250 lbs. to the square inch. Immediately above this vessel is a rectangular-shaped metallic case H. A vulcanite vessel M is fitted into the case H, as vulcanite resists the action of caustic soda.

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[*] See Transactions, North of England Institute of Mining Engineers, vol. xxxi., pp. 197--203.

This vessel (see Figs. 641, 642, and 643) has a perforated false bottom, and is divided into compartments by division plates. The central division plate extends from the bottom of the vessel to within a short distance of the top, whilst the other two divisional plates, one on either side of the central one, reach from the perforated false bottom to the top of the vessel. When made ready for use, the

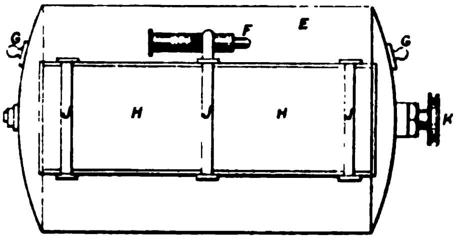

Fig. 638.

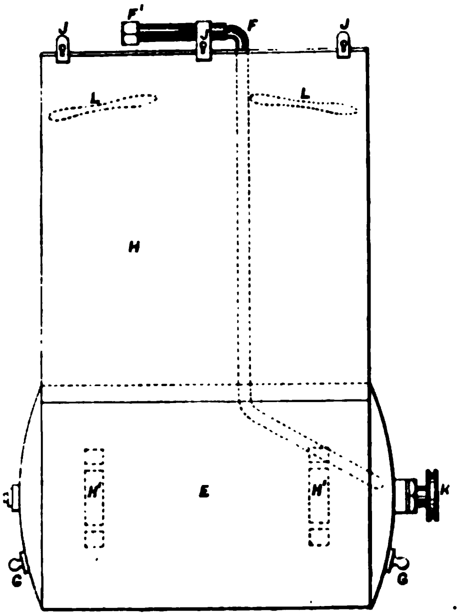

Fig. 639.

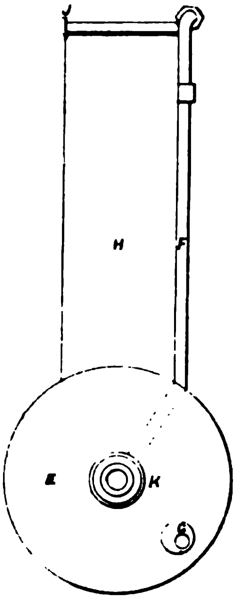

Fig. 640.

FLEUSS APPARATUS.

compartments of this vessel are filled with tow and caustic stick soda. The vessel is provided with a lid, made air-tight by an india-rubber washer placed between the lid and the vessel. Two pipes pass from the lid, the one marked N, Fig. 644, forming a passage through which the exhaled vitiated air is led into the end compartment, while by the other marked O, the air after coursing upward and downward through the compartments of the vessel M can pass back to the interior of the mask or breathing apparatus to be again inhaled.

The pipe O is made with a branch pipe O', standing out from it, from the end

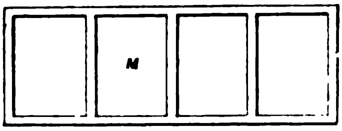

Fig. 641.

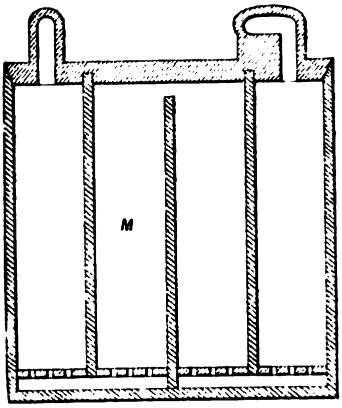

Fig. 642.

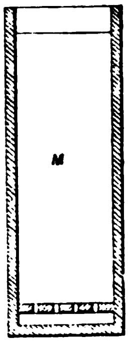

Fig. 643.

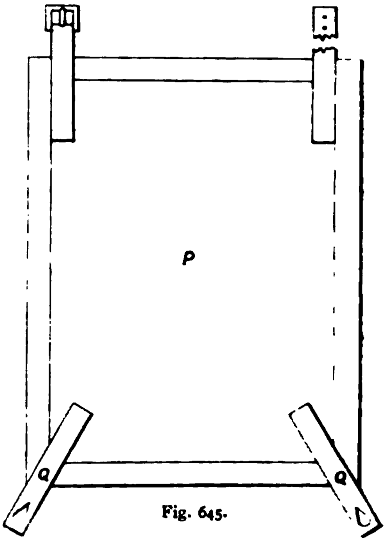

Fig. 645.

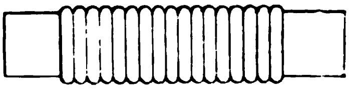

Fig. 645A.

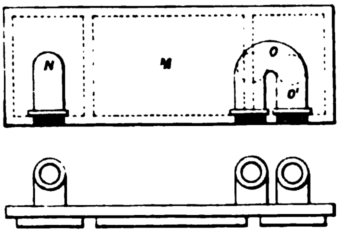

Fig. 644.

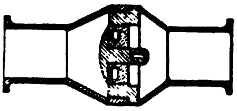

Fig. 646.

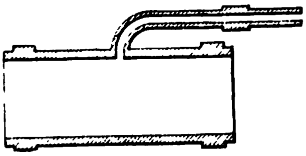

Fig. 647.

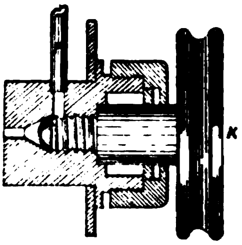

Fig. 648.

Fleuss Apparatus.

of which a flexible pipe is led to the interior of an air-bag P, Fig. 645. This bag acts as a flexible air reservoir, which expands when air is exhaled and contracts when air is again drawn from it into the lungs.

The inlet and outlet tubes on the mask are connected respectively to the inlet and outlet tubes on the lid of the vessel M by elastic tubes of india-rubber. Each elastic tube is provided at one end with a metal valve (see Fig. 646). The inhaling valve opens towards the mask, the exhaling valve away from it. Each elastic tube is formed with corrugations, as shown in Fig. 645A, so that it may readily stretch. The ends of the short tubes on the mask, as well as those on the lid of the vessel M and the ends of the valve-pieces, have each a projecting flange around them, so that when the ends of the connecting elastic tubes are simply stretched over these flanges, they are in sufficiently close contact to form air-tight joints.

To restore to the air its needed quantity of oxygen, a small pipe F, Figs. 638, 639, and 640, leads the oxygen from the metallic case E in which the store of oxygen, under pressure, is placed, and after passing through a flexible tube connected to it at F′, Fig. 639, is led into the flexible tube before alluded to, which is in connection with the air-bag. At the attachment of this smaller pipe with the larger, a short length of metal tube is used, as shown in Fig. 647.

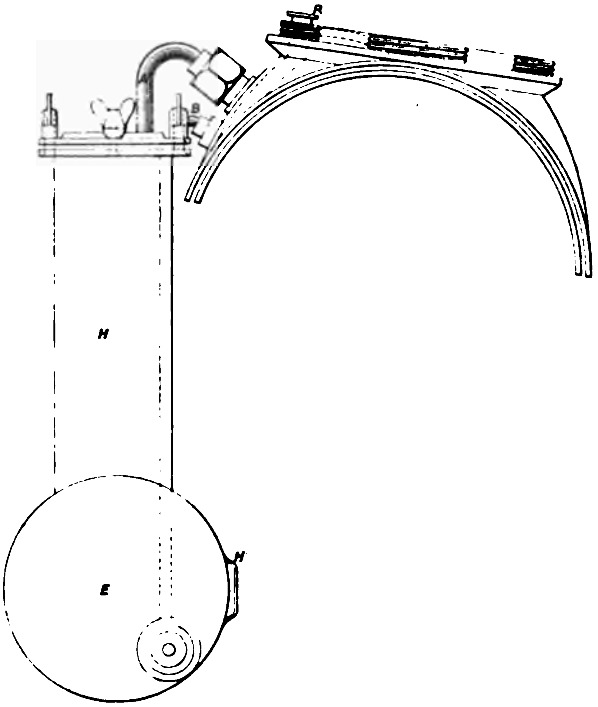

Fig. 649.—FLEUSS APPARATUS.

The passage of oxygen from the vessel E to the air-bag is regulated by a valve K, shown more particularly in detail in Fig. 648. If the screw K be turned. the valve can be lifted, more or less, away from its seat, consequently leaving a larger, or smaller, passage through which the oxygen passes out from the vessel E to the small pipe F. The loops Q, Fig. 645, at the lower part of the air-bag, are passed over the studs G at the ends of the vessel E, Figs. 638, 639 and 640, when the apparatus is being adjusted to the wearer. The straps at the upper part of the air-bag, Fig. 645, form a loop when buckled together, which hangs over the shoulder of the wearer and is connected to the apparatus at L L, Fig. 639. The lid of the vessel M is held down, and the vessel kept securely in the metallic case H, by bars J, passed across the top of the case through eyes which project up from it as shown in Figs. 638, 639 and 640.

The apparatus, as just described, is a suitable arrangement for wearing in foul air, but when used for enabling persons in an ordinary diver's dress to work under water, without fresh air, a modification of the apparatus, as shown in Figs. 649 to 652, is used.

Fig. 649 is a side elevation of the ordinary metallic shoulder-piece of a diver's dress, with the apparatus connected to it. There is a clip-plate all round the edge of the shoulder-piece, by which the opening at the top of the diver's dress is clamped in the usual manner, and a tight joint made between them. Fig. 650 is a front elevation of the helmet, which is secured to the shoulder-piece and locked thereto by giving it a partial turn. A mask is worn over the nose and mouth of the wearer inside the helmet, a front and side view of which are shown in Figs. 651 and 652 respectively.

When the apparatus is used for deep-water diving, the case H, containing the caustic stick soda, must be closed by a strong metallic cover, with metallic pipes leading to and from it, and the vessel M, which holds the caustic soda, is formed with a flange round the top. Above the top of the vessel M is placed a strip of sheet india-rubber, and above this the metallic cover. The cover is kept securely in place by screws, as shown in Fig. 649. The pipes which take the foul air to, and conduct the purified air from, this vessel, are secured to the head piece, as shown at A, Fig. 649.

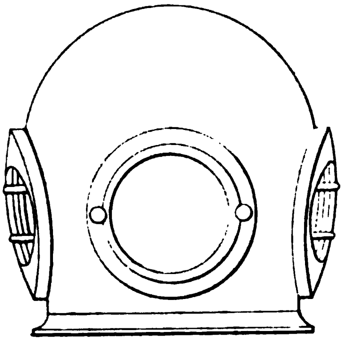

Fig. 650.

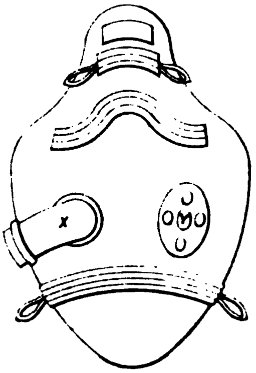

Fig. 651.

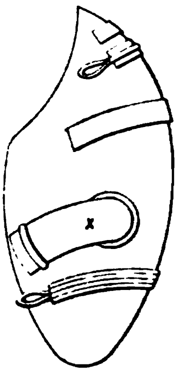

Fig. 652.

FLEUSS APPARATUS.

The purified air passes straight into the dress, which serves the purpose of the air-bag in Fig. 645, and the exhaled air is forced by the lungs through the pipe X, Fig. 651, and through a flexible tube to R, Fig. 649, from thence passing through the purifying vessel. A small valve is fixed on the mask at Y to prevent the exhaled air mixing with the purified air in the dress; and another valve is put on the outlet pipe, so as to prevent the exhaled air returning before being purified. A small pipe from the oxygen reservoir is also connected by a union to the shoulder-piece B, Fig. 649, and a continuation of the pipe leads into the interior of the helmet. The mask, or mouth-piece, Figs. 651 and 652, is not wanted to protect the eyes, and is consequently made smaller than the one shown in Figs. 635 and 636. H′, H′, Figs. 639 and 649, are metal loops formed on the exterior of the oxygen reservoir, through which a belt is passed and buckled round the waist of the person using the diving dress.

FLEUSS LAMP.

Accompanying the apparatus is a lamp which has been devised to burn when surrounded by gases which do not support combustion. The apparatus would be of no use in exploring the dark recesses of the mine without light, and this is obtained by means of the Fleuss lamp, shown in Fig. 653. In it, a metallic sphere N serves as a reservoir for compressed oxygen, and upon the sphere is fixed a spirit-lamp M, with a moveable pin D, for carrying a lime, and a moveable platinum tube G, capable of adjustment, for directing the stream of oxygen through the flame of the spirit-lamp on to the lime.

A tube O traverses the spirit reservoir of the lamp M, and the platinum tube G forms its upper extremity, while below it passes into the sphere N, which holds the oxygen and communicates with a jamb-cock P, capable of adjustment from without the sphere N, which enables the user to control and regulate the stream of oxygen for producing more or less light at pleasure. The spirit-lamp reservoir M forms part of the spherical oxygen reservoir, and is provided at its outer part with a screw thread and collar L, on which is screwed a double hood E, E′, the joint being made air-tight by means of a washer. The hood E consists of an elongated cylinder with domed top and flanged base and having within a screw-ring K for attaching to, or detaching from the screw thread and collar L, on the spirit-lamp M. To the flanged base is soldered or brazed a somewhat similar cylinder E′, of greater diameter, so as to form an annular space between the two cylinders, which is filled with water. The cylinder E is provided with a glass disc C facing the flame of the lamp, and the cylinder E′ similarly has a glass disc or bull's-eye B, by which means the rays of light from the interior of the lamp pass through the water contained by the two cylinders to the exterior. The products of combustion are carried into the space filled with water through a sensitive valve H, fitted into the cylinder E near the base, and pass outwards through a valve A in the centre of the dome above the water level to the exterior of the lamp. The outer cylinder E′ is provided with a suitable handle for carrying the lamp. A cap J is fitted to it, which, when removed, exposes the valve H in the inner cylinder and facilitates its renewal. The spherical oxygen reservoir N is fitted with a suitable connection R, through which is admitted a fresh charge of compressed oxygen when the reservoir has been exhausted. A metallic ring S, attached to the lower portion of the globe, forms a base on which the lamp will stand, and lamps used for submarine operations have a weight enclosed within this ring. The object of the weight is to counterbalance the buoyancy of the lamp. In recent forms of the lamp a worm and screw motion is added to the valve P, so as to regulate the admission of oxygen to the greatest nicety.

The oxygen is supplied by the patentees in wrought-iron bottles, 6½ inches outside diameter, 3 feet 3 inches long and ¼ inch thick, 16 hours supply tested up to 1,000 lbs. per inch, and filled with oxygen at about 600 lbs. to the inch for £5 5*s*. each, £4 10*s*. being for the vessel and 15*s*. for the oxygen. The contents of these bottles can be conveyed to the lamp and breathing reservoir in less than a minute.

The cost of the apparatus for breathing in foul air is about £20, and of the lamp about £14. For exploring purposes it is most useful, and in underground roadways of 3 feet 6 inches high or upwards, travelling with it is not difficult. It weighs about 28 lbs., however, and is too cumbersome to admit of the wearer ridding, timbering, or doing similar work. A man accustomed to the apparatus may remain 3 or 4 hours in it while surrounded by unbreathable gases with one charge of oxygen, but if he exerts himself violently or is nervous or excited the supply will be exhausted much sooner.

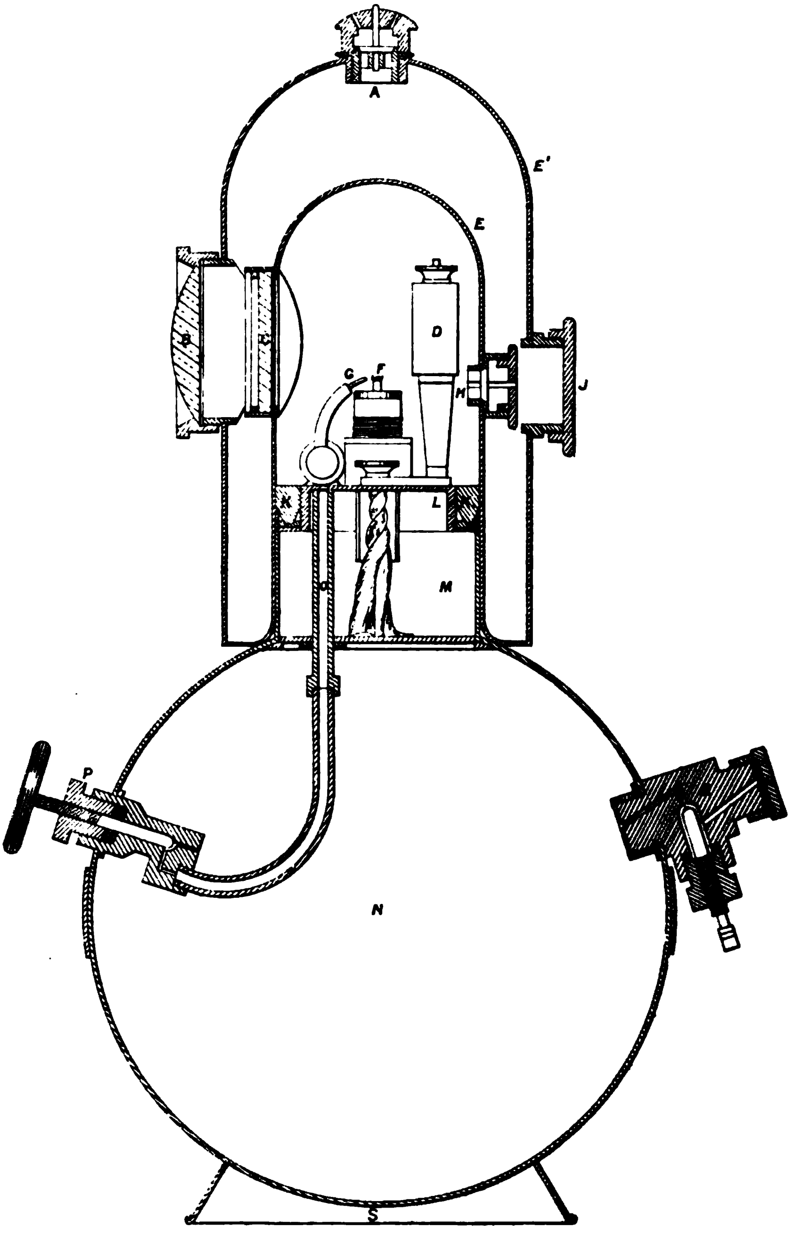

Fig. 653.—THE FLEUSS LAMP.

The supply of oxygen being under the control of the wearer may be urged as an objection to the apparatus. If he become nervous and supply himself too freely he will exhaust the store before it is safe to do so; no other inconvenience would arise, as an overdose of oxygen is not injurious. This objection would be removed if some regulator could be attached which would maintain a continuous and regular supply of oxygen.

The lamp will burn about 4 hours. When men enter the mine, with the apparatus fitted on, they proceed as far as possible with safety and comfort to themselves without putting the breathing appliance to their mouths. At the point where it becomes necessary to do so a saving of time may be effected by leaving there a store of bottles containing compressed oxygen, so that on returning to it the apparatus may be replenished.

There should always be 3 persons or more together when exploring in dangerous gases, and they should withdraw at once if any one of them get into difficulty.

At Killingworth the apparatus was used for the purpose of saving life, and there two men carried out a fainting man while the third carried the lamps. After the Seaham Colliery explosion on Sept. 8, 1880, resulting in 164 deaths, the Fleuss apparatus was used for exploring purposes.

As an instance of its use under water, a case at the Severn Tunnel may be mentioned. The making of this tunnel was beset with great difficulties. When the headings on the opposite sides of the river had approached to within 138 yards of joining each other, a large spring was tapped. In order to master it, new shafts were sunk on both sides of the spring, and by using powerful pumps in these the head of water was gradually lowered.

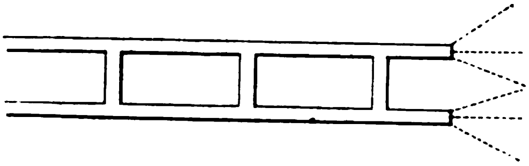

Fig. 654.—Boring Exploring Places

Beside the water from the large spring a considerable quantity came from the river heading through a flood door. To get rid of this for a time became of much importance, and a noted diver was engaged to make an attempt to close the flood door. The man made a most plucky attempt to get to it in his ordinary diving dress; but his air pipe floated so hard against the rough roof of the heading that although he got within 70 yards of the door, he was obliged to turn back and had extreme difficulty in retracing his steps.

Fleuss, the inventor of the apparatus described, was then invited down to close the door. He went down the shaft in company with Lambert, the skilled diver who had previously failed to shut the door, to the mouth of the heading, but not liking the appearances presented to his view there he came up again and declined to go further. Lambert then put on the Fleuss apparatus, and after one failure succeeded in closing the door.

EXPLORING FOR WATER.

In approaching, from the dip, old wastes containing water, it is necessary to drive a single place or a pair of places in the direction of the waste and to bore horizontally in the coal. The chisel used for this purpose is usually 1½ inches in diameter. In most cases one exploring drift is sufficient, but if two are required —one as an intake and the other the return—and both are within the statutory distance of 40 yards from the old workings, each drift should have three bore holes, one front hole and one flank hole on each side, the latter being at an angle of about 45° with the former (see Fig. 654), and bored at regular distances, say every 4 or 5 yards. The length of the front hole will be subject to circumstances, depending upon the nature of the coal and the head of water in the waste. The Mines Act says that the drift shall not exceed 8 feet in width at any point within 40 yards of the dangerous accumulation, and there shall be constantly kept at a sufficient distance, not being less than 5 yards in advance, at least one bore hole near the centre of the working and sufficient flank holes on each side. If 10 yards of barrier be desired, the front hole should be 11 or 12 yards, and then 1 or 2 yards of coal worked off, the hole being again bored the same distance after each such removal. A few fir plugs from 4 to 6 feet long (pointed, tapered, and hooped with iron at the head, and, if the pressure is likely to be very great, having cross-pieces attached so that the force of 2 or 3 men and a heavy hammer could be applied for entering and driving them) should always be ready at hand so as to stop the flow of water as soon as tapped; it can afterwards be allowed to run at convenient times. Suppose 5 yards to be the minimum length of hole, and 20 fathoms the minimum pressure, the length of hole should be increased 1 yard for every 10 fathoms. As the boreholes approach the waste the water will probably ooze through to the borehole and give some indication of the proximity to the waste, before actually holing into it. Mr. Greenwell says that the quantity of water which such boreholes will run per minute may be calculated by the rule given in answer 161, Chapter XVIII., where an example is worked out.

UNDERGROUND DAMS.

It is sometimes necessary to put in dams in the underground workings; for instance, exploring parties in narrow places may meet with large quantities of water which it is desirable to keep back, or the necessity may arise from other causes. In selecting the situation for a dam, care must be taken to choose one free from slips and faults of all descriptions, and it should be prepared, if this be practicable, with the pick, and no explosives used, because the shots shake the coal or stone. The sides, top, and bottom must be dressed perfectly smooth. Wood or brick may be used, the former being preferable, because the least yielding of the masonry abutments causes fracture and breakage. The requirements of a dam are that it shall be perfectly water-tight, and be able to resist the pressure brought to bear upon it. Masonry dams consist of two or more concentric arches placed several feet apart, the space between them being filled up with clay and well rammed. Sound, well-seasoned oak is the best material for wood dams. They are used in pieces from 3 to 8 feet in length (depending upon the pressure which the dam is required to resist), square, and tapered, and the radius of the inner circle should be from 18 to 30 feet. They are laid down in rows beside each other, and one upon another, with their larger ends towards the water. They are carefully prepared on the surface, and to ensure fitting accurately they are built together there, and each piece numbered before going into the mine. To allow for the pressure pushing the dam forward, it is advisable that its seat be continued of its tapering form for a few feet. Figs. 655 and 656 will assist in explaining the dam.

The sides, bottom and top having been accurately dressed, a layer of tarred flannel should be laid next to the coal or stone, and the pieces of wood built up. It is necessary to insert three metal pipes as the building proceeds; one, marked A on the sketch, about a foot from the bottom, and of a size to allow the water to run through, or if the quantity of water is very large two such pipes may be inserted. A second pipe B, about 2 feet from the bottom and 18 inches in diameter, is laid, to allow of the men passing through during the building and wedging of the dam. It must be remembered that it is necessary to have the wedging done on the side from which the pressure comes, viz., on the inside of the dam. Another pipe C, about an inch in diameter, is placed near the roof. The water is conveyed by boxes to the water pipe or pipes A in the dam, and the pieces having all been built up, the wedging is proceeded with from the inside. Fir wedges, 12 inches long by 3 inches broad and an inch thick at the head, are first driven in, and after these have been driven at all the joints and round the pipes, smaller wedges of oak may be driven, an iron chisel being used to prepare places for their insertion. When no more wooden wedges can be got in a few steel wedges may be made to enter, more especially between the wood and the stone or coal. After the wedging is completed, the workmen drive a plug of wood

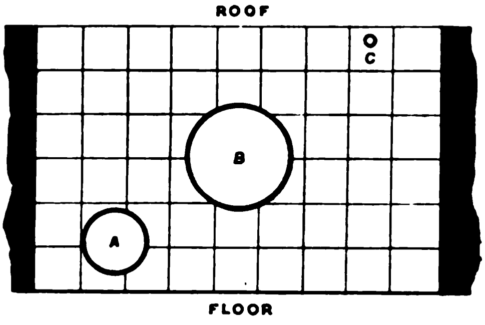

Fig. 655.

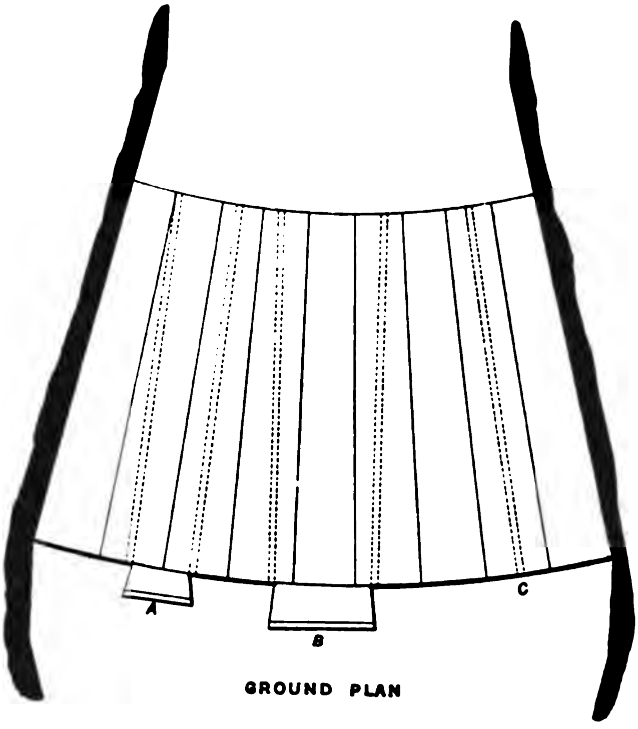

Fig. 656.

UNDERGROUND DAM.

which has been prepared for the purpose into the water pipe A, but if practicable the pipe C should be continued by means of iron pipes to such a level as will outset the water, otherwise it must be plugged. The workmen then retire through the pipe B, and a plug which has been prepared and carried in before the erection of the dam, and which is covered near its larger end with vulcanised india-rubber so as to ensure a water-tight joint with the pipe, is drawn in by means of a rope, and tightened by attaching the rope to a lever, or a windlass. The subsequent pressure of the water forces in the plug still more firmly. A dam of this description will resist a pressure of from 50 to 100 fathoms of water. The pressure the dam has to resist will be found by the following rule:—

H = head of water in feet.

P = pressure in lbs. per square foot = $62 \cdot 4\ H$.

p = pressure in lbs. per square inch = $\cdot 43\ H$.

WATER-BLASTS.

Sometimes water accumulates in workings temporarily abandoned, and reaches a considerable height in the shaft, and in such a case, after the opening into the seam has been "drowned," thereby excluding the passage of air, the air left in the rise workings will become compressed as the water rises in the shaft (see Fig. 657). If gas be given off from the coal, the pressure of the gas will also be increased, and both air and gas are thus pent up. In taking out the water on resuming work at the colliery, it is quite likely that a sudden rush of air and gas may take place, owing to the elasticity of these bodies, and when this happens, the air will force itself through the water with great violence and noise, and the level of the water in the shaft and workings will fall suddenly—it may be some fathoms—and produce what is called a "water-blast." Great care is therefore needed in unwatering a mine after inundation, for several instances are recorded where water-blasts on a large scale have occurred.

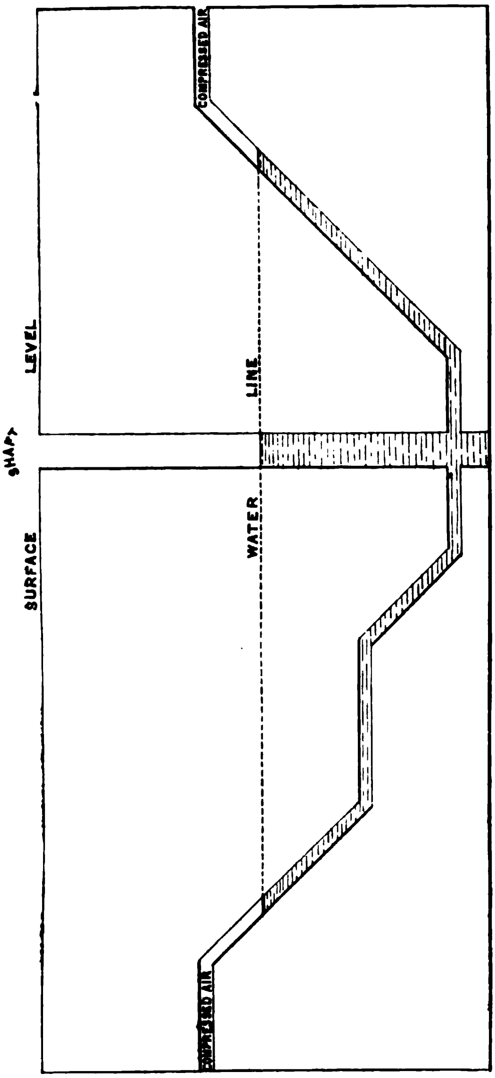

Fig. 657.—Sketch to Illustrate the Phenomenon of a Water-blast.

UNDERGROUND AND SURFACE FIRES.

Undergound fires are of frequent occurrence in mining operations, and when they occur are exceedingly difficult to deal with. The heat itself is intense, and the air near the fire is vitiated by the products of the combustion, and if firedamp exist, there is a constant danger of an explosion added to the already existing evil.

Fires are frequent in the workings and wastes of some mines, and where naked lights are used their occurrence is to be expected sooner or later, for there is generally plenty of inflammable material ready. They may arise from such causes as a light being set to a body of dry material, in contact with the coal, such as old props, canvas or deal bratticing, doors, &c., or sparks from an underground boiler or furnace, or an explosion of firedamp, or of a shot. A frequent cause is spontaneous combustion. The fire usually extends towards the fresh air current entering the mine.

If a seam of coal contain iron pyrites, and heaps of small pyritous coal lie about exposed to a moist atmosphere, the conditions were at one time supposed to be favourable to spontaneous combustion. It is very doubtful if pyrites ever leads to fire. It seems rather to arise from the heating of porous coal which absorbs oxygen from air and slowly undergoes combustion, for coal seams which have no iron pyrites are also subject to spontaneous combustion. When heaps of such porous coal are exposed to a moist atmosphere, oxidation takes place, heat is generated, and, under certain circumstances, combustion ensues. The nature of the roof affects the spontaneous combustion of coal. Beneath a sandstone roof spontaneous combustion is unlikely to occur, because the heat escapes as fast as it is generated. With a soft shale roof, which is impervious to the gases formed, the heat is confined, unless there are fissures which allow the gases to escape.

As the presence underground of heaps of small coal is necessary to spontaneous combustion, in order to avoid fires from this cause, all small coal should be sent out to the surface if this be practicable. Another plan is to prevent the air from coming in contact with the small coal in the wastes by erecting air-tight barriers. Another method, and probably the most effective, is to send such a volume of air to the dangerous masses of coal as to keep their temperature down.

To extinguish an underground fire, the first procedure is to isolate that portion of workings where the fire exists. By a reference to the colliery plan, a line is decided upon which has the fewest number of openings crossing it, and air-tight stoppings or barriers must be erected in each of these places, and as time is of great consequence, these must be proceeded with without delay. If the air could be effectually kept from the fire in this way, it would soon cease, but it is almost impossible to close all crevices, and it has been found where nothing more was done that, although the fierceness of the fire ceased at the time, directly operations were begun again and the air admitted, the smouldering fire burst out afresh. This being so, after the barriers are erected and the intensity of the fire stopped, recourse is had to other means to finally extinguish the fire, if such means are available.

The most effectual plan is by flooding the mine, which may be done by stopping the pumps and allowing the water to rise, or by sending down water from the surface. This is, however, a costly method of putting out a fire, for the water has to be pumped out again after the fire has ceased, and generally much damage is done to the roads and workings by flooding. This is especially the case where the floor or roof is softish or of a clayey nature.

Another plan is to produce carbonic acid gas in large quantities and direct it by pipes on to the fire. Carbonic anhydride may be sent to any portion of the mine—to the extreme rise for instance—which could not be reached by water except by flooding the whole mine.

It is sometimes impossible even to enter the mine after the coal has been set on fire following an explosion of firedamp. In this case the shafts themselves must be sealed. For this purpose a scaffold is fixed a few feet down the shaft. Or it may be suspended from balks of timber laid across the top by means of stout chains, and it should fit truly round the shaft; a quantity of plastic clay is then thrown on the scaffold, and a stratum of water may be placed on the top of this to ensure all being air-tight. Cast-iron pipes of 6 inches diameter may be placed through the scaffold before the clay is thrown down, reaching to the surface, and furnished there with valves opening outwards only, so as to admit of after observations, such as the analysing of escaping gases, or testing them with the safety lamp. An inch gas-pipe may also be placed through the scaffold, reaching upwards to the surface, and furnished with stop-cocks, so that a water gauge or pressure gauge may be applied and the pressure inwards or outwards ascertained.

On Dec. 14, 1892, twelve persons were suffocated by smoke from a fire occurring at an underground haulage engine in Bamburlong colliery, caused by the attendant, a boy, attempting to thaw the cylinder by pouring some paraffin oil upon it and then setting fire to the oil.

On 11th April, 1893, at the Great Western Colliery, a fire occurred at an underground haulage engine worked by compressed air and lighted from an electric source, by which 63 lives were lost through suffocation, and some hundreds of other workmen placed in jeopardy. It was supposed to have originated by sparks from the brake blocks setting fire to brattice cloth fixed under the engine to shelter the engine man and to keep dust from the engine which was fixed immediately over a main intake and haulage road.

These disasters emphasize the danger to be apprehended from fire breaking out at underground engines, and show the necessity for greater precautionary measures than have hitherto been generally adopted to guard against the risk of loss of life and property.

Increased safety will result by excluding all woodwork in the construction of engine seats, flooring and ladders; reducing the quantity of combustible material as much as possible, and exercising the greatest care in the use of oils and in the disposal of dirty cotton waste. Petroleum and other mineral oils of a low flash point should be rigidly kept away from such situations. As far as possible engines and their surroundings should be kept clean and free from deposits of rubbish and dust, as well as from combustible or inflammable material.

In situations where the roof or sides require support, the gallery in which the engine is situated, and for ten or twenty yards on each side of it, should be arched or walled.

In the case of a serious fire occurring in the intake airway, this road at once ceases to be available for the egress of those persons who may happen to be on the in-bye side.

It has been suggested that if the various districts of a colliery could be isolated by means of close-fitting doors, on finding anything seriously wrong with the ventilation, such as a violent concussion or a cessation of the air-current, or a smoky atmosphere, the workmen in any particular portion could close these safety-doors and prevent after-damp or smoke from reaching them. They would then be able to exist for some time on the air enclosed within their own section of the mine. The force of an explosion, however, causes large falls of the roof which may prevent workmen from reaching such doors; or, in attempting to get to the doors the workmen may fall victims to after-damp; or if the workmen were to succeed in reaching the doors it would probably be only to find them immovable or blown away. If the explosion were not sufficient to cause these ill effects, it would possibly leave the distant parts so undisturbed that work would proceed in them without interruption or alarm, and therefore without any attempt on the part of the workmen to close the doors. They might in that case remain in ignorance of the explosion and be overtaken by the after-damp. Even when a district is shut off from air or after-damp reaching it, it cannot long remain habitable, for the carbonic acid gas given off by the men will vitiate the air, and probably the seam will give off large quantities of gas to add to the impurity of the atmosphere. The length of time which must elapse before the stagnant air becomes irrespirable will depend upon the volume of gases emitted by the seam, by the number of the workmen and horses, and the cubical contents of the space in which they are confined. An objection to the use of such doors is, that in case of false alarm the wrong doors may be closed, and the self-imprisoned miners thus create for themselves a danger when none menaces them. Telephonic communication from the surface to the different parts of the workings may be the means of avoiding such self-imprisonment, but it cannot be relied upon as a means to warn miners of approaching danger, because the falls of roof would probably damage the wires, and might make the speaking stations inaccessible.

It is not sufficient that engine-houses and their approaches should be constructed of fireproof materials; a few hand grenades and portable chemical fire-engines should always be kept within easy reach of the engine-man, even if an adequate water supply is available.

Engine-houses that are dry and warm are in special danger of fire if anything of an inflammable nature is allowed to remain in them. In these the use of petroleum-engines is not advisable. Underground boiler-fires are a source of danger, and should, as far as possible, be avoided.

In view of the possibility of its being the only escape-road, it is highly desirable that every workman should be acquainted with the way out from his own working place along the return airway; that all return airways should be of ample sectional area, and be kept free from obstructions and water lodgments; that where connected by air-staples or blind-pits, these should be properly fenced around and provided with strong ladders subject to regular examination; that side openings off travelling roads should be kept fenced, or large and legibly-printed direction-boards be put up, and constantly maintained in good order, at every change in the direction to show the way out; that at the escape-shaft apparatus should always be at hand for immediate use, and also proper landing and signalling appliances; that where the raising apparatus is not in daily use, it be tested at regular intervals to ensure its being in working order in cases of emergency; and also that at regular intervals two or three of the workmen in each district should accompany the fireman or wasteman through the return, so that the whole of the men, by going a few at a time, may be kept well acquainted with the route and the alterations frequently taking place in it.

Fires on the surface, although not usually attended with loss of life, frequently place the shafts, and consequently all persons in the underground workings, in great danger. For this reason, not less than for the safety of valuable property, the greatest care and vigilance are required to prevent their occurrence.

A very fair idea of the terrible effects of underground fires is obtained from a perusal of the following examples:—

During the night of May 1st, 1882, a fire broke out in the underground workings, probably at an engine and boiler, of the Baddesley Colliery, near Atherstone in Warwickshire. Nine men had previously gone down to work in the night shift, and on the fire being discovered about 10 o'clock p.m., a search party descended the shaft to help the imprisoned miners. At 8.30 a.m. the next morning an explosion occurred in the mine, with the result that many of the rescue party perished at the time, and others after surviving in a critical condition for a few days, so that the total number of deaths reached 23.

In spite of all efforts the fire raged for a considerable time, so that the pits had to be sealed before it was extinguished and all the bodies recovered. The pit covers were not removed until Nov. 23rd, 1882, when much carbonic acid gas was found in the workings. The fan was started to remove this and to restore ventilation, in order to allow of explorations for the recovery of the bodies, but owing to the continuance of the fire some of the bodies were not recovered until some years later.

Some members of the exploring party on this occasion distinguished themselves in their brave endeavours to extinguish the fire and to reach Mr. Dugdale, the owner, and two others (all three being themselves explorers), who had ventured into the mine and were in imminent danger. The heroic conduct—a sample, happily, of what is often witnessed on such occasions—met with unusual recognition at the hands of Her Majesty, who conferred on the survivors the Albert medal—to some the first class and to others the second.

On May 25, 1888, it was discovered that the pit-framing and surface sheds at the Maritime Colliery, Pontypridd, were on fire. The fire occurred at night when there were some 30 men in the pit, for whose safety much concern was felt. The working of the ventilating fan was stopped in order to prevent the flame from being sucked into the downcast shaft, and so spreading into the workings. The fire was restricted to the surface, and was eventually extinguished, all the men having been brought up in safety by the upcast shaft. The danger was lest some of the burning mass should fall into the shaft. The men imprisoned in the mine kept clear of the descending smoke by opening the separation doors between the shafts. The fire was caused by a comet paraffin lamp used by the banksmen, which ignited the tram oil with which the platform was saturated, and spread to the pit-frame, which was covered with pitch.

On July 4, 1893, a slight explosion of fire-damp at Combs' Pit, Thornhill Colliery, near Dewsbury, Yorkshire, fired some fittings in the shaft, the burning of which was infinitely more disastrous than the explosion itself.

The Combs pit, 11 feet in diameter, is the downcast and drawing shaft, and is sunk to a depth of 165 yards from the surface to the Blocking-bed, passing through the New Hards seam at a depth of 115 yards, and the Wheatley seam at a depth of 140 yards from the surface. The Wheatley seam is thus 25 yards below the New Hards, and an equal distance above the Blocking coal.

A wooden staging or scaffold was fixed in the shaft at the 140-yard level, in such a way as nearly to cover over the shaft below. An opening of about 4 square feet was left on either side of this staging. By this means the air for the most part went into the Wheatley workings, none other being carried on at the time of the explosion, while a scale passed through the sides of the staging into the Blocking bed. The latest recorded measurement of the air previous to the explosion showed there were 19,600 cubic feet of air per minute going through the Wheatley workings, and it was estimated that about 2,000 cubic feet descended to the Blocking bed.

The upcast or Ingham pit is 13 feet in diameter. At the top of this shaft a Guibal fan, 30 feet in diameter, running at a speed of 48 revolutions per minute, produced the ventilation.

Both shafts were originally sunk to the New Hards; at a later date the downcast was continued to the Blocking bed, and a staple pit sunk near the upcast from the New Hards to the Blocking bed.

The air passing at the sides of the staging was coursed through the Blocking bed, ascending the staple pit, after which it joined the Wheatley seam air, the re-united volumes of air then flowing to the upcast pit.

The downcast shaft was sunk through a fault which was known to give off a small quantity of fire-damp under the staging, and some attempt had previously been made to use it for lighting the pit bottom. With this object in view a pipe had been laid having one end let into the brickwork of the shaft and reaching up the shaft to the Wheatley seam porch, but there was not sufficient gas thus collected to be serviceable.

Safety-lamps were used in the workings, but open lights were allowed on the main roads for a considerable distance from the shaft, and oil-lamps were used in the porches of the downcast shaft at the Wheatley seam.

It is believed that the gas lighted at one of these oil-lamps, and extended along the pipe alluded to, or along the troughs or boxes used to convey the haulage rope under the iron plates to the feeder of gas under the staging. The feeder was a small one, the pit wet and coal-dust absent, and the effects of the explosion would have been slight, but that the flame ignited and almost entirely consumed the wooden conductors between the Wheatley and Blocking seams and the wooden staging. The dense smoke resulting from the fire together with the after-damp was carried by the ventilating current into the Wheatley seam workings, causing the death of 139 out of 146 persons then at work, and the seven survivors were more or less injured by the smoke.

The Maritime and Thornhill Colliery accidents point to the desirability of using iron pit-frames, and of avoiding the use of wooden fittings in shafts.

On August 31, 1893, a fire broke out in the winding engine-house at the Sandwell Park Colliery. The pit rope was burned through, and on becoming severed, the cage dropped down the shaft, a distance of 450 yards, and was broken to pieces. It is supposed that some men repairing the machinery carelessly left about the engine-house a quantity of greasy material which became accidentally ignited.

In addition to all possible precautions against fires, there should be proper provision at the colliery for promptly dealing with them on their outbreak, for no precautions, it will be remembered, can guard against incendiary fires.

Some seams of coal, such as the Cockshead and the Bullhurst seams in North Staffordshire, are peculiarly liable to gob-fires arising from spontaneous combustion. If the preliminary heatings are promptly discovered they may be successfully dealt with by isolating them and excluding the air, but a fully-developed gob-fire is extremely difficult to allay. Whatever precautionary measures are adopted to guard against gob-fires a constant watchfulness must be maintained in seams subject to them, in order to discover and deal with their first intimations rather than their far more difficult developments.

Messrs. Sinclair & Co., 19, Eldon-street, Finsbury, London, E.C., manufacture trustworthy appliances for rapidly dealing with fire, which should be at hand for any emergency.

The most important of these is Sinclair's *chemical extincteur*, shown in Fig. 658. It is made in different sizes, the No. 5 holding 7 gallons, and the No. 6 10 gallons of chemically treated water, each gallon of which has more fire-extinguishing power than 20 gallons of ordinary water. The gases generated from the preparation contained in the iron cylindrical vessel are inimical to combustion, and, being of greater density than the air, smother the flame into which they are discharged. The engines are made very handy for use, having an arrangement of straps for rapid attachment to the back of the operator, whereby they can be lifted and carried to the seat of the fire without delay. They are self-acting and always ready for immediate use. By means of the short hose-pipe connected at one end to the bottom of the engine and terminating at the other in a jet, the operator can turn the cock and direct the issuing stream into the fiercest part of the fire and play upon it as he thinks best. In some positions the bucket extincteur may be more convenient. It contains the same chemically treated water as the other, but has a handle on the top by which it may be carried from place to place, and it may be rested on the ground while the jet at the end of the hose is directed to the fire. The bucket form of the extincteur contains 3 gallons of the prepared liquid. Smaller quantities are sold in glass jars to form *hand grenades*.

A number of each kind of extincteurs and hand grenades should be placed in the surface and underground engine-houses and in other conspicuous positions above and below ground so as to be handy for immediate use.

These portable extincteurs have been in use for many years, and have been successful in putting out many fires which otherwise would have had most disastrous results. They are of great value in exploring a mine after an explosion, as smouldering fires may quickly be rendered harmless, and after-explorations proceed in safety.

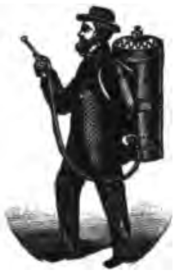
Fig. 658.—Fire Extincteur.

Large and serious fires which cannot be extinguished by simple and portable apparatus require a large stream of plain or chemically treated water thrown upon them continuously and with force. Sinclair's combined *chemical and manual fire engine*, shown in Fig. 659, has been designed for this purpose. It is mounted differently for surface use, having 3 plain iron wheels and a drag-handle, but, as shown in the drawing, it is specially adapted for use in any underground roadway, which is at least 5 feet high and laid with rails for trams. It is carried on four flanged wheels, with collar and set screw, so that the axles are adjustable to whatever gauge of rails may be used at a given colliery. The height of the lever bars in full play does not exceed 4½ feet. Ball valves are used in the pumps to prevent choking from sand or other substances. The lever bars can be disconnected to allow the engine to be lowered in the cage. Its compact form allows of its being easily moved about the workings underground, and also about the surface tramways. Each of the two compartments in the tank arranged above the wheels contains 42 gallons of water. The engine can be worked by four men, two at each lever bar which sets the pumps in motion, but better still by six.

A charge consists of 8 lbs. of prepared acid and 4 lbs. of prepared alkali. The two chemicals are put separately into the two compartments, one in each, and are pumped together into the air chamber where they become mixed; from thence they pass along the delivery hose, which terminates in a suitable jet which is directed by the operator towards the fire. Carbonic acid gas is then liberated with immediate effect. With the engine there should be supplied ample lengths of 1½-inch best hose, which are required both for suction and delivery. One end of the suction hose is fitted with a brass rose for preventing the entry of chips, &c., with the water. There are four pumps, about 4½ inches in diameter, two suction and two delivery, so that a continuous stream is drawn into the tank while another is being discharged from it. By this means about 30 gallons of water containing about 250 gallons of carbonic acid gas may be thrown a distance of about 75 feet. A good water supply is, of course, necessary.

A serious fire at the Whitwood Collieries, Normanton, was subdued in a few hours by means of one of these fire-engines, on February 25, 1876. When the fire was first discovered about 40 yards of the seam of coal were on fire and dense masses of smoke were being discharged from the top of the upcast shaft.

An appliance which enables the wearer to breathe in a smoky atmosphere may be of use at a time of fire. A heavy apparatus like the Fleuss would be unbearably hot, but Messrs. Sinclair & Co. supply a very convenient *fireman's respirator* which is capable of rendering valuable aid in rescue work. It is shown in Fig. 660, and is the invention of Dr. Tyndall, being a combination of his respirator of cotton-wool moistened with glycerine and Dr. Stenhouse's charcoal respirator. An improvement in its design is due to Mr. Sinclair. Charcoal has the property of absorbing offensive gases. Provided with the fireman's respirator, which may be worn without inconvenience, a man may remain a long time in a smoky atmosphere, the eyes being protected by elastic goggles.

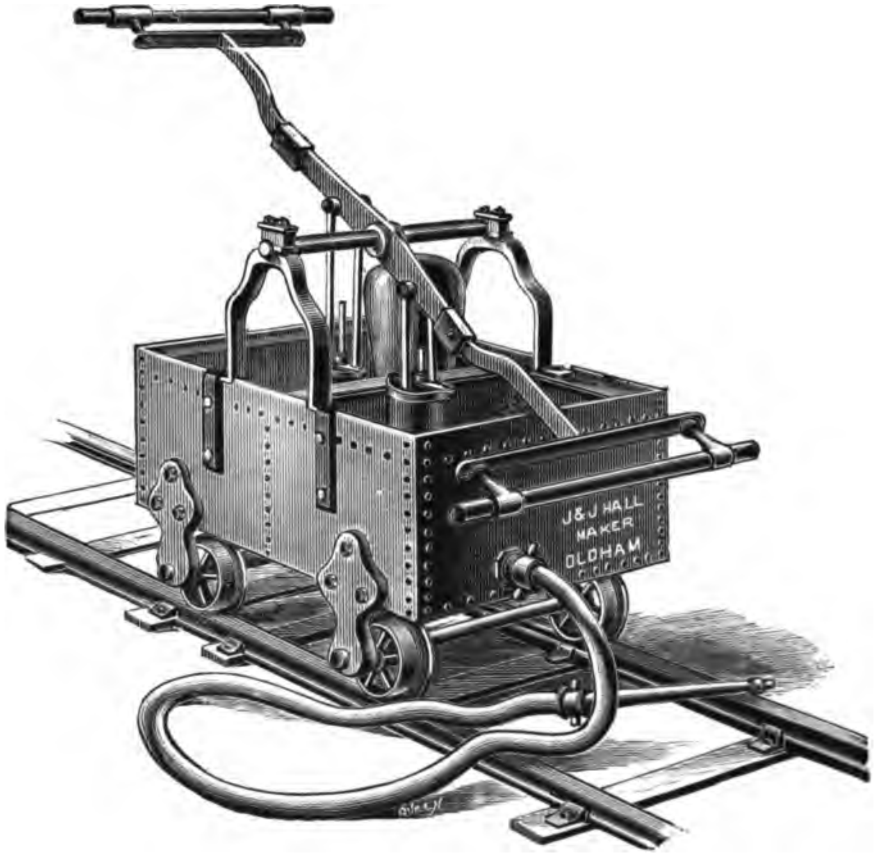

Fig. 659.—Combined Chemical and Manual Fire Engine, Mounted on Tram Wheels.

TESTING THE ROOF.

It has been customary to judge of the safety of the roof by general appearances and by the sound produced on tapping it with a small hammer or other suitable article.

No one who carefully studies the reports of H.M. Inspectors of Mines can fail to be struck with the abundant proof that to place reliance merely on sounding the roof and looking there for weaknesses is entirely insufficient to prevent accidents and fatalities by falls of the roof. It seems clear that where experienced miners and subordinate officials, in spite of all their efforts, fail to detect dangerous insecurities in the roof, it behoves owners and managers to adopt other measures. In some districts this has been attempted by fixing a maximum distance for props, whatever may be the appearance of the roof, and supplying additional props where defects or suspicious changes are detected.

FAULTS, AND METHODS OF DEALING WITH THEM.

Faults are frequently met with in mining operations. When the workings meet with interruptions which have not been proved elsewhere, to ascertain if the fault is a rise or dip one, observe the angle which the leader of the slip makes with the floor. If the angle be obtuse, as shown in Fig. 661, the leader will have appeared in the roof first, and the probability is that the coal is thrown down on the other side of the fault. If, on the other hand, the angle be acute, as shown in Fig. 662, the leader will have appeared in the floor first, and the probability is that the coal is thrown up on the other side of the fault. These remarks hold good generally, but in the case of overlap faults, which are prevalent in some districts, the reverse obtains, as seen in Fig. 663. Frequently the rate of dip is considerably altered as the workings on a seam of coal approach a fault, and this irregularity indicates the coming disturbance.

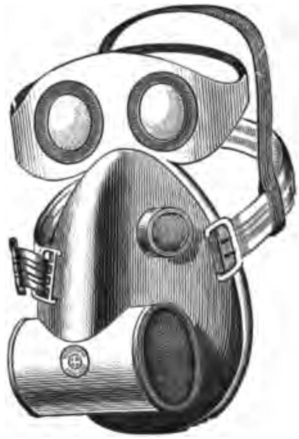

Fig. 660.—FIREMAN'S RESPIRATOR.

Rolls in the coal seams are not, strictly speaking, faults, as there is no interruption in the continuity of the seam, but the coal describes an abrupt curve rising rapidly from the beginning of the roll in a curved line to its summit and as rapidly dipping back to the end of the roll, from which the seam continues in its normal condition, or the roll may be curved in the opposite direction, and its lowest point be considerably below the level of the seam where it resumes its normal condition.

Sometimes the continuity of a coal seam is broken without any disturbance in the level of the two ends of the seam, which are separated by a mass of unproductive material, in Somersetshire called "dead ground." At others the two ends of the seam, instead of terminating abruptly against the barren ground, will taper away in wedge-shaped pieces of coal against it, and this kind of fault is called a "nip out" or "want."

After proving a fault and ascertaining the amount of dislocation, unless large, a permanent road should be made through it. In a perfectly level seam, and the fault a downthrow or "dipper," the bottom should be taken up from a point a few yards on the out-bye side of the fault, and the road continued through it at a regular gradient suitable for haulage, say, of 2 or 3 inches to the yard, till the seam is cut on the other side of the fault. If the fault is an upthrow or "riser," the top should be taken down and the coal reached by an easy rising road. If a dip fault cut the coal off in the water-levels, where the seam of coal has a certain rise, and it is desired to keep strictly to the water-level course, the water-levels must be turned in the direction of the rise, and by keeping truly water-level, but with the drift turned at a suitable angle, the coal will be cut on the other side of the fault. If, however, the fault be an upthrow, and it is important not to "lose level," the water-levels should be given a "down-hill" turn, so as to cut the coal on the other side of the fault, and yet keep the water-level course. Examples have been worked out at the end of Chapter VIII. in explanation of this, see Answers 46, 47 and 49.

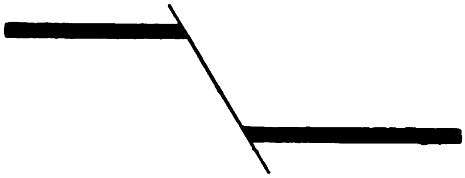

Fig. 661.—Downthrow Fault.

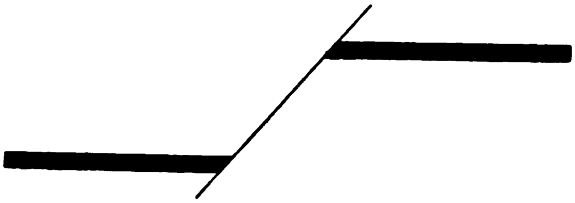

Fig. 662.—Upthrow Fault.

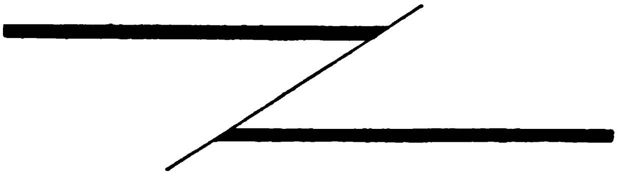

Fig. 663.—Overlap Fault.

STEAM INDICATORS.

The original indicator was invented by Watt, for the purpose of accurately ascertaining the internal condition of the steam engine, the state of the vacuum, the amount and variations in the pressure of steam at every point of the stroke, the cushioning, the condition of the slides, whether the ports are opened and closed at the proper time, &c., in his engines.

Watt's indicator, in a form for tracing out a complete diagram, is shown in Fig. 664. It consists of a cylinder L, about 1 inch in diameter and 6 inches long, fitted accurately with a solid piston P. At the bottom, the cylinder is fitted with a steam-cock K, which is screwed into an opening made in the cylinder-cover, or into an opening in the cylinder of the engine itself, close to the end. At the top, the cylinder is open, so that the upper side of the piston P is subject to atmospheric pressure. The piston is fitted with a small rod on its upper side, which carries at one end a pencil R, so as to mark on a board moving to and fro horizontally in the frame A B C D. The board receives its motion by means of a cord, shown in the Fig., which is fastened to some reciprocating part of the engine, the period of whose motion is identical with that of the engine piston. The weight M at the other end of the cord causes the return motion of the board. The spiral spring N controls the vertical motion of the piston.

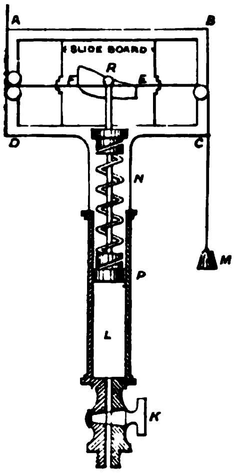

Fig. 664.—Watt's Indicator.

As first introduced by Watt, there was no sliding board to the instrument, but the pencil moved in vertical direction in front of a graduated scale. By this means the pressure of steam in the cylinder, or the amount of vacuum, could be ascertained. The addition of the sliding board, however, enables a complete diagram to be traced on it, by means of which the steam-pressure and vacuum at any point of the stroke is obtained.

When the instrument is screwed into the steam-engine cylinder-cover, the string passing upwards at A is attached to some moving part of the mechanism so that the board receives a motion corresponding to the engine piston, but with a reduced travel. If the board be now moved to and fro, the pencil traces the horizontal line E F, technically called the *atmospheric line*. On turning the cock, K, steam is admitted from the engine-cylinder into the lower end of the indicator-cylinder, and presses upwards against the piston P. When the pressure of the steam balances that of the external atmosphere the piston remains at rest. If the steam-pressure (or uncondensed vapour below the piston) be either greater or less than that of the atmosphere, the pencil will rise or fall, and the curve drawn will be the result of combining the motion of the pencil with that of the board.

The steam indicator invented by Watt has been much improved, but the general principle remains the same. Perhaps the best known and most frequently used at collieries is *Richards's*, which will be understood on reference to Fig. 665. *c* is a screw for fastening the indicator to the cylinder. The handle on sketch is to open the connection between the cylinder of the engine and the indicator, and by means of which the steam is admitted under the piston *d*, working in the indicator cylinder *fm*, which has the end *m* always open to the atmosphere. The piston *d*, with its rod *e* of the indicator, are shown in dotted lines, the former being half a square inch in sectional area. The dotted slanting lines show the spring which keeps the piston down, and against which the steam has to act in forcing up the piston *d*. A complete set of these springs and scales corresponding thereto, suitable for working at different pressures, are usually supplied with the instrument. *fr* is the barrel round which the paper is wrapped, the paper being made fast by two clips. Inside this barrel is a spring, the object of which is to effect the return of the drum, after it has been rotated by the string.

Although the movement of the drum has no connection with that of the indicator piston, when attached to a steam cylinder for use, both are made to move simultaneously, so that when the barrel has moved nearly round once, during which the piston goes up, the force of the spring causes it to return, as the indicator piston goes down. The graduated scale is to measure the pressure of steam and the vacuum, but it is not marked on modern indicators. A piece of whipcord passes round the pulley *gs*, which gives motion to the barrel. While the piston of the engine moves several feet, that of the indicator moves up less than an inch, and the barrel has to move round four or five inches in the same time. Motion is given to the barrel from the piston crosshead, and levers are used to reduce this motion. If the length of the diagram be three inches, and the length of stroke three feet, we have to proportion the levers as 3 : 36, or as 1 : 12, and the required motion is obtained. The indicator barrel is moved round by the string which is attached to its proper relative position of the lever, actuating the pulley *gs*, and with the pulley, the barrel. The arm *hh* carries the parallel motion *kjpn*, the pencil being at *l*. By means of this parallel motion the stroke of the indicator piston is, say, from 1 to 2 on the sketch, but the pencil is required to have more motion, say from the lower 15 to 25 on the scale, and this it gets through the application of the parallel motion. This is readily seen, for the end of the indicator piston rod is attached to the lever *pn* at *o*, and the end *n* being fixed, the motion of the indicator is multiplied in the proportion of *no* to *op*. In Richards's indicator this multiplier is about 3·5, and the motion being thus increased, the pencil indicates on an enlarged scale the least variation of pressure or action.

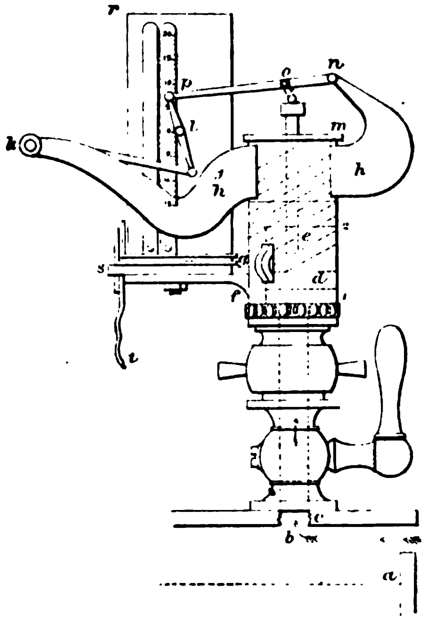

Fig. 665.—Richards's Indicator.

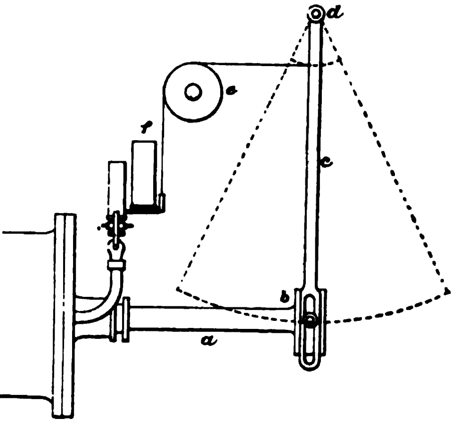

Fig. 666.—Pulley and Lever Indicator Gear—Over Motion.

There are different methods of arranging the lever or levers to the crosshead or piston-rod, sometimes two levers being used, sometimes one, and these may be applied either with under motion or over motion. Fig. 666 shows the gear with over motion.

The indicator is shown at *f*. The lever *c* is hung on a pin, *d*, supported by a bracket, or anything equally applicable. It is looped at its other extremity, so as to allow for the travel of the pin on the crosshead, *b*. The barrel of the indicator, *f*, derives its motion from the correct point on the lever, *c*, being connected by a cord passing quarter round the pulley *e*, and then continued to the pulley on the barrel.

Supposing now the indicator to be attached in this way to a vertical condensing engine, its action is as follows. As soon as the attachment is completed, the lever at the crosshead causes the barrel of the indicator to move from right to left, and a straight horizontal line will be drawn by the pencil. Generally, the pencil is allowed to mark this line several times, and it is called the atmospheric line, because it coincides with the atmospheric pressure: all parts of the diagram above that line show the pressure above the atmosphere, and all parts below it show the vacuum, so that the top part of the diagram is called the "steam," and the bottom the "vacuum." It has been stated that if only the barrel be made to move, a horizontal line is traced, but if now the motion of the barrel be stayed, and the steam admitted to the indicator by turning the handle for the purpose, the pencil would be driven straight up, or a vertical line would be traced. When both the barrel and the indicator piston move together, the result is a line compounded of the two motions.

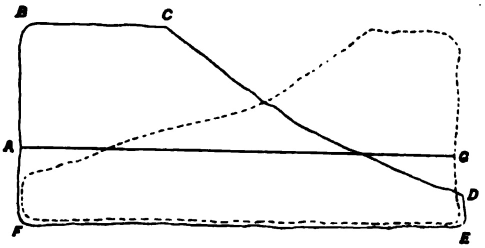

Fig. 667.—Indicator Diagram.

After the indicator has been attached to the crosshead at one end of the cylinder, say the top, having marked the atmospheric line a few times, the handle of the indicator, which admits the steam to it, is turned at the instant the top port opens, the piston of the engine being at the top of its stroke. The moment the steam enters the cylinder it drives the piston down in the direction of *a* on the indicator sketch, Fig. 665, and also enters the indicator in the direction of *b*, forcing the piston of the indicator up. This causes the vertical line AB, Fig. 667, to be drawn on the paper. The steam continues entering at its normal pressure, the piston of the engine goes down, and, so long as the pressure remains the same, the pencil of the indicator remains at the same height, and the barrel moves round, causing the line BC, Fig. 667, to be drawn.

When the pencil gets to C, the valve has closed the port, and the steam is left to expand; as it expands the pressure decreases, becoming less and less, and the pencil falls gradually lower and lower to D. When it gets to D, the upper port is opened to the exhaust, the steam rushes out in a contrary direction to the arrows on Fig. 665, and the pencil falls to E, Fig. 667. There is now a vacuum above the piston of the engine and below that of the indicator, and the vacuum line E F is traced. When the pencil gets to F the piston has arrived at the end of its stroke, cushioning takes place and the pencil rises at once to A.

Similarly the dotted lines show a diagram from the other side of the cylinder, so that each figure represents the action of the indicator through an up-and-down stroke or a complete revolution of the crank.

As illustrating the use to be made of the indicator diagram, suppose Fig. 668 to have been taken. The first thing to do is to divide the length of the figure into 10 equal divisions and then to set off half of one space from each extremity and arrange the remainder equi-distant. Lines are drawn at right angles with the atmospheric line from each intersection or divisional point, then with the scale of the diagram each ordinate is measured and figured above and below the atmospheric line. The total sum of each set of ordinates is then obtained by simply adding the several lengths together, and the mean is got by dividing the total by the number of ordinates—in this case 10. In Fig. 668 the mean pressure of the steam is 15·575 lbs. on the square inch, and the vacuum obtained is equal to an atmospheric pressure of 9·69 lbs. on the square inch. The effective pressure on the piston then is 15·575 + 9·69 = 25·265 lbs. per square inch, and the horse-power of the engine may be found from this, knowing also the length of stroke and the number of strokes the engine is going per minute. It will be observed that the remarks on Fig. 668, R 110, S 27½, V 26, mean that the engine is going 110 revolutions per minute with a steam pressure in the boiler of 27½ lbs. and the vacuum gauge marks 26 inches or 13 lbs. pressure. It will be noticed that the highest record of full supply of steam in the cylinder is 24·2 lbs. showing a loss due to friction and radiation in passing from the boiler to the cylinder, also that the difference in the vacuum pressure is owing to the temperature in the cylinder being higher than in the exhaust steam pipe at the condenser end.

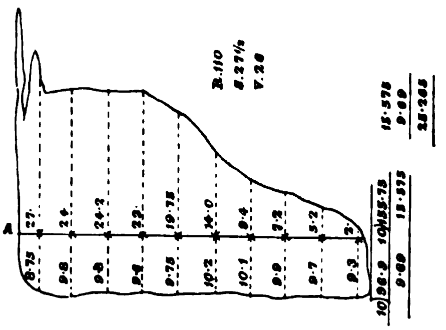

Fig. 668.—Indicator Diagram.

To be strictly accurate in working horse-power of engines a diagram should be taken on both sides of the piston, even where the action of the steam on the one side is repeated on the other, and a mean of the two diagrams adopted as representing the effective steam pressure, but for the sake of clearness the method is here shown with a single diagram.

In diagrams taken from non-condensing engines the line traced will not, of course, at any point descend below the atmospheric line, but will in fact be above it owing to the back pressure of steam on the other side of the piston. Therefore, in measuring the ordinates on a diagram obtained from a non-condensing engine, measure with the scale from the atmospheric line to the upper line traced on the diagram and also from the atmospheric line to the lower line of the diagram. Having obtained the average of each set of ordinates, subtract the one from the other, that is deduct the average back pressure from the full pressure to obtain the average effective pressure. The average back pressure is seldom less than 3 lbs. even in good engines.

Goodeve, in his *Text Book of the Steam Engine*, thus writes of the indicator diagram of a single-acting engine:—

"In the single-acting engine two diagrams must be taken, one from the top and the other from the bottom of the cylinder. These diagrams are quite unlike in form, for the action during the down-stroke is not repeated during the up-stroke as in a double-acting engine, and our first task will be to comprehend the reasons of the particular conformation observed. For this purpose reference is made to a diagram taken from a Cornish pumping engine, having a cylinder 70 inches in diameter and making 4 strokes per minute, under a mean pressure of 15·1 lbs. per square inch. The figure is reduced from one on a larger scale, so that the indicator spring would extend one inch on the reduced diagram for a steam pressure of 40 lbs. per square inch.

"One card is taken from the top and the other from the bottom of the cylinder and each must be interpreted in its turn.

"As far as the upper card, Fig. 669, is concerned, that figure indicates the admission and cut-off steam, together with the opening of the equilibrium valve, which corresponds to imperfect condensation in our normal diagram. The lower card has reference to the state of things below the piston where the equilibrium and exhaust valves are opened consecutively.

"Beginning at the point A, with the piston at rest at the top of the cylinder, we note that the pressure rises until the down-stroke commences, when the steam line B C D is traced out. The portion B C is horizontal and the cut-off takes place at C. It is common for the steam line B C to drop considerably before the cut-off begins, especially in large engines. The line D E indicates that the equilibrium valve is opened, and that the steam pressure has fallen somewhat during the circulation which takes place. At the point E the equilibrium valve is closed and compression or cushioning begins, just as in a double-acting engine. At the point A the piston is coming to rest, and there is a drop in the curve which is often much more marked than in the present example, and which indicates loss of pressure before the down-stroke begins. Such loss would be due to leakage of the compressed steam round the circumference of the piston, or perhaps to loss of heat.

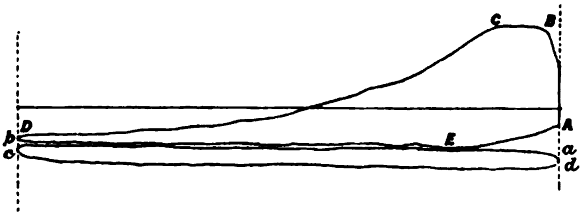

Fig. 669.—Indicator Diagram.

"As to the lower card, the nearly horizontal line, *b a*, shows that the equilibrium valve is opened. When compression begins at E, above the piston, expansion will also begin to much less extent below it, and there will be a slight drop towards the end of *b a*. Otherwise, the lines D E and *b a* nearly coincide, and would do so absolutely, if there were no disturbing causes at work; but the diagram shows some difference of pressure at the two ends of the cylinder when the equilibrium valve is open.

"With regard to work done, the piston is driven down by the steam from above it, as opposed to the back pressure of the exhausted space underneath, and that part of the action is fully determined by comparison of the lines B C D and *d c*. But the whole work done by the steam in the double stroke is, according to our principles, obtained by a careful measurement of the areas of the enclosed figures.

"At first sight, the student might imagine that the horse-power may be calculated by simply noting the pressures indicated by the steam and exhaust-lines, the cutting away of any part of the intermediate area—as by compression or by want of coincidence of the lines D E and *b a*—affecting only the up-stroke when the weight of the pump-rods is the moving force. But a little consideration will show that such a notion is erroneous, and that the compression of steam in the up-stroke, and the resistance to the motion of the piston due to inequality of pressure when the equilibrium valve is open, must be deducted from the total efficiency. The steam opposes the piston in its ascent to some degree, and this gives rise to negative work, which must be deducted from the positive work accomplished in the down-stroke. In other words, during the down-stroke the steam does the work, and during the up-stroke work is done upon the steam.

"It follows, therefore, that the portion of unoccupied space between the two intermediate horizontal lines is a veritable subtraction from the efficiency of the agent.

"We pass on to calculate the horse-power in the case of a single-acting pumping-engine, having a cylinder 112 inches in diameter, with a stroke 9·166 feet, and making 7·5 strokes per minute.

"Referring to the diagram, Fig. 670, where the steam pressures are noted, and taking each group of numbers in order, there is, above the atmospheric line, a series amounting in all to 56·5. Below the atmospheric line the first series amounts to 48·5, and the second series gives 39·3.

"Hence the mean pressure of steam $= \frac{1}{10}(56{\cdot}5 + 48{\cdot}5 + 39{\cdot}3) = 14{\cdot}43$,

$$\therefore \text{H.-P.} = \frac{14{\cdot}43 \times 3{\cdot}14159 \times 56 \times 56 \times 9{\cdot}166 \times 7{\cdot}5}{33{,}000} = 296{\cdot}5."$$

In engines working under a varying load, such as winding-engines, it is useful to have diagrams taken of every stroke made during such variation of load, and for this purpose an arrangement of indicator capable of taking continuous diagrams is now used.

In the improved Richards Indicator made by Messrs. Schäffer and Budenberg of 1, Southgate, Deansgate, Manchester, a screw arrangement can be supplied for raising the paper drum of the indicator by hand for the purpose of taking a number of diagrams in succession. This arrangement is very suitable for colliery winding or hauling engines as a paper of diagrams can be obtained throughout one wind, arranged in the order of taking on the paper. These indicators are very carefully made and each instrument is thoroughly tested before being sent out. The tension of the spring in the drum can be regulated to suit the speed of the engine, by loosening the upper nut on the drum spindle and turning the disc holding the spring until the required tension is obtained, the nut being then again screwed home. The small pulley for guiding the cord is arranged on a swivel joint allowing the pulley to take any convenient position. The indicator is provided with every convenience for examining and cleaning, inserting the cord, &c.

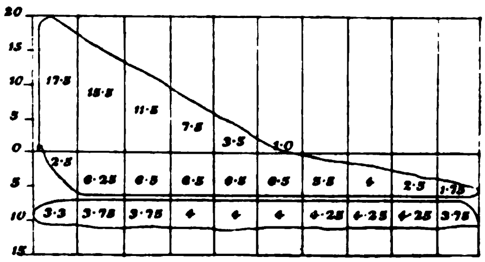

Fig. 670.—INDICATOR DIAGRAM.

Fig. 671 shows an exterior view of the *Thompson Indicator*, made by Messrs. Schäffer and Budenberg, and designed to overcome some objections attending the use of Richards's, by reducing the inertia of the parallel motion, so as to lessen the oscillations of the pencil, and render the instrument applicable at higher speeds.

In the novel parallel motion of the Thompson indicator, the lever carrying the pencil is very lightly made, so as to bend slightly and give the pencil a gentle elastic pressure against the paper; this arrangement does not re-act disadvantageously upon the parallel motion as does that in the Richards indicator.

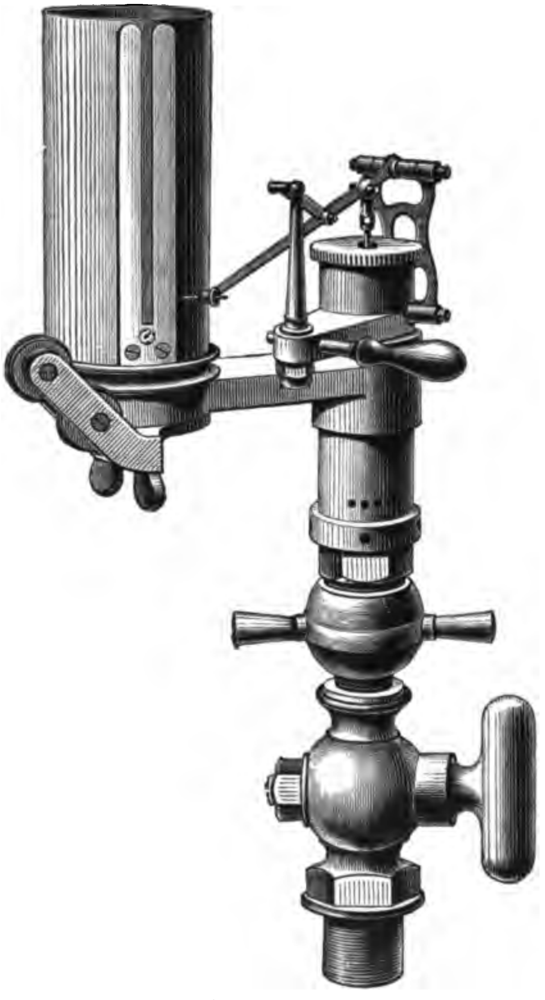

Fig. 671.—The Thompson Indicator.

In Fig. 671 the indicator piston is connected to the main horizontal rod of the parallel motion by a connecting rod, passing inside the piston rod, which is made hollow so as to allow of this. The attachment to the piston is by means of a small double ball joint. This gives a joint which is quite free without having any play and allows of a means of adjustment by taking up any play resulting from wear on the joint. The parallel motion is carefully designed to ensure that the pencil point describes a straight line and that the motion of the pencil point is precisely proportional to the displacement of the indicator piston throughout the stroke. To suit the higher speeds for which the indicator is intended, the passage of the cock and connection is made ½ inch, against ⅜ inch in that of the Richards indicator. The piston motion is multiplied in the proportion of 1 to 4, as is also that of the Richards as made by Messrs. Schäffer and Budenberg. The stroke of the piston is only ¾ inch as against ⅞ inch, the maximum height of the diagram being consequently 3 inches, against 3⅜ inches, in the Richards. In all other respects the Thompson and Richards indicators are similar. The former is also made in a small size, which is intended more especially for taking diagrams at high speeds. In it the parallel motion is made very light, so that the oscillations of the pencil at the highest speeds are considerably reduced; a stronger spring than in the large instrument being invariably used. This instrument complete with fittings weighs only 5 lbs., as against 11 lbs. in the case of the large Thompson, and a still greater weight in the ordinary Richards indicator. Its greater portability gives it an additional advantage over the other instruments. The large Thompson indicator has been successfully employed at a speed of 400 revolutions per minute, whilst the small

Thompson may be used up to 600 revolutions per minute if a sufficiently strong spring is inserted.

If desired the apparatus can be supplied with a rising motion of the drum operated by hand for taking any number of diagrams successively on the same paper.

Messrs. Schäffer and Budenberg also make a *Double Indicator* which is a novelty in construction. In the ordinary indicator, the upper or steam-pressure curve drawn on the diagram does not in reality pertain to the back-pressure curve drawn beneath it, but belongs rather to that back-pressure curve which would result if the other side of the piston were indicated simultaneously; and the calculation from an ordinary diagram of the work done by the engine during a stroke is based upon the assumption that the action is identical during two successive strokes on both sides of the piston.

The object of the double indicator is to be enabled to place both sides of the piston in communication with the indicator at the same time, if desired, and in such a way as to record on the diagram, the actual resultant effective pressure on the piston throughout the stroke. This instrument consists practically of a combination of two indicators acting upon the pencil simultaneously, and causing the latter to record directly the resultant-pressure upon the engine piston. The double indicator has two pistons fixed upon the same spindle, and when both indicator cocks are open, the pistons are subject respectively to the pressures on both sides of the engine piston, the pressure being, of course, in opposite directions in the case of non-condensing engines, and in the same direction in condensing engines. There is only one spring, and this is employed in compression upwards or downwards, according to the direction of the resulting pressure on the engine piston, the arrangement of the spring being such that it can only be used in compression. The diagram traced by this indicator is therefore situated partly above and partly beneath the atmospheric line, and the distance of the diagram from the atmospheric line at every point represents the actual resultant-pressure on the piston. The areas of the diagram above and below the atmospheric line show respectively the work of the steam on each side of the piston, and the area of the whole diagram represents the actual work performed during one revolution. If only one indicator cock be opened at a time, separate diagrams from each cylinder end may be obtained, and by taking these on the same paper as the combined diagram, a complete record is obtained which shows all the details of the separate diagrams, and facilitates the calculation of the whole diagram.

A peculiarity of the double indicator is that its indications are independent of the barometric pressure, the piston rod being equilibrated for this pressure, whilst in all ordinary indicators the atmospheric pressure acts upon the upper side of the piston.

The instrument has not become very popular owing to the diagrams obtained from it differing to such extent from those of the ordinary indicator as to puzzle the occasional user.

PRESSURE AND VACUUM GAUGES.

The first form of gauge used to determine steam-pressure was the mercury column, and it is still used as the standard of comparison when graduating existing gauges. The indications of the mercury column are most accurate and reliable in careful hands, but it is inconvenient for general use, and impossible in cases where there is oscillation, as on locomotives. Fig. 672—the ***Bourdon***—

shows the type of gauge now in general use, which is the invention of M. Bourdon, a Frenchman.* It is stated that the idea occurred to him while repairing the copper pipe of a still which had become flattened. In trying to restore it to its normal shape he forced water into it under high pressure, and observed that during the process the flattened tube had a tendency to uncoil itself, and this was the more apparent as the pressure of the fluid used was increased. This principle he afterwards applied in the Bourdon pressure-gauge by using in it a flattened tube of hard elastic brass bent upon itself, closed at one end, and open to receive the fluid-pressure acting along the interior at the other. As the pressure fluid enters the tube it unbends, and the extent of unbending depends on the amount of pressure applied. A standard mercury column was used in order to ascertain the pressure equivalent to different amounts of unbending, and an indicating pointer attached showing the extent of unbending or the corresponding pressure, and the gauge as thus made is convenient for carrying and for general use. When carefully made its indications are accurate and reliable.

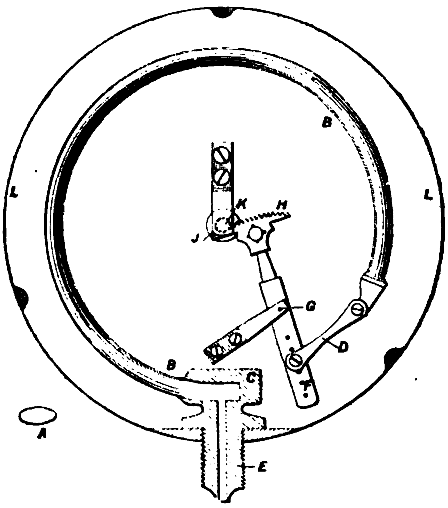

Fig. 672.—The Bourdon Pressure-Gauge.

In Fig. 672 an ordinary form of pressure-gauge is shown partly in section. The brass pressure tube, B, shown in transverse section at A, is securely soldered to the piece, C, terminating the screw, E, and is closed at the other end. The steam or other fluid under pressure is admitted by the small opening in E, and fills the tube, B. As a result, the tube tends to straighten itself, and a link, D, pulls upon a lever, F, which moves on the fulcrum, G, and by means of a toothed rack, H, rotates a small pinion, J, and with it the spindle, K. The spindle, K, carries the pointer, which moves round the dial, and indicates by its position the pressure of the steam or other fluid. It will be observed that a very small movement of the tube, in the effort to unbend, is increased to a large movement at the end of the pointer, and for that reason it is highly important that all slack or back-lash should be avoided as much as possible. It is equally necessary that friction be reduced to a minimum, or the pointer will not indicate the same during the return motion, *i.e.*, during a reduction of pressure. In the ordinary form of these gauges the brass case, L, is used as a framework upon which to attach the various parts, and when well made is quite satisfactory. Where, however, the fastenings are in the least degree insecure, such gauges are likely to give inaccurate readings. If the screw, E, be the least slack in the case, it allows movement of the tube, B, by vibration, or by accidental push or touch, and consequently may indicate pressure when none exists, or on the other hand may require a considerable pressure before it indicates anything at all. Any shifting of the brackets carrying the fulcrum, G, or the spindle, K, may also seriously affect the pointer indication. In the gauges manufactured by Messrs. Schäffer and Budenberg, of Manchester, no part of the mechanism is attached to the case, but is all carried on one piece of metal, cast solid with the screw, to which the Bourdon tube is attached. This method avoids possibility of displacement due to defective fastenings, or inaccurate readings, if the screw should shake in its case.

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* The Germans say that it is the invention of a German, Schinz, and in that country the tubes are known by his name.

Figs. 673, 674, and 675 show sectional views of such gauges, Fig. 673 being a concentric, and Fig. 674 an eccentric, Bourdon pressure, or vacuum-gauge, while Fig. 675 is a steel tube gauge specially constructed for indicating high steam, gas, or hydraulic pressure.

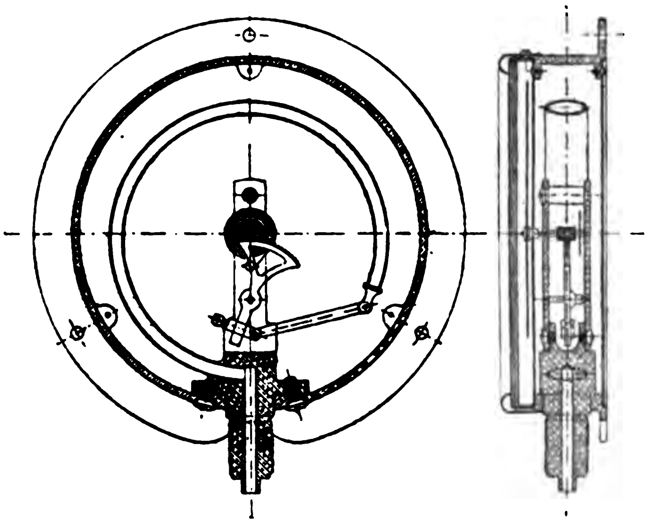

Fig. 673.—Schäffer and Budenberg Concentric Bourdon Pressure or Vacuum-Gauge.

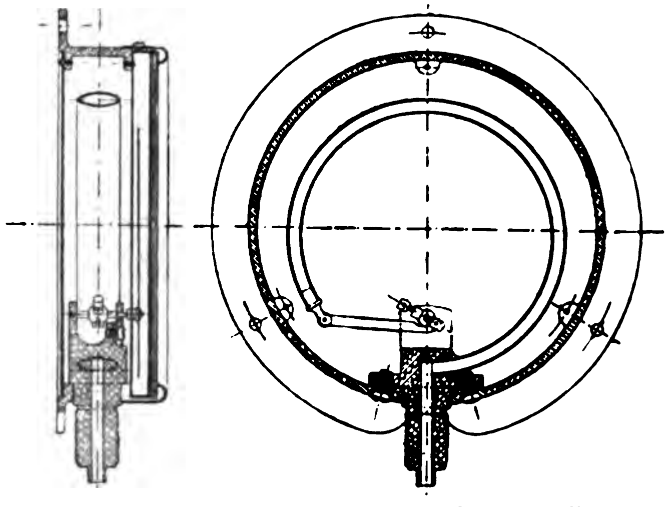

Fig. 674.—Schäffer and Budenberg Eccentric Bourdon Pressure or Vacuum-Gauge.

In Fig. 673 the pointer-spindle is set in motion by means of a pinion and toothed quadrant. In Fig. 674 no toothed quadrant is used, the end of the

Bourdon tube being attached direct to a lever on the pointer-spindle. In Fig. 675 the toothed quadrant is retained, but the Bourdon tube is constructed of steel instead of hard brass. These gauges were originally only intended for hydraulic purposes, but the principle has been so far modified as to allow of its being applied to gauges for all ordinary steam pressures, and they register the lower pressures with as great accuracy as the higher ones. When intended to indicate hydraulic pressure a tube nearly circular in section is adopted; in the steel-tube gauge suitable for steam pressures the tube is flattened considerably so as to be more flexible.

Ordinary steam-pressure gauges with brass tubes are unable to indicate constantly such high pressures as 300 lbs. per square inch, and still more so pressures very much higher which are needed for high pressure tubing, hydraulic presses, &c. In Fig. 675 the type of gauge is designed to overcome these difficulties, and may be made to indicate different ranges of pressures varying from 120 lbs. to 30 tons on the square inch. The springs are manufactured from English-made steel rods of the highest quality attainable, the rods being cut to the proper length of the springs required; they are bored out, and turned off outside, except the ends, to which a suitable thread is screwed. The straight tubes are then heated to a certain temperature, and bent, the bending operation being followed by a flattening and shaping process. After tempering, one end is screwed into the stem or principal part of the gauge, through which the pressure is carried on from the generator, and upon the other end is screwed a solid piece of metal or cap, to which the rod is fixed, transmitting the movements of the steel spring by suitable gearing to the pointer and dial of the gauge.

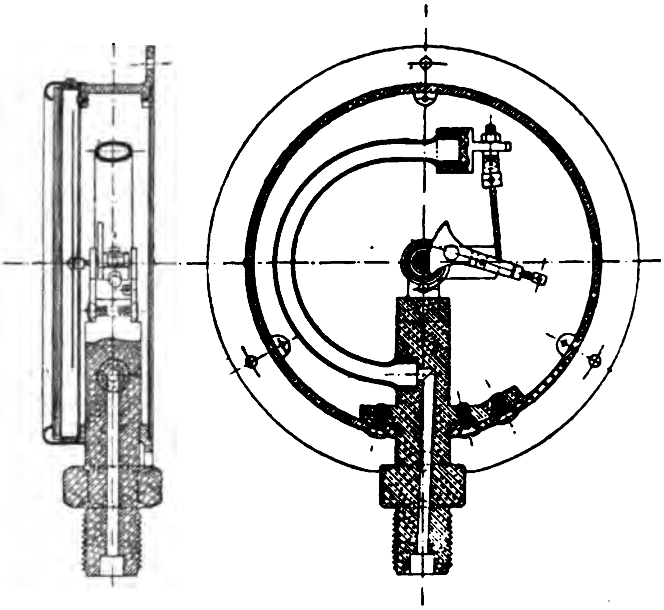

Fig. 675.—Schäffer and Budenberg Steel Tube Gauge.

Although the steel tube springs stand perfectly well without corrosion, if constantly under water, they cannot resist it when intermittently employed, as this allows air to be admitted, following the water, and results in signs of corrosion. This difficulty has been overcome by timing, for which process

Messrs. Schäffer and Budenberg have obtained a patent. By it both the interior and exterior of the steel tubes are plated with tin. They thus effectually resist corrosion, and may be used as test gauges with any required intervals of time between periods of employment. They are exceedingly durable, retaining perfect elasticity, and have the great advantage that all soldering is avoided, which is requisite in the Bourdon, and other usual constructions in which the spring consists of a brass tube.

These high-pressure gauges, in addition to their application to hydraulic presses, are exceedingly useful in ascertaining the breaking strain of iron and steel bars, chain cables, water pressure in accumulators, air compressors, as also in connection with the manufacture of torpedoes, for which purposes they are used at the Arsenal at Woolwich, Sir William Armstrong & Co.'s, and Sir Joseph Whitworth & Co.'s, and other works.

Duplex gauges have two complete and perfectly distinct gauge works and tubes communicating with the same inlet, and they therefore check their own indications. They are made in two forms by Messrs. Schäffer and Budenberg, in one of which, adapted for a steel tube test gauge, the two pointers, when at zero near the top of the gauge, are some distance apart, and move in opposite directions, when operated by pressure from the generator. The dial face is divided into two corresponding even divisions, each being contained in rather less than a half circle on the dial, and numbered up to the range of pressure the gauge is made for. The two pointers should always indicate the same pressure. In the other form the pointers overlap each other, so as to indicate on the same scale, the arbor of one of the pointers being made hollow, while the other is passed through the former.

The dial plate of each ordinary gauge made by Messrs. Schäffer and Budenberg, is graduated separately, and by actual comparison with a standard mercury column. When made, the gauge is left with a blank white dial, and in this state is screwed to a connection leading to an open mercury column and a force pump. Motion is given to the pump, which forces up the mercury on a measured scale, and on which is marked the number of pounds per square inch, corresponding to the height of the column. The column ascends the whole height of the building, and the position of the mercury is read by assistants placed at various levels. Marks are made upon the dial of the gauge, corresponding to every increase of 1 lb. pressure, and the dial is afterwards finished with the graduations as thus determined experimentally.

For the purpose of comparing different gauges a hydraulic cylinder is used, the piston of which is actuated by a screw and hand-wheel. The gauges are attached to couplings prepared for them, and their indications may be compared with a standard tested gauge by merely turning the hand-wheel, and forcing the fluid used (preferably oil) into each.

The great height which would be required for the purpose, renders it impracticable to test the steel tube gauges by comparison with a mercury column; they are, therefore, tested by pumping up the pressure on the gauge, and against a loaded ram, having a definite known area and load. Special arrangements are made to reduce the friction to a minimum, and allowance is made in the graduations for the friction, the amount of which is determined by the aid of the mercurial column.

Bourdon vacuum gauges are constructed on the same principles as pressure-gauges, but are distinct in consequence of their more delicate and sensitive character, designed to suit the slight range of low pressures to be indicated. In their construction the tube springs are made lighter and much flatter in section than those used in the pressure-gauges. It is of the utmost importance to reduce friction to a minimum, and for that purpose, multiplying gear is dispensed with;

the link from the end of the tube spring being attached direct to the lever on the pointer spindle similar to Fig. 674. To avoid oscillations resulting from slight vibration, the Bourdon tube must be proportioned so that small changes of pressure produce ample power to move the indicating gear and pointer. In a properly made and proportioned instrument the pointer should always come back to zero on being released from the pressure deflecting it, however slowly it is moved, and if when pressure is on the gauge, it be deflected and then allowed to return, it should invariably settle at the true pressure. The most trying test is when the deflection of the pointer is slow, followed by a slow return. To test vacuum gauges, they are placed on a stand alongside a tube immersed in mercury and an air-pump. A scale of inches is attached to the mercury tube and the air-pump is connected to a coupling for carrying the gauge and also to the top of the mercury tube. On starting the air-pump the air is removed from the gauge and above the mercury and the scale of inches above the mercury level shows the vacuum.

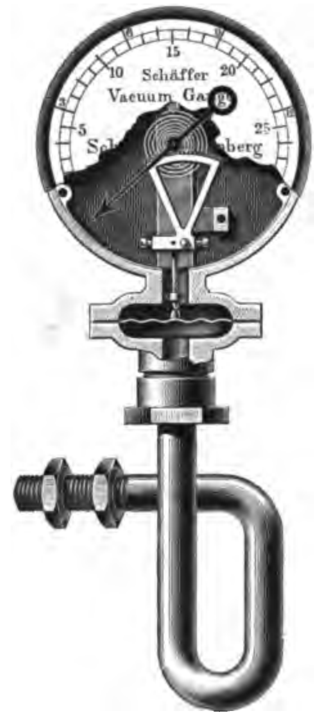

Fig. 676.—Schäffer Diaphragm Gauge.

Beside the Bourdon, Messrs. Schäffer & Budenberg make a diaphragm gauge known as the *Schäffer*, and shown in Fig. 676. This gauge was patented by Schäffer as early as 1859, *i.e.*, prior to the invention of the Bourdon gauge, and is available for pressures up to 300 lbs. per square inch. Its most noticeable feature is the corrugated steel diaphragm shown in the drawing, which serves to receive the pressure of steam or a fluid. In pressure-gauges the steel diaphragm is backed up by a spring thrusting against the steam pressure, and the movement of the diaphragm is communicated to the pointer by the lever-toothed quadrant acting on a pinion attached to the pointer spindle. The helical spring surrounding the spindle, is intended to take up all backlash, by keeping the pinion teeth always bearing upon one side against the quadrant teeth. To protect the steel diaphragm from the corrosive action of the steam it is covered by a thin pliable sheet of silver foil shaped to fit the corrugations. This sheet is so thin as to allow free movement of the diaphragm whilst effectively protecting it from corrosion.

LIGHTNING DESCENDING SHAFTS.

A danger to which collieries are liable is the descent of lightning to the underground workings during a thunderstorm. Well authenticated instances have been placed on record which leave no doubt that lightning has been observed to traverse the underground workings; but in all instances without serious damage having resulted from it. Some mines are so placed that they seem to invite the descent of lightning into the workings. The colliery may be situated on elevated ground, the underground workings be dry, and the ropes, rapper

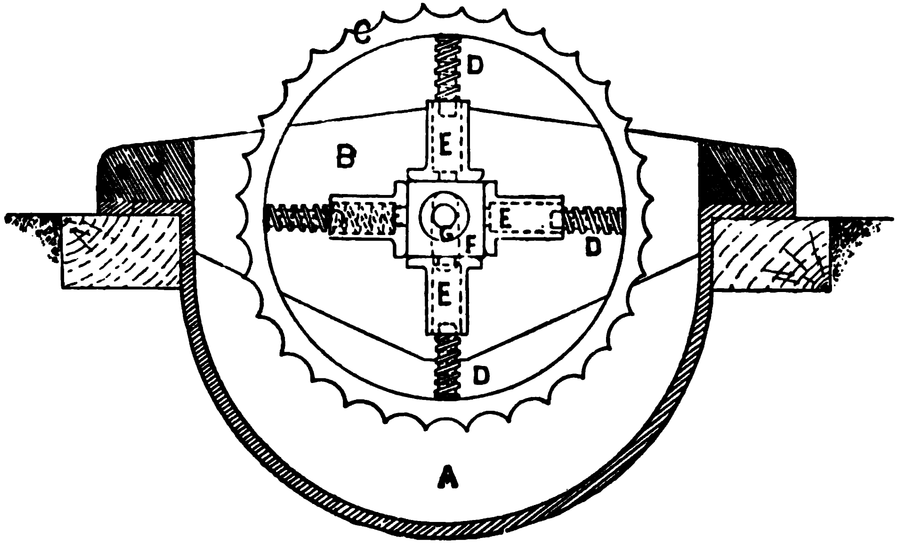

Fig. 677.—Section.

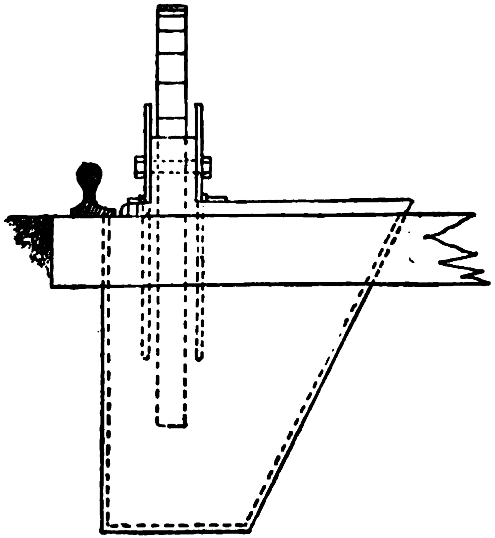

Fig. 678.—End View.

Dunford and Emens Patent Automatic Greasing Apparatus.

wires, iron guides, steam or water pipes afford the means of conducting the electric spark to the flat sheets at the landing stage, and these again being connected with the rails leave an unbroken course for the lightning, especially if the rails are "fished" at the joints. Whether it is possible for the lightning to fire an explosive mixture of air and gas in the workings is not considered certain although surely highly probable. The fact that an explosion occurred at Risca some years ago during a thunderstorm has given rise to some discussion on the subject.

To prevent the possibility of lightning discharges entering pits, the mouths of the shafts should be efficiently protected, and this may be done by fixing lightning conductors to all lofty chimneys and buildings in the neighbourhood, and these conductors should terminate with a good earth-plate sunk in the ground, or preferably in the bed of a stream or a reservoir of water. The lightning conductors should, of course, be placed sufficiently above the chimneys or buildings. If these are at some distance from the shaft it may be advisable to fix a conductor at the headstock of the pit, taking care to carry it as high as possible and to have the end carried to an earth-plate.

AUTOMATIC GREASING APPARATUS.

Many forms of tub-greasers have been designed with a view to save labour in applying the necessary lubricant to colliery tubs, and to avoid wasting the lubricant in the act of its application. Some of them do their work most inefficiently, and are liable to get out of order and damage the rolling stock they are intended to lubricate. There can be no doubt, however, that a well-designed automatic tub-greaser is calculated to effect a considerable saving at large collieries. The following description of Dunford and Emens' ***patent automatic greasing apparatus*** (Figs. 677 and 678) is given by Messrs. Dunford Brothers, of Newcastle-on-Tyne:—

"A—Semicircular trough, fixed to the sleepers, to contain the grease, with bevelled side to incline the grease towards the wheel. B—Two side plates to carry the wheel, with slots to allow of the wheel being easily raised or lowered. C—Corrugated wheel, with 4 short arms. The wheel revolving in the grease, raises in its corrugations a sufficient quantity of grease to efficiently lubricate the axles. D—Four best tempered steel spiral springs, which hold the wheel in position, and allow of its eccentric action. E—Four cases, to hold the springs, the short arms of the wheel being inside the springs and below the level of the top of the cases. F—Square boss, on which the spring cases slide, allowing the wheel to assume an eccentric position when struck at an angle. G—Centre piece, on which the square boss revolves, allowing the wheel to have a rotary motion; a bolt and nut passing through the centre piece holds it firm to the side plates.

"The apparatus is intended to be placed in a convenient situation, one on each side between the rails of the tramway, so that each tub of a train passing over it in either direction will receive on the journals of each of its axles an adequate quantity of the grease lubricant, any ***excess*** being ***returned*** to the ***reservoir***.

"It consists of a semi-circular trough containing the lubricant, the upper end of which is level with the ground, and covered in to keep out dirt or coal dust, except the space through which a wheel works, the rim of which is in breadth suitable for the axle journal, and has formed across it, all round, concave recesses which contain the lubricant. The rim of the wheel, instead of being rigidly connected to the boss or centre by arms, has an elastic connection by means of four spiral springs enclosed in tubular cases interposed between the rim and the boss (which has four flat faces), upon which the feet of the tubular cases enclosing the springs slide freely, the result being that when the axle of a running tub comes into contact with one of the concave recesses in the rim, the journal receives a charge of lubricant, and, besides revolving the rim, pushes it in the direction in which it is running, the springs yielding, and the feet of the enclosing cases sliding on the flat faces of the boss, the rim being pushed temporarily into a position eccentric to the centre of its axle till the tub axle has passed over, when it returns to its normal position, to be acted upon in the same manner by the next following axle, which receives a fresh charge of lubricant. Thus the rim will suit itself to tubs having wheels of different diameters to the extent of several inches.

"The greasing wheel on one side of the line is entirely unconnected with the wheel on the other side; each acts freely to administer a charge of grease to the journal on its own side of the line of tubs.

"Where the tub wheels revolve on the axles this greaser is useless, but is very efficient for tubs to which the wheels are fixed to the axle and the whole revolve."

The Hardy Patent Pick Company, Limited, Sheffield, supply a *self-lubricating pedestal* for colliery tubs which is designed also to protect the axles of the tubs from dust. This is of much importance, especially in dusty mines, where ordinary bearings get full of coal dust which renders useless the newly applied lubricant. The construction of the bearing is very simple. The top portion bearing the load is similar to the old form of pedestal, but underneath a steel dish is fitted, which keeps the axle free from dirt and at the same time prevents it from leaving its bearing. The steel dish is stamped out of one sheet of metal, and is shaped to hold a quantity of felt or wool, which bears against the axle on the under side, and being soaked with the lubricant, keeps the journal well oiled. When first charged about 5 oz. of oil is required in the operation, but when necessary to replenish, a charge of 2 oz. is sufficient. It is said that the tubs will work two or three months without further attention, after once oiling, and the journals remain clean and well-oiled throughout that time. A tub with outside bearings may have the lubricant applied to the journals at any time as the tub rests on its wheels, but with inside bearings it is necessary to turn the tub by means of a lifter and then the oil may be poured through a hole which leads into a large hollow space in the top of the bearing plate. On returning the tub to its usual position on its wheels the oil runs down into the wool or felt.

CHAPTER XVII.

COLLIERY EXPLOSIONS.

Early Explosions—First accurate Statistics—Early Investigations into Causes of Explosions—Table of some Notable Explosions—Public Inquiry into the cause of an Explosion and Reports to Parliament thereon—Possibility of some Coal-dusts being Inflammable—Different Ways in which Fire-damp is Ignited—Pressure resulting from an Explosion and its Effect—Absence of Force at Point of Origin—Later Development—Relief at Shafts, usually—Direction taken by Blast—Deposits of Coked Coal-dust left by Explosion—Charred Timbers and other Evidence of its Passage left by Explosion—Damage at Downcast Shaft from large Explosion—Derangement of Ventilation by Blast—Work Necessary for Recovery of Workings—Anomalies of Evidence—Conflicting Evidence—Disagreement of Experts as to Cause and Direction of an Explosion—Complex Explosions—Evidence of great Force exerted by some Explosions—Underground Fires Caused by Explosions—Coal-dust a Source of Danger—Its Influence in Intensifying an Explosion—Explosions restricted to Mines containing Fire-damp—Reasons for Supposing Coal-dust has aggravated Explosions—Necessity for systematic Damping of Dusty Roadways—Prevention of Accumulations of Coal-dust on Main Roads—Influence of very Slight per centage of Fire-damp on Atmosphere impregnated with Coal-dust—Prohibition by Mines Act, 1887, of firing Gunpowder Shots in Dusty Mines unless previously watered—Accumulations of Coal-dust in Longwall and in Pillar and Stall Working—Where most Dangerous Coal-dust is found—Chemical Changes from Exposure of Coal-dust to Atmosphere—Difficulties in Conducting Experiments with Coal-dust—Necessity for the extension of knowledge on Coal-dust—Coal-dust Explosions on the Surface—Some Particulars of the Altofts Explosion—Difficulty to Dogmatize as to a Coal-dust Explosion—Possibility of Fire-damp reaching Roadways in a Longwall Working through Fissures and so causing Explosion—Explosions in wet Mines—Possible Uninflammability of the Coal-dust in the Radstock and Forest of Dean Mines—The most likely part for an Explosion—Order of Firing a Series of Shots—Firing Shots between Shifts—Remedial Measures.

In spite of the vast improvement in colliery management, explosions continue to occur from time to time, as they have done since the early days of mining. With difficulty they can be traced as having occurred in the British Isles as far back as the middle of the seventeenth century. Mining records were not officially kept until the commencement of 1851. Since that time Statistics and Reports have been carefully prepared by Inspectors appointed for the purpose.

But few workmen were employed in the first mines. As the industry grew and assumed more important proportions, the liability to serious loss of life through accident also increased. Thus in 1708, when women as well as men worked underground, a colliery explosion occurred at Fatfield which resulted in the death of 69 persons. The following quaint account of the disaster, by the Rev. Dr. Arthur Charlett, is given in the *Transactions of the Royal Society of London* :—

"On Wednesday, the 18th day of August, 1708, at Fatfield, in the parish of Chester-le-Street, about 3 o'clock in the morning, by the sudden eruption of a violent fire, which discharged itself at the mouths of three pits with as great a noise as the firing of cannon, or the loudest claps of thunder, 69 persons were destroyed in an instant. Three of them, viz., two men and a woman, were blown quite up from the bottom of the shaft, 57 fathoms deep, into the air, to a considerable distance from the mouth of the pit; one of the men with his head almost off, and the woman with her bowels hanging about her heels. The machine by which the coals were drawn up, and it is of a great weight, was blown off by the force of the blast ; and what is more wonderful, the fish which were in the rivulet that runs 20 yards under the level, and at as great a distance from the mouth of one of the pits, were in great numbers taken up dead, floating on the water, by several of the inhabitants."

About 1710 a colliery explosion occurred at Bensham resulting in the loss of from 70 to 80 lives.

Although the number of mines increased and the workings became more extensive as time went on, these two explosions were the most serious recorded until the present century. In March, 1767, 39 lives were lost at Fatfield and in October, 1799, a similar number were lost at Lumley.

An explosion occurred at Hebburn Colliery, in October, 1805, which destroyed 35 lives, and another in November, 1805, at Exclose, resulted in 38 deaths.

A much more disastrous explosion than any previously recorded, occurred at the Felling Pit, in Durham, whereby 92 lives were sacrificed in May, 1812. The magnitude of this disaster attracted considerable attention, which led to the formation of an association at Sunderland, the object of which was to guard the miners' interests. Nevertheless explosions continued with unabated force and frequency.

In 1829 the House of Lords began to investigate the cause of accidents in mines, and published their first report in 1833.

On June 18, 1835, a very disastrous explosion occurred at Wallsend Colliery, which caused the death of 102 persons.

Various efforts to secure information about these disasters were made by Select Committees and Royal Commissions, with a view to taking precautions against such catastrophes. Faraday and Lyell conducted an exhaustive inquiry into the Haswell Colliery disaster which occurred in September, 1844, and resulted in 95 deaths. These very careful investigators presented their report to the Home Secretary in 1845. In it they stated their opinion that the destructive force of the explosion was not due entirely to the ignition of a mixture of firedamp and air, but that it was greatly increased by the fine dry particles of coal-dust which had collected on the collars, props, sides and floor of the mine.

Following closely on this came the appointment of Government Inspectors towards the end of 1850, whose careful supervision has done much to improve collieries generally, but has not put an end to explosions.

The most notable explosions that have occurred since then, each involving the loss of 100 lives or more, are shown in the following table:—

Name of Colliery.	Date of the Explosion.	No. of Lives Lost.
Cymmer	March, 1856 . .	114
Lund Hill	19th February, 1857 .	189
Risca	1st December, 1860 .	130
*The Oaks	12th December, 1866 .	334
Ferndale	November, 1867 . .	178
Swaithe Main	6th December, 1875 .	143
Blantyre	22nd October, 1877 .	207
Haydock	7th June, 1878 . .	189
Abercarne	11th September, 1878 .	268
Risca	15th July, 1880 . .	120
Seaham	8th September, 1880 .	164
Penygraig	10th December, 1880 .	101
Pendlebury, near Manchester . .	18th June, 1885 . .	178
Llanerch	6th February, 1890 . .	176
Park Slip, Aberkenfig, near Bridgend .	26th August, 1892 .	112
†Combs Pit, Thornhill Collieries, near Dewsbury	4th July, 1893 . .	139
Albion Colliery, Cilfynydd, near Pontypridd	23rd June, 1894 . .	290

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* 334 were killed by the first explosion and 27 afterwards by closely following explosions, making a total of 361.

† A slight explosion of firedamp fired some shaft fittings: the smoke and afterdamp carried into the workings caused the 139 deaths.

Explosions stand out in bold contrast to other sources of mining disasters in consequence of their frequently appalling nature and widespread devastation. It is true that large numbers of lives have been lost in single accidents other than explosions, such as the inundation of Heaton Colliery, Northumberland, on the 3rd May, 1815, by which 75 lives were lost, and the accident at Hartley Colliery, in Northumberland, on the 16th January, 1862, when 204 persons were fatally entombed by the breaking of a pumping-engine beam, half of which fell into and closed the single shaft by carrying with it the shaft fixtures. This accident led to the prohibition of single shafts at large collieries. More lives have been sacrificed, however, in single catastrophes by explosion than by any other cause. The most disastrous ever recorded was that at the Oaks Colliery, Yorkshire, on the 12th Dec. 1866, when 334 men and boys were killed, which was followed by other explosions on the next day, when 27 explorers met their death. On the same day, the 13th, another explosion occurred at Talk-o'-the-Hill Colliery, in Staffordshire, at which 91 men perished.

The annual loss of life by explosions is less, on the average, than by falls of roof and sides, accidents which, although of daily occurrence, seldom cause the death of more than one or two persons at once, and on this account are unheard of outside their immediate locality. In only four out of the past forty years have the deaths by explosions exceeded those by falls of roof and sides, viz. in 1857, 1866, 1878, and 1880. Naturally it is in those years and on occasions when many lives have been lost by a single explosion, that public attention has been awakened, rather than at other times when, though the death rate was larger, there was less to strike the popular imagination. But it is an astounding fact, that during this period of forty years the aggregate annual loss of life has only twice been below 900, viz. in 1864 and 1888.

The following table shows the number of deaths which have occurred in coal mines from all causes, and the proportion due to explosions, falls of roof and sides, and miscellaneous causes, for the period 1851–90:—

Number of deaths from explosions at collieries from 1851 to 1890. Inclusive.	Per cent. of the total.	Number of deaths from falls of roof and sides from 1851 to 1890. Inclusive.	Per cent. of the total.	Number of deaths from miscellaneous causes, above and below ground from 1851 to 1890. Inclusive.	Per cent. of the total.	Total number of deaths from all kinds of accidents from 1851 to 1890. Inclusive.
9,055	21·4	17,047	40·3	16,177	38·3	42,279
Averages per year 226		426		405		1,057

It may here be noted that only one or two explosions have been known to occur in metalliferous mines; and thus the ratio of deaths from falls of roof and sides to the total number is very much greater than in coal mines.

A comparison of the above with the following table will show that the deaths by explosions have not been so numerous during the decade 1881–90, as in former years, and that improvement has been made in the prevention of explosions. If we remember that the number of persons employed is considerably greater now than formerly, it will be evident that the improvement is even greater than would appear from the tables.

TABLE SHOWING THE NUMBER OF DEATHS WHICH HAVE OCCURRED IN COAL MINES FROM ALL CAUSES, AND THE PROPORTION DUE TO EXPLOSIONS, FALLS OF ROOF AND SIDES, AND MISCELLANEOUS CAUSES FOR THE DECADE 1881-90.

	Number of deaths from explosions at collieries from 1881 to 1890. Inclusive.	Per cent. of the total.	Number of deaths from falls of roof and sides from 1881 to 1890. Inclusive.	Per cent. of the total.	Number of deaths from miscellaneous causes, above and below ground from 1881 to 1890. Inclusive.	Per cent. of the total.	Total number of deaths from all kinds of accidents from 1881 to 1890. Inclusive.
	1,661	16·2	4,609	44·8	4,016	39·0	10,286
Averages per year	166		461		402		1,029

TABLE FOR THE YEAR 1891.

Number of deaths from explosions at collieries for the year 1891.	Per cent. of the total.	Number of deaths from falls of roof and sides for the year 1891.	Per cent. of the total.	Number of deaths from miscellaneous causes, above and below ground for the year 1891.	Per cent. of the total.	Total number of deaths from all kinds of accidents for the year 1891.
51	5·2	476	48·6	452	46·2	979

The year 1891 was one of the most favourable ones since accurate statistics were compiled, there having been only 51 deaths caused by 23 separate explosions. In 1888 the same number of deaths also occurred from 15 separate explosions, the smallest number on record in one year.

In the year 1892, 12 separate explosions caused 123 deaths. A single explosion, viz., the Park Slip, resulted in the loss of 112 lives, no other serious explosion having occurred in that year.

The largest number of deaths occurred in 1866, when 73 explosions caused the loss of 654 lives.

The average loss of life from explosions during the period 1851–91 (inclusive) was 223; and the average number of explosions during the same period was 51 per annum.

It is evident therefore that for many years colliery explosions have been the subject of the most anxious thought and investigation. If it had been an easy one to deal with, explosions would be things of the past, but in spite of all legislative and other precautions, they are still frequently with us and demand for their prevention the sleepless vigilance of all concerned.

The position of the colliery manager immediately after an explosion is indescribably trying, but he may always count upon the sympathy and willing help of neighbouring managers, mining engineers, and Government inspectors, in the arduous work of rescue and the recovery of the mine.

Shortly afterwards, if deaths have occurred, a searching public inquest is instituted in order to ascertain the cause. Government inspectors of the district, and occasionally the chief inspector from a neighbouring district, will attend and bring to bear on the calamity the skill and experience of experts. The results of their investigation are not only stated at the inquest, but are subsequently communicated to the Home Office in the form of a Parliamentary Report. The Home Office is frequently represented as well by counsel, who also reports on the matter, and especially as to whether any contravention of the Act has taken place. Should the catastrophe have resulted from any such contravention, or should the colliery have been worked at the time carelessly, the Government then institutes a prosecution of the offending parties. The very carefully prepared official reports are the best sources of information that can be obtained on the subject of explosions.

Until comparatively recent years all explosions were accepted as being, broadly, the result of the ignition of fire-damp, on mixing with the atmosphere of the mine—such fire-damp having either (*a*) accumulated gradually; or (*b*) having been suddenly given off, as in a "blower," where the quantity is small; or (*c*) from an "outburst" where the quantity is large.

Lately the impression, rightly or wrongly, has been gaining ground that many, if not all the serious explosions have been the result of the ignition of fire-damp and coal-dust, and that their severity has been owing to the latter rather than the former.

Another matter which has increasingly occupied the attention of experts, especially since about 1870, is the fact that a considerable number of serious explosions have occurred simultaneously with the firing of a shot, thus raising the general question as to the safety of blasting in fiery mines.

Probably the most convincing argument in favour of the theory that coal-dust is the principal factor in causing explosions of magnitude, is the fact that several such explosions have been almost entirely confined to the intake airways, which, although generally the most dusty roads in a mine, are those in which fire-damp is least liable to be found.

So far as experiments have been made in this and other countries, it has at least been conclusively proved that a small percentage of fire-damp, harmless in itself, even in the presence of a shock accompanied by flame, will actually explode under a similar shock in a gallery thickly strewn with coal-dust.

Hence it is that coal-dust is generally regarded as an element of danger in all mines more or less fiery.

Some, indeed, are of opinion that an ordinary atmosphere with the addition of coal-dust may, under certain circumstances, be inflammable—a theory which has now been confirmed by the Camerton explosion, and by Mr. Hall's experiments: see remarks on *Coal Dust*, at pp. 700, 701.

The whole question, however, as to the effect of coal-dust in a mine has assumed such importance, and expert opinion upon it is so conflicting, that a Royal Commission has now investigated and reported fully on the subject of coal dust explosions.

In enumerating the causes of explosions, we may add to the above-named (*a*), (*b*) and (*c*), where fire-damp forms a factor, a fourth (*d*) in which fire-damp and coal-dust are combined, and a fifth (*e*) in which it may be caused by coal-dust alone, under such extraordinary conditions, however, as are seldom likely to be present during the ordinary working of a mine.

Any of the above, (*a*), (*b*), and (*c*), when mixed with air in proper proportions, may, of course, be easily ignited by—

(i.) a naked light,
(ii.) a defective safety-lamp,
(iii.) the flame from a shot,
(iv.) the flame from any burning material.

(*d*) may also be ignited by any of these means if the fire-damp is present in sufficient quantity to form an explosive mixture with the air; but, even if not, it may be ignited by (iii.), or by (v.) the flame from another explosion if the fire-damp formed so small a proportion as 2 or 2½ per cent. of the atmosphere: (*e*) may be ignited by (iii.) and (v.), but it seems highly improbable that it can be by any other means, excepting perhaps that of (iv.) under very extraordinary conditions, such as a fiercely burning fire.

Other possible causes of ignition of fire-damp are the descent of lightning into the workings, sparks produced by picks or even by falling stones, electric sparks, and those caused by friction of materials, all of which, although remote, are sufficiently important to require their being guarded against as far as practicable.

Wherever an explosion of fire-damp occurs, a very high temperature is immediately generated, and although we can only conjecture what happens in the mine afterwards, it is probable that great pressure proceeds in every direction from the point of origin. As the closely-packed or hard sides of a roadway are able to resist the pressure, the force is spent on the roadway itself as it proceeds in an outward direction, and, unless the seat of the explosion is the face of a *cul de sac*, also in an inward direction. Unless the fire-damp present is very small in amount, the pressure, owing to its sudden expansion on ignition, is sufficient to form a block to the incoming air in that section of the mine, and the intake airway between the point of origin and the downcast shaft remains for the moment charged with air whose motion has been stayed. If, then, there be little fire-damp at the initial point of ignition, and the explosion is not extended by coal-dust or by further explosions of fire-damp, it will soon exhaust itself without doing much damage. But if the fringe of a large volume of fire-damp (the bulk of which is not of itself explosive owing to want of oxygen) be ignited, and there are abundant facilities for its becoming rapidly mixed with air, as will be the case where a large current is passing through an airway of ample section, then the explosion is certain to be very serious. Similarly, if from the gradual accumulation of fire-damp and its admixture with the air either by diffusion or mechanically, a considerable volume of explosive mixture has been formed, a more violent explosion would be the immediate result, even though a smaller quantity of fire-damp might have been exploded. If to either of the above conditions be added long dust-strewn passages in unbroken communication, the resultant explosion would probably be indefinitely aggravated. Experience in minor explosions, and observations after colossal ones appear to show conclusively, that an explosion does not assert its full force at the point of origin, but that it requires a certain time to develop its strength. Generally, little damage is to be seen at the initial point, but from 50 to 100 yards away the explosion-wave attains its maximum, and this may continue over a long distance before varying in intensity and eventually dying away. As a proof of the absence of force at the point of origin, it may be stated that after an explosion, workmen's tools have been found at that point set up at the side of the road as usual, no fall of roof either having occurred there. The violence exerted by an explosion is due, as we have said, to the sudden expansion of the air and gases from the heat of combustion in a more or less confined space. The expansion begins, no doubt, directly the explosion is initiated, but for a brief period it is only sufficient to compress the air in the ramifications of the mine, then possibly to form currents of its own, some of which would be turned from their previous direction and finally to find relief at the shafts. Probably the full-blown flame forming the explosion-wave fills the roadway sectionally, and traverses it in close contact all round its perimeter or very nearly so, its length being of course unknown, but probably varying considerably. It may also be assumed, if the roadway near the nascent explosion-wave contains coal-dust of the most inflammable nature, that the maximum strength of the initial explosion will be thereby increased, becoming fiercer, and exerting greater force owing to the ignition of the gases contained in the dust. If not met with in the earlier course of the blast, coal-dust may at a later period revive into fresh animation the enfeebled flame, which thereupon bursts into renewed vigour as great as the first or even greater, for the explosion-wave in its progress whirls the dust into eddying currents, and so completely mixes it with the air that it becomes highly combustible, and adds indefinitely to the force and destructiveness of the explosion. In consequence of this accession of force through the combustion of coal-dust and the diminishing resistance offered as the air is expelled at the shafts, the most violent effects of an explosion are sometimes found about a point beyond which all further trace of its passage is lost. An explosion may be intensified in effect by the compressed air immediately preceding the explosion-wave blowing down the road timbers, thus causing the roof to collapse and possibly liberating additional fire-damp from above to feed the flame. Whether, however, the additional force gained is supplied by coal-dust or fire-damp, the result is practically the same. The outward current is compressed, and as it is discharged at the shafts, is throttled. The violent compression so caused, is often sufficient to overthrow all obstacles, such as timbers, &c. lining the roadway, and whenever the flame is reinforced by coal-dust or gas, it bursts out into flashes of great vehemence with accelerated flame-speed and increased expansive force. The evidence of such terrific spurts having occurred is often unmistakable. In an explosion at the face, the coal seam forms a solid wall or dead end which prevents the flame from travelling inward; well-made stoppings also prevent its passage in some directions, but there are always available the inward and outward openings previously coursed by the air-current and possibly others. It has frequently been observed that the direction taken by the blast was that opposite to the intake, that is, *against* the air, but this is not invariably so; sometimes it passes *with* the current of air to the returns, which, if the pressure be sufficient, are attacked with great violence.

Generally, there is nothing to lead to the inference that the flame divides at its point of origin, but an explosion at the Abercanaid Colliery, near Merthyr Tydfil, on 23rd August, 1891, is officially reported to have shown signs of radiation from its seat. The usual outward direction taken by the explosion-wave is usually ascribed to a natural tendency to travel against the air-current, in order to secure a supply of fresh oxygen, as is found in the case of a coal-seam fire; and it is probable that it would take this course quite irrespective of the coal-dust which, as being found chiefly in the intake airways, has sometimes been considered to be the only cause.

It is believed that an explosion-wave does not invariably travel smoothly and uninterruptedly to the end of its course. Immediately behind such a wave, a partial vacuum is created by the expulsion of air, and also by the condensation of steam in the afterdamp, and this probably has a considerable influence in retarding the onward force. Indeed, some of the effects of an explosion can only be explained by supposing a partial retreat of the wave towards its point of origin. In serious explosions the forward movement is, however, quickly resumed, as in every case the partial vacuum is quickly destroyed.

Survivors from explosions have testified to a push, or suction, felt even at a distance from the explosion, and also to the reverse currents of air set in motion by it.

It is the opinion of some experts that this suction draws out more fire-damp from the working coal face, from goaves, and fissures or hidden cavities above the roadways, and that in this way the flame is fed. But it seems highly probable that an explosion strong enough to create a vacuum sufficient to produce such a suction, would also be sufficient to consume all the oxygen, not merely at the point of ignition, but within a considerable radius. If this be so, there would be no air to form an explosive mixture at the rear, except what might rush in from side openings after the flame had passed. This action might possibly take place with sufficient rapidity to augment the initial explosion, but much would depend on its position. It would be most likely to occur where there are branches off the starting point, or at the face of a longwall working, and least likely in a *cul de sac* having no openings near the face. On the whole, it is probable that any liberation of gas caused in this way, would take place after the danger of ignition had passed, unless (as might easily be the case) some burning material were left behind.

Coked coal-dust is often deposited on timbers, &c., on the side which is exposed to the full blast of an explosion; but it is also sometimes found on the opposite, the sheltered side, this being due to the recoils above mentioned. It is sometimes found on both sides as two distinct deposits, the result of both forces. The coked dust is sometimes deposited on the roof, sometimes on the sides, and sometimes on both. Wherever coked dust is found it existed before the explosion as pure coal-dust. There appears no reason to doubt that an explosion originated and propagated by fire-damp may leave coked dust, although the dust may have had little or no influence in augmenting the explosion. This is especially likely to occur at or near *culs de sac* into which tongues of fire have penetrated, and, being baffled, have had time to fuse the dust. This feature has, in fact, been observed in some explosions, notably the Llanerch, in Monmouthshire, on 6th Feb. 1890.

At times the dust, instead of taking the form of a deposit, seems to have been more highly heated and to have charred the timbers. Both coked dust and charred timbers are often but not always found after either a pure fire-damp or coal-dust explosion in portions of the roadway which are more or less dusty, but not infrequently they are absent from dusty sections of the roadway known to have been traversed by the explosion-wave. Here and there the coked dust will be found at points apparently where the coal-dust in the road was pure. If only a small quantity of coal-dust is fired in the passage of flame, the traces of coked dust will be slight, but the absence of abundant deposits of coked dust should not be taken as positive proof that coal-dust has been an unimportant factor in the explosion. Again, here and there charred timbers are found at separate and distinct parts of the mine. In many parts the roads are clean for long distances and give no indication that the blast has passed along, the bark on the timbers preserving its look of freshness and remaining uninjured and unsoiled. Besides the coked dust and charred timbers in the roadway, the sides and roof are often blackened from the smoke of the coal-dust consumed, while occasionally signs are found of the singeing of light materials such as the frayed ends of brattice cloth sheets or wooden doors, and also of scorching on the timbers and doors.

In addition to such indications of route as coked dust deposits, charred timbers, and signs of burning or scorching, any one of which may or may not be found, there may be other evidence of the course of an explosion: tubs are thrown over, the timbers knocked down, the roof-stone falls, stoppings, ventilating doors, bratticing and air-crossings are blown away. Long lengths of fallen roadways are caused by explosions and take months to restore before the pit is entirely recovered, more especially where the roof is friable and liable to heavy falls.

It rarely happens that both shafts are damaged in an explosion, but in most severe cases the downcast is more or less affected, and frequently so much so that hours elapse before it is sufficiently repaired to permit a descent. Although the upcast usually escapes, it is rarely safe to descend this shaft owing to the afterdamp which may begin to ascend immediately after the explosion. The ventilating fan, usually placed at a distance of 15 or 20 yards from the shaft, is seldom injured, and the shaft itself, if provided with a cover which yields to the blast, is soon repaired.

The destruction of the means of guiding the air-currents to the face often leaves the whole mine practically unventilated, owing to the tendency of the air to take the nearest route from the downcast to the upcast shaft. The falls have to be sufficiently cleared to allow of access by explorers and the erection of temporary doors, stoppings, &c., before any rescue party can penetrate to workings cut off from ventilation. The more speedily this work is done the greater is the hope of reaching any men who may have escaped the first effects of the explosion. Frequently, the afterdamp spreads and overtakes many such survivors, while some in their alarm, while making for the shaft, follow the course of the intake airway and so run towards the afterdamp. In many cases the return airway would have been the safer road for escape, at least for some distance, but as the returns are not ordinarily used as travelling roads to and from the working faces, they are almost unknown to the general body of workmen.

The direction in which ventilating doors are blown out of their frames, and sometimes the slant of props when slightly deflected from the upright, give evidence of the direction of the blast. In roads branching out of that taken by the explosion-wave doors at a distance, and not exposed to great outward pressure, may be drawn inward towards the blast by the suction succeeding it. In portions of the roadway which are left clean, no other evidence may be available by which to trace the course of the explosion-wave, and it becomes more than ever important to be able to draw correct inferences from such appearances.

The bodies of workmen killed by the explosion-wave show signs of burning, as might be expected, while those killed by the afterdamp do not. And yet trustworthy instances are recorded of bodies having been found which bore no signs of burning, although they were in roads along which the blast must have passed. In some cases inflammable substances such as straw have been passed over by the flame and left unscathed. Possibly there may be peculiarities and irregularities in the combustion of the flame itself, it being deflected or caused to swerve in its course by the crooks and undulations of the road or by other causes. The impact caused by a want of alignment or a want of uniformity in the rise or dip of a gallery, may be followed by reactions which shield from flame contact very slight patches of the perimeter. It would thus be possible for a body found in such a protected position to be free from burning, although deprived of life by the intense heat before the arrival of the deadly afterdamp.

Sometimes it is exceedingly difficult to tell whether the direction of the blast has been outward or inward, the evidence for both being equally balanced. This may be from rebounds of the blast, which may be caused by heavy falls of the roof immediately in front of the explosion-wave, or in the midst of it, such falls interrupting and pushing back the wave in its progress. These rebounds are also probably caused by inrushes of air into the vacuum behind the explosion-wave, and possibly also by wet sections of the road which, although not long enough to entirely overcome and choke the flame, yet oppose its passage sufficiently to produce rebounds before the explosion-wave leaps over them in its forward journey.

Owing to difficulties arising from conflicting evidence, or from the interpretation of facts, experts often disagree not only as to the cause, but as to the initial point of an explosion and also as to the direction taken by the blast.

Some explosions, instead of leaving behind them indications of the roadways having been swept by one mass of flame, present very complicated features, and are very puzzling to investigators. *Complex explosions* are those in which the results seem to point to the conclusion that fire-damp has been ignited in two or more places at the same, or almost the same instant, the explosion-wave from each travelling in different directions. This is to be carefully distinguished from a single branch explosion-wave, leaving the main body at the junction of roads and then running in a separate course contemporaneously with it afterwards. On the first explosion-wave being ignited a vacuum is formed in its rear towards which air rushes from all directions open to it; consequently fresh fire-damp is liberated from the face, the sides, roofs, and floors of the roadways and from the goaves remote from the seat of the explosion. The destruction of the vacuum may be followed immediately by a very violent inward pressure which creates a current sufficiently strong to force the flame of a safety-lamp through the gauze on to what has now become an explosive mixture and so ignite it. If instead of safety-lamps, naked lights are in use, no inward pressure would be necessary to produce one or more explosions at a distance. Again, there is the possibility of a blower being suddenly tapped by a fall far from any lights. The blower at first fouls the atmosphere in its immediate vicinity, and soon brings it up to the explosive point. As the issue continues the explosive mixture may gradually tail back through any channels open to it until it reaches lights, or shots at the moment of firing, in two or three places at the same time and so set up separate explosions. Or if the fire-damp from the blower does not penetrate more than one road in being given off, it may be ignited at its outward end by a defective safety-lamp or open light, and if the resulting explosion-wave then travel against the current impregnated with the gas, the expansion following the initial explosion may drive back the body of explosive gas upon an open light, defective safety-lamp, or flame from a shot, and so bring about a second explosion during the continuance of the first.

A blower of this magnitude would not be likely to exhaust itself before an examination of the position had been made, and would thus be detected, unless access to that part of the mine were prevented by standing fire, or heavy falls of roof. If on examination within a day or two, there is no abnormal issue of gas, this would be reasonable evidence against the theory that a blower had been tapped.

Under certain conditions of roof, and where the roof yields fire-damp freely, it may be that spaces exist, out of sight yet communicating with the air of the mine through cracks. These spaces being filled with an explosive mixture might form a train of fire, and thus ignite some accumulation of gas at a considerable distance. The condition most favourable for this, would be where the roof for a few feet immediately above the coal subsided gradually upon the gob as the face advanced, yet the next stratum above consisting of hard rock several feet in thickness only subsided at intervals of some weeks for a certain distance back from the face. Thus, a cavity for gas would be formed, which might extend for hundreds of yards following the line of faces. That this is so, is frequently shown by abnormal quantities of gas coming off when a sudden weighting of a strong upper roof occurs, while at other times no gas can be detected. For weeks the gas filling this hidden space may remain comparatively pure, but sooner or later some communications with the atmosphere of the mine will be effected, and will thus lead to the formation of a highly explosive mixture, mechanically, as well as by changes in pressure and by diffusion. A train may be thus prepared and lie ready in waiting throughout a network of crevices and fractures communicating with a distant place containing a body of gas which may have escaped detection. It is thus conceivable that if an explosion fires this train, the flame may be carried along until it reaches the distant gas when it would set up a second full explosion, which would then run its course independently but almost simultaneously with the first.

Another theory to account for these complex explosions which has found favour with some scientific authorities is that the pressure resulting from the first explosion is sufficient to cause an inflammable mixture in another portion of the mine to part with some latent heat which may of itself ignite an explosive mixture in that place. That it is possible by compression to ignite a certain percentage of fire-damp present in air may be proved on a small scale by means of Dr. Angus

Smith's compression-syringe. This instrument consists of a glass tube having a closed end, and fitted with an air-tight piston. The tube is filled with the explosive mixture, which is rapidly compressed by the piston. The temperature of the mixture within the tube then rises. If 5 per cent. or more of marsh gas be present in the atmosphere the increased temperature generally causes the combustion of the gas, and a flash is seen in the tube. If a small quantity of platinum-black is placed at the closed end of the tube, a smaller percentage of marsh gas may be sufficient to cause a flash, but the proportion must never be less than 2·5 per cent. of the gas, or no flash occurs.

Splits from the original explosion-wave may occur at branch roads and be fed by coal-dust. Different currents of these splits may meet afterwards, which would account for the indications of different directions noticeable after some explosions; it is probable, indeed, in some instances what has been taken as evidence of separate explosions has been erroneously interpreted, and that only one explosion has occurred which has divided and become re-united in one or more places.

There is generally abundant evidence of the enormous force exerted by an explosion. Besides the broken trams, torn-up rails, and other damage done to property, there is the further fact that dead bodies have not infrequently been found with heads or limbs blown off, and sometimes entirely stripped of clothing. Some of the victims appear to have been overtaken when pursuing their usual duties without thought of danger.

Many explosions have no doubt been averted by a careful retreat of the workmen under the protection of their safety-lamps on the sudden appearance amongst them of outbursts of fire-damp, which have subsequently filled extensive workings.

An instance of great force exerted is recorded where an explosion in the Udston colliery was the means of causing a rush of wind in an adjoining one separated by a barrier of coal 40 yards wide.

Next to the explosion itself and the loss of life and property which it involves, the most serious frequent consequence is an underground fire. Some smouldering material, the result of the conflagration, is fanned into a flame as soon as ventilation is partially restored, and in its turn sets fire to the coal, or other combustible material at hand. These fires may break out in one or more places. Until they are extinguished explorers can do little towards restoring the pit, as there is a constant dread of further explosions occurring from fresh accumulations of gas being kindled at the fires. Thus it often happens that the mine has to be flooded in order to extinguish the fires.

As to the part played by coal-dust in colliery explosions it is only possible, by means of experiments, to arrive at an approximation of the truth. It is, however, well established that coal-dust under certain conditions is a very dangerous element in the mine, for although a particular dust may not be explosive by itself it is capable of ignition, and when properly raised into a dust-cloud in an atmosphere containing 1 or 2 per cent. of fire-damp, the mixture becomes highly explosive. There are strong grounds for the opinion that certain kinds of coal-dust will render an atmosphere explosive even in the absence of fire damp. When an explosion-wave is started, it is probable that the intense heat thrown out and the violent agitation of the dust particles by the pressure proceeding from it may convert an otherwise harmless dust into an inflammable one. The high initial temperature causes a distillation of the dust; the dust then parts with large quantities of gases of high temperature. The fine particles of a plentiful coal-dust thoroughly and uniformly mixed with air in the gallery of a colliery may thus render the atmosphere immediately in front of the nascent flame a highly inflammatory one.

Given, then, a coal-dust of sufficient fineness, quantity and dryness and an explosion-wave initiated by a strong ignition of fire-damp, and an indefinite extension of the explosion is inevitable.

Many other kinds of dust are known to be inflammable. Flour-dust explosions have occurred at mills in different parts of the country, which have been followed by patient and careful inquiry. It was formerly the custom to crush corn by a pair of mill-stones revolving rapidly in opposite directions. On the corn-feed being stopped, without the cessation of the machinery, sparks have been produced from the friction of the stones which have ignited the fine flour-dust in the immediate neighbourhood, and passing through a channel of agitated flour-dust have produced a violent explosion in a large room at a distance.

Similarly, explosions of malt-dust have occurred in kilns used for drying and grinding malt. Ground cork-dust, too, made in linoleum works, has ignited and caused explosions. Probably many other dusts of combustible substances, if sufficiently fine and diffused in proper proportions in the atmosphere, will explode or ignite on the application of a sufficient volume of flame.

On the one hand it may be affirmed that all the largest explosions have occurred in mines which are more or less fiery; and on the other hand, that no large explosion has ever occurred in a very wet mine.

Small explosions occur in all kinds of mines, wet and dry, and two have occurred in mines in which fire-damp was not known to exist, or had not been previously found, viz., one at the Camerton Colliery and another at Timsbury.

In those cases where under apparently normal conditions a serious explosion has occurred, traversing extensive workings all in full operation,—thus precluding the idea of fire-damp having previously been present in large quantities, or long stretches of explosive currents, in accessible parts of the mine, no other theory appears tenable than that of a sudden and immense outburst of fire-damp, or of an explosion intensified and magnified a thousandfold by coal-dust. If a sudden outburst, it would seem to require an appreciable time to have elapsed between the discharge of gas and its ignition, so as to form a large explosive body pervading the whole of an extensive mine. On the other hand, if coal-dust has been chiefly instrumental in an extensive explosion, it is not necessary to assume any large emission of gas. The latter alternative, therefore, seems by far the more probable.

Explosions have in many cases been confined to dusty roads, and have stopped where the dust has ended, although the course pursued was not that in which the most fire-damp might have been expected. In a case where a double intake, made up of duplicate airways, extends from the downcast shaft, and then merges into a single one, the explosion-wave has been observed, after reaching the merging point in its outward course, to select the dusty road without penetrating far along the other, which was free from dust. Where the choice of routes exists between a dusty and a clean road, the explosion-wave generally apparently follows the dusty one, even if the clean one presents a straight and more direct passage to the shaft. Of course the choice would be the same if fire-damp were in one of two roads, but of all places in the mine the least likely for gas to be found are the main intakes. In the case of two roads, free from dust, one containing a current up to the explosive point, and the other comparatively pure air, an explosion would certainly follow the former, while it would only expand into the latter and not travel far in it.

Then again, as explosions almost invariably traverse the intakes outwardly, the natural point at which to expect an explosion to exhaust itself is at the downcast shaft, or before reaching it. Where the upcast is near the downcast, this shaft also is frequently affected, not so much by the passage of the blast through the returns, as by its passage through openings near the downcast, the separation doors in which do not form an invulnerable barrier. The noise of an explosion heard on the surface is usually followed closely by ejections of smoke and dust up one or more of the shafts. The vibrations caused on the surface are somewhat similar to the slight shocks of earthquakes in this country, causing pictures on walls to swing and crockery to be broken. Large subsidences of the roof or "weightings" have been known to produce similar shocks. The products of combustion from a pure fire-damp explosion, however deleterious, are invisible excepting that which takes the form of steam partially condensed, and the dark smoke and dust vomited from the shaft are due, like the deposits on the props and sides of the roadways, to the coal-dust in the workings. If a gallery were absolutely free from all dust, then an explosion of fire-damp and air in it would leave that gallery unsoiled, and however great the resultant rush in the shaft, no dark smoke would accompany it.

An explosion is generally, but not invariably, restricted to the seam, level, or plane on which it originates. In some cases, although the explosion proper has not reached another plane, afterdamp has extended into a different seam at another level, resulting in the death of those in the immediate vicinity of the shafts.

If an explosion-wave which reaches the shaft does not exhaust itself there, but crosses it and continues its ravages along another roadway; or if it rushes downward to pursue its course in the workings of another seam; or, if after travelling towards the shaft, it abruptly leaves that course for a road at a considerable angle, there must be some cause for these vagaries, and renewals of energy. A sufficient cause would be found in a further accession either of fire-damp, or of coal-dust, or both, and this would be far more probable than the theory, that the initial force of the explosion, not being expended at the point of divergence, took that course in preference to the shafts. As there is no part of a fiery mine more likely usually to be free from fire-damp than the main intakes at their junction with the downcast shaft, the extension of an explosion along another intake points irresistibly to the conclusion that coal-dust alone gives this fresh vitality to the blast. In support of this view there is the fact that it is near the downcast shaft that the very finest and most inflammable kind of coal-dust is likely to be found. It may be lying near the downcast shaft in two different seams, not widely separated by intervening strata. In that case even the dampness of the shaft itself may not form a barrier to the passage of flame from a higher to a lower level. But if the shaft were dry the passage of flame would of course be far easier.

It will be readily seen then that coal-dust is an element of danger in mines even slightly fiery. Its removal should therefore receive constant attention, but as it is almost impracticable to remove it completely, roadways should be systematically watered so as to damp the dust, and thus render it harmless. This matter is secondary only in importance to dealing with the fire-damp itself. Besides watering the main haulage and all other roads which otherwise would become dusty, accumulations of rubbish, whether arising from falls of the roof, or coal off the tubs, should be carefully prevented, and the sides, the refuge-holes, and the roadways generally kept clean throughout. If rubbish is allowed to be placed on the sides temporarily, owing to falls during working hours, it should be removed immediately afterward and stowed or sent out. It is unfortunate that some roadway surfaces are so affected by watering as to make it objectionable if not impracticable to keep them constantly moistened. The ill effects, however, often diminish in the course of time. Where watering injures the roadway accumulations of all kinds must be prevented by systematic removal. The labour involved will be greatly reduced if dust-proof tubs are used in the mine. The best of these have steel or iron bodies. If wood is used the boards should be tongued and grooved, to prevent leakage as much as possible. Covers of tarpaulin or other material are of little use. A better plan is to water the trains before they start from the inner ends of the engine-planes. As a rule such precautions against accumulations of dust are more economical in the long run than any attempt to deal with it afterward. Attention, too, should be paid to the state of the rails. New ones should be laid down before the old ones are frayed out by wear, and no inequalities of level should mark the joints. When well-laid and properly maintained, the jarring of the tubs will be reduced to a minimum.

Experiments made on the surface several years ago by Mr. W. Galloway and Sir Frederick Abel, established the fact that 2 per cent. of fire-damp mixed with an atmosphere properly impregnated with coal-dust renders the mixture explosive. It could then be ignited by an ordinary naked light. Here again is the danger of coal-dust emphasised, for an open light would not cause an explosion in a pure atmosphere mixed with 2 per cent. of fire-damp. Before anything like an explosion can occur in an ordinary atmosphere, it has usually been considered that there must be present about 7 per cent. of fire-damp. The Royal Commissioners on Accidents in Mines, 1886, state that when a little more than 4 per cent. of marsh gas is present, flame may be transmitted under circumstances which occur in a mine. A shot-firer or fireman examining for gas may fail to detect 2 per cent. of fire-damp if he uses an ordinary safety-lamp, for the cap or aureole then shown is not easily distinguished.

In different makes of lamp, as the quantity of fire-damp is increased beyond 3 per cent., the cap or aureole is enlarged in various forms until with a sufficient amount of gas to render the atmosphere inflammable, it burns as it enters the lamp; the usual air inlet-channels being thus filled with the burning gas to the exclusion of an air supply, the lamp-flame is extinguished. With an ordinary Davy-lamp there is free entry all round through the gauze, and if an inflammable mixture passes through, the whole of the gauze cylinder becomes filled with the blue flame produced by the burning gas. In a still atmosphere, or in a gentle current, the Mueseler and many other types of lamps, while held in an erect position, do not allow the flame from the burning gas to come into contact with the gauze at all. Tried under these favourable circumstances, these lamps are some safeguard against defective gauzes, but no protection of this kind can be relied on where eddying, oblique, or down currents play about a lamp, or where it is tilted or oscillated. Knowing the danger of flame from firing shots, and the consequent clouds of dust raised in what may be, with the usual fireman's means of testing, a slight but undistinguishable mixture of firedamp and air, and also the greater danger of blown-out shots with their long projections of flame and sudden propulsions of wind into gusts, the Mines Act of 1887 expressly forbids any firing of gunpowder shots in dry and dusty mines, unless the dust is first laid by watering all places within a radius of 20 yards of the shot-hole, although no gas may be detected there. In a fiery mine there is a constant emission of fire-damp at the working face, and even if free from accumulations of gas in the goaves and elsewhere, the current passing is most likely to be charged to the extent of from ½ to 1 per cent. and even more, the percentage increasing as the working approaches the return airways, where it frequently is just beginning to show a cap. Where no cap is detected, the current would be deemed safe.

Thus, with the air in this condition, any violent agitation of dust, such as would result probably from a heavy shot, and especially from a blown-out shot, the atmosphere containing the otherwise harmless percentage of gas becomes an explosive one. Thus, any extra discharge of fire-damp from the face or checking of the air-current through falls, may suddenly produce a highly explosive mixture because of the raising of the dust. In a longwall face, happily, a thick layer of coal-dust is never found, owing to the rapid advance of the working. It is improbable, therefore, that it can be raised in a longwall face sufficiently to prolong an explosion without the aid of fire-damp. By far the larger number of shots fired in some collieries are at the coal face; if the coal being worked is hard and clings fast to the roof-stone, shot-firing is resorted to. Most of the shot-holes are, however, placed near the roof after the colliers have holed under the coal, so there is little risk of a blown-out shot, and the danger of raising the dust is not nearly so great as where shots are placed in the floor, as these when blown out, raise the whole of the dust that may be present. If gunpowder were used, the blown-out shot would send forth a stream of flame into the raised dust. Hence the necessity for watering, but if this be not resorted to, other explosives must be used. Some seams of coal have no suitable material for holing in except over or in the seam, and if these have to be blasted, the shot-holes are placed near the floor, but by far the larger number of seams are holed under. Besides the shots fired in the coal very near the face, the roof or floor must probably be blasted to make sufficient height for the roadway. The shot-holes are here bored in the shale or rock, and the dust resulting will be a coarse and impure mixture. The whole of the dust in the longwall faces and closely adjoining roadways is always comparatively new, the old dust being buried and left behind, and this newly-made dust is probably seldom present in such quantities and fineness as to be really dangerous.

Dust is liable to accumulate at and near the face of pillar and stall workings to a greater extent than in longwall. In thin seams the rails are frequently only laid to a point several yards back from the face, and the coal is then rolled along towards the tub, and is much broken and trampled in transit. The side cutting of the coal, which is often necessary as well as the holing, especially in openings in the solid, results in much small being made. Where there is no suitable underclay, the holing is made in the coal itself. This wastes a few inches of the lower portion of the seam which is cut out as small coal by the miner's pick. In working out the pillars the coal is usually much crushed, the whole weight of the strata to the surface being especially severe on small pillars in deep mines.

It is on the main haulage roads, however, where the most dangerous accumulations of the coal-dust occur, and where, from its fineness, it is sometimes called " coal flour." It reaches from a point a little way out from the working face right to the downcast shaft. On the engine-planes the trains are often run at great speed, and dry coal carried in them is subjected to shaking, jolting, and jarring. The dust is also blown off the trams travelling against the intake air currents, and gradually settles on some part of the roadway, the finest particles remaining longer in suspension, settling for the most part on the timbers, roof, and sides. The vibration, when in transit, riddles the dust through joints and holes in the trams or tubs. When the screens on the surface are near the downcast shaft (and this is generally the case), much coal-dust of extreme fineness is carried with the current down the shaft, and most of this fine dust reaches the main intakes, where it adheres to the roof and sides, hangs like a fringe on the under side of collars, and covers the floor to a depth of several inches if allowed to remain undisturbed.

From the fact that very widespread explosions are not of frequent occurrence, and that dust-strewn roadways have escaped for years, their condition is generally regarded as safe so long as shots are not fired in them, and no explosion takes place in some other part of the mine. The ordinary traffic of the mine which allows of the accumulation is obviously not sufficient to raise the dust into a cloud so as to be exploded by open lights such as are frequently carried by the workmen and others between the downcast shaft and the lamp-stations, engine-galleries, and stables. If, however, an explosion occurs, the dust is much more violently agitated, and then, probably, the atmosphere becomes so impregnated with the dust as to render it fatal to anyone inhaling it.

Although the main haulage roads are those where the largest quantity of dust is likely to be found, there are instances where, probably, an equal amount is deposited in the return air-ways. These are cases where the seams of coal yield an exceptional amount of fine dust in the actual process of working. From the faces it is carried by the air-current into the return air-ways, and settles there. The more active the ventilation, the more coal-dust will find its way into the returns.

The comparatively high temperature which prevails in most underground workings, doubtless, assists the fine division of dust-particles. Dust remaining a long time subjected to this heat becomes drier, and it may be that it absorbs oxygen or undergoes some chemical change which makes it more dangerous. On the other hand, long exposure to the atmosphere probably causes it to part with some of its volatile gases. It is known that the capacity for absorbing oxygen is greater in freshly worked coal than in that which has lain some time, but it appears that the newly-made coal-dust is not so inflammable as old dust. The chemical changes which take place would therefore appear to increase the dangerous character of coal-dust, but the precise nature of these changes is still unknown and requires investigation.

Since the date of Mr. Galloway's and Sir Frederick Abel's experiments already referred to, many other investigations have been made with a view to satisfy the doubts prevailing, and to ascertain the precise circumstances, if any, under which coal-dust and air may be ignited. It is obvious that such experiments must be very costly. They cannot be made in a mine for fear of serious damage. They have, therefore, for the most part, been carried on in specially constructed boxes on the surface. Results, both of a negative and positive kind, have been obtained. In nearly all the experiments, the firing of a cannon has been taken to represent the concussion and flame from a shot, and has been discharged into a gallery containing (*a*) coal-dust and air alone; and (*b*) coal-dust, air and fire-damp. Although in the case of (*a*) it was said that ignition resulted in some of these experiments, the conditions were such as would not be found in a mine. The Camerton explosion and Mr. Hall's experiments conclusively prove that coal-dust and air may be ignited.

Slight ignitions of coal-dust with air only are stated to have occurred about the colliery screens, from which clouds of coal-dust have been raised and carried by the air to some gas-light or other large flame. Some of these ignitions have merely been a momentary " flare-up " of the flame of a gas-jet, perhaps owing to a paucity of coal-dust particles. A more serious ignition of coal-dust only, resulting in loss of life, is reported to have occurred in a hopper on the surface of Brancepeth. In this case a large quantity of dust was thrown from a height, raising a dense cloud, which ignited at an oil-lamp when the hopper was being cleaned out—a case which could hardly occur in the limited height of underground roadways. These ignitions showed no signs of great explosive force, but this may have been due to their occurring where there was ample space for free expansion from the heat.

The Altofts colliery explosion was officially reported to be due entirely to the firing of coal-dust. It has indeed been suggested that a very slight percentage of fire-damp may have been present to aid the initiation of that explosion. It is urged, in support of this theory, that probably no serious colliery explosion has ever occurred, except in a fiery mine. Wet mines are sometimes fiery, and explosions have occurred in them. These would be due to fire-damp alone. Some dusty mines are free from fire-damp, and have certainly had no coal-dust explosion during the last 50 years and probably never at all. Seams of this sort are being worked in the Radstock portion of the Somerset coalfield, and in the Forest of Dean. Explosions of coal-dust have now occurred in two of these mines, which are free from fire-damp, and leave no room for question as to the inflammability of coal-dust. Still there are doubts in the minds of some as to whether fire-damp, although undetected, did not play its part in the Altofts explosion although ascribed to coal-dust alone.

The most conclusive evidence of a coal-dust explosion of magnitude in a mine was that given in the case of Altofts, in Yorkshire, which occurred on Oct. 2nd, 1886, and in which 22 men and boys were killed. It is the only explosion which has been attributed by a jury wholly to coal-dust.

The Silkstone seam in which the explosion happened was worked by longwall, and the colliery had been working for 23 years at the time of the occurrence without any previous explosion. During the first 20 years open lights were used in the mine, although a little fire-damp was given off, but for three years previous to the explosion, safety-lamps were used in the workings. Safety-lamps had been gradually introduced during the preceding year or two, and were used throughout the workings at the time of the explosion. Naked lights were, however, used on the main haulage road. This road, on which an endless chain worked, was about 4½ feet out of alignment at a point 500 yards from the shaft, and 970 yards from the nearest coal-face. In order to straighten this road gunpowder shots were fired—the first that had been fired at this colliery on any main haulage road—and one of these shots is supposed to have ignited the coal-dust, and caused the explosion. The roof from the shaft pillar inwards was bad, and upheld by timbers. From 40,000 to 50,000 cubic feet of air per minute passed along this road, which was a main intake. There were three shots fired in the haulage road. From the positions in which the bodies of the road-repairers, who were also the shot-lighters, were found, it was believed that these shots were not fired simultaneously. The men appeared as if shielding themselves in refuge-holes from No. 3 shot, which was 20 yards to the westward or inside of Nos. 1 and 2. Some of the men were found opposite Nos. 1 and 2 shots, and the conclusion arrived at therefore, was that these two were fired harmlessly together or singly and No. 3 a little time afterwards. It was thought unlikely that any fire-damp would be there, but even if any were lurking in roof-crannies it would probably have been dislodged by the concussion of the first shots, and carried in with the air-current before the firing of the last shot. No. 3 was assumed to be an overcharged gunpowder shot. In an overcharged shot the excess of explosive inserted in the shot-hole beyond that actually required to do the work is spent in flame streaming into the roadway after the manner, but not to so great an extent, as a blown-out shot. The supposition is that the concussion of the shots raised the dust into a cloud, and the stream of flame from the last shot ignited it. The explosion-wave took a course against the intake current, and on reaching the end of the west chain road, which was very dusty, instead of exhausting itself at the shaft it turned almost at right-angles along No. 1 chain road, which also was very dusty, and died out where the pure coal-dust ceased, and was covered by ripping-stone dust. Some fires were subsequently left in the pit owing to the ignition of the gas issuing from fissures in the floor from a lower seam. Since the explosion safety-lamps only are used on the roads, as well as in the working-places; shot-firing has been abolished, and given place to drilling machines and the wedge.

From these facts it will be seen how difficult it is to dogmatize as to a coal-dust explosion. Notwithstanding that the mine was not very fiery and that the explosion occurred on a main intake, along which so large a volume of air swept, objectors will urge the possibility that a slight per centage of fire-damp aided the explosion. Indeed, many think it extremely unlikely that an explosion can originate on any main intake at all worthy of the name. It should be borne in mind that however long the roadways may have been in existence, there is always the possibility of gas finding its way into them, either from the roof, floor, or sides. In the case of a main intake through solid rock across the measures, and free from fissures, open pores, and all possible hiding places, there would be no hesitancy in saying an explosion could not take place there from fire-damp. Where, however, the roadway is through a longwall working, with an admittedly bad roof, and subject to slight issues of fire-damp from an underlying seam, it is quite possible for gas to reach the roadway and cause an explosion. Notwithstanding the fact that in longwall workings most of the gas is drained off as the face advances, leaving the roadways beyond a certain variable distance comparatively free from the influx of gas, and that there may be a sweeping ventilation, it is still possible, and even probable, that gas will lurk in cracks and cavities above timbers, being fed from the surrounding strata. Concussions of shots may help the liberation of fire-damp by opening channels amidst previously separated networks of holes. The flame from a later overcharged shot may belch forth into this, and although the amount of fire-damp is small, it helps to initiate an explosion which afterwards, it will be admitted, can be intensified by coal-dust alone. It is also conceivable that the inner end of a shot-hole may be in communication with explosive gas, which could not be discovered by the most careful examination with a safety-lamp, and that the flame from the shot may fire the explosive mixture at the back end of the shot-hole.

Although the returns at the Altofts Colliery were tested with a Liveing fire-damp detector after the explosion, no trace of fire-damp could be found, so that it could not have been present except in a quantity under ¼ per cent. This perhaps proves that generally the ventilation was efficient, and the total quantity of fire-damp given off was small, but it does not remove the possibility of a small quantity having been in the roof at the time that the final shot was fired.

With regard to explosions which occur in wet mines there can be no question that these are due entirely to fire-damp. In these there may be evidence of the exertion of great force, but there are not the peculiarities of route, such as are noticeable in dusty mines, nor do the ravages committed extend so far through the ramifications of the mine. Fire-damp must always claim the careful manager's first and foremost attention, but not to the exclusion of all else.

We have referred to the dusty mines of Radstock and the Forest of Dean, in which, with the exceptions of Camerton and Timsbury, no fatal explosion has occurred for many years, and in which no fire-damp is given off. It is possible that the precise conditions under which coal-dust can be ignited, are often absent. It may be impossible to ignite some coal-dusts under any conditions. Perhaps it is only occasionally that the precise conditions are present in any mine. In the Radstock district the engine-planes and wagon-roads, owing to faults, irregularities, and undulations in the seams, are made almost as much in the strata above or below the coal-seam, as in the seam itself. Any coal-dust gathering from the transit of coal in tubs over such roads would be rendered impure by the grinding under foot on the stone floor. The proportion of coal sent over the roads to tubs of rubbish, is about two to one. The rubbish is mostly that obtained in working seams too thin to admit of its being gobbed. The tubs of rubbish in their outward trips are subjected to the same jerking and bumping as the coal tubs, and result in a similar sifting through joints or badly fitting parts and dropping of larger material. The subsequent pulverising by the ordinary traffic possibly leaves a very impure dust. Too much stress must not, therefore, be laid on the fact that although gunpowder shots are fired here, only two noteworthy explosions of the coal-dust have happened.

All who are familiar with mining operations, whatever their views as to coal-dust, admit the danger attending the firing of shots in fiery mines. This danger is more accentuated in the minds of those who believe that coal-dust alone is under some circumstances inflammable. There is the fact standing out prominently that, since about 1870 especially, a large proportion of explosions have been attributed to the practice of shot-firing. In many instances, the evidence showed conclusively, from the position of the bodies of men engaged in blasting operations, that they were at the time of death shielding themselves from the effects of a shot, or at points which they would take in order to warn others from approaching the place of firing. Where shots have been exploded by electricity, there is the further evidence of the battery in possession of the charge-man. There is usually the evidence of the shot itself, either by the work it has done or the discovery of a newly-made shot-hole, which has evidently been fired, but has resulted in a blown-out shot. Then there is often no other possible theory if the evidence of shot-firing is not accepted. There are no indications of unlocked or imperfect safety-lamps, of open lights, or of matches about the workmen throughout the mine. There is further the blast itself which has clearly emanated from the position of the shot, leaving the evidence as to the cause in many instances incontrovertible.

The working face or its immediate neighbourhood is naturally the most probable part from which to expect an explosion, both because it is there that most fire-damp is found, and there the shot-firing generally takes place. The fire-damp given off at the face either normally or from blowers and outbursts, consists mainly of marsh gas—carbonic acid and nitrogen being present in the newly-discharged gas only in small proportions; whereas accumulations of gases in goaves have varying proportions of these gases and air, and will be influenced by the chemical change proceeding in the gobbed or fallen material of the goaves. There may be other parts of the mine in which shot-firing is occasionally necessary, such as stone or cross-measure drifts, or on the intakes which require widening or enlarging, but no manager would willingly allow shot-firing in the returns of a fiery mine in which the air-current is almost sure to be tainted with fire-damp. In all cases, competent shot-firers should be appointed, and the requirements of the Mines Act, 1887, carried out. If, in spite of the prescribed precautions, an explosion takes place, it must be owing to some unsuspected inflammable mixture or to a sudden inrush of fire-damp. Coal-dust was not mentioned as a cause of danger in any Act previous to 1887, in which year it was made obligatory to water the dust in the neighbourhood of a gunpowder shot. It is known that the flame from a blown-out shot of a 1½ or 2 lb. charge of gunpowder, tamped with clay, may extend as far as 20 feet from the hole, and possibly a good deal further if with a strong air current, or in a narrow passage. If the air is even slightly dust-laden, or if coal-dust tamping is used (which however is illegal), the flame may be greatly extended. There is always the possibility of fire-damp reaching a place exactly at the moment of shot-firing from a blower, outburst, or fall of the roof close by. Should more than one shot be fired at about the same place and time, the first may release gas which would explode at the next. To avoid this risk, the series should be fired in the reverse direction to the air current.

It was owing to the danger of firing shots on dusty main-intakes, and the verdict after the Altofts Colliery explosion (which occurred when the Royal Commission was sitting in 1886), that the law was passed requiring the watering of dust, with the alternative of using a flameless explosive or one protected with water. It is to be regretted that the Legislature, while imposing these restrictions, did not also maintain the clause which had been introduced in the Mines Act of 1872, whereby shot-firing was only allowed when the workmen ordinarily employed were out of the mine, or of that part in which the shot was to be fired. Whilst not reducing the risk of explosion from shot-firing, this would materially reduce the risk of life. Careful managers, aware of these risks in fiery mines, and yet unable wholly to dispense with blasting, still continue to enforce the above-mentioned regulation of 1872, all shots being fired after the day-shift men have ascended the shaft, the shot-lighters themselves being the only men in the pit. Others have abolished shot-firing entirely and substituted wedges for coal-getting, and in some instances for ripping roof, although at a considerably enhanced cost.

Refractory coal-seams and very hard rippings, however, require blasting; to dispense with explosives, would mean an additional cost which the colliery could not bear.

In order, as far as possible, to prevent explosions, and limit their fatal effects when they occur, it is absolutely necessary to—

(1) render harmless all issues of fire-damp, by constant and sufficient ventilation;
(2) prevent, as far as practicable, coal-dust from being blown off the trams or carried down from the surface screens by the downcast current, and thus reduce the deposition of coal-dust on roadways;
(3) thoroughly damp all coal-dust which is unavoidably deposited in roadways;
(4) exclude all naked lights;
(5) prohibit all explosives and fuzes for blasting which are liable to produce flame outside the shot-holes;
(6) fire shots only during the intervals between shifts, and fire them electrically;
(7) employ the most competent men at all work involving risk and responsibility;
(8) enforce rigid discipline and strict compliance with all rules and regulations.

CHAPTER XVIII.

MISCELLANEOUS QUESTIONS AND ANSWERS.

Denudation—Meeting of Cages in a Shaft with Flat Winding Ropes—Outstroke and Wayleave—Pressure of Water against a Barrier—Applying the Result Obtained from Boring—Discharge of Water from a Bore-hole—Pressure per Square Inch Produced by an Air-Compressing Engine—Description of Anthracite, Semi-Bituminous, Bituminous, Cannel, and Lignite Coals—Dynamometer—Elements of Rocks—Relative Hardness of "Rock Masses"—Beds and Veins of Minerals—Difference of Vein, True Fissure Vein, and Lode.

Question 152.—What is meant by denudation?

By denudation is meant the wearing and carrying away of the solid materials of the land by the agency of water. Rivers so carry away portions of the land through which they flow; the tidal currents of the ocean lay bare the rocky materials of its shores by wearing away their superficial deposits and by sweeping away the solid masses. The effect of the tidal current of the German Ocean is to sweep away large masses along the eastern coast of England and to make a gradual inroad in the land annually. The soil and other matters transported by the action of rivers from inland situations are brought to the sea and borne by the ocean currents to a much greater distance from their original site. It has been computed that the Hoang-Ho, one of the largest and most rapid rivers in China, brought down in a single hour two million feet of earth. The 1890 Ordnance survey in Yorkshire proved an erosion of the coast there. A portion of the coast line between Great Colden and Dimlington, has been almost uniformly encroached upon, since 1852, by the sea for about 215 feet, or at the rate of 5 feet 10 inches per annum.

Question 153.—At what point in the shaft would the cages "meet" if flat ropes, $\frac{3}{4}$ of an inch thick, are used for winding; the drums are 10 feet in diameter at the lift, and the depth of the pit 612 feet?

In winding with a flat-rope drum each coil of rope is wrapped upon the preceding coil every revolution that is made, and as the cage is drawn up from the pit bottom the diameter of the drum constantly increases and attains its maximum diameter when the cage reaches the surface. Again, as the cage is being lowered, the effective diameter of the drum constantly decreases and reaches its minimum when the cage has arrived at the shaft bottom. The increase in the diameter of the drum, the first revolution will be equal to the rope's thickness (because the rope in bending on the drum is lengthened on the convex side and shortened on the concave side equally, the centre of the rope retaining its former length), and for each successive revolution after the first the increase of the drum's effective diameter will amount to twice the thickness of the rope.

Let d = depth of pit in inches.
„ t = thickness of rope in inches.
„ x = diameter of drum in inches.
„ n = number of revolutions.

Then the diameter of the drum at the first and succeeding revolutions will be—

$$\begin{aligned} x + t &= \text{1st diameter.} \\ x + 3\,t &= \text{2nd do.} \\ x + 5\,t &= \text{3rd do.} \end{aligned}$$

and so on, and at the nth revolution the diameter will be represented by $x + t\,(2\,n - 1)$. The circumference will therefore be—

$$\begin{aligned} \pi(x + t) &= \text{1st revolution.} \\ \pi(x + 3t) &= \text{2nd } ,, \\ \pi(x + 5t) &= \text{3rd } ,, \\ \text{And } \pi\left\{x + t\,(2n - 1)\right\} &= n\text{th } ,, \end{aligned}$$

Since this series of terms are in arithmetical progression, their sum may be obtained in the usual way, and it will be, $n\pi(x + nt)$ which is equal to the length of rope wound on the drum, and therefore the depth of the pit. Therefore, $d = n\pi\,(x + nt)$. Or by denoting the depth of the pit in yards by D, then

$$D = n\,(x + nt) \times {\cdot}087266,$$

from which the following formulæ are deduced:—

$$D = n\,(x + nt) \times {\cdot}0872 \qquad (1)$$

$$x = \frac{11{\cdot}45915\,D}{n} - nt \qquad (2)$$

$$t = \frac{11{\cdot}45915\,D}{n^2} - \frac{x}{n} \qquad (3)$$

$$n = \frac{\sqrt{11{\cdot}45915\,Dt + \left(\frac{x}{2}\right)^2} - \frac{x}{2}}{t} \qquad (4)$$

$$\text{Thus } D = (nx + n^2t)\,{\cdot}0872$$

$$D = {\cdot}0872\,nx + {\cdot}0872\,n^2t$$

$${\cdot}0872\,nx = D - {\cdot}0872\,n^2t$$

$$x = \frac{D}{{\cdot}0872\,n} - nt$$

$$x = \frac{11{\cdot}45915\,D}{n} - nt \qquad (2)$$

$$nt = \frac{11{\cdot}45915\,D}{n} - x$$

$$t = \frac{11{\cdot}45915\,D}{n^2} - \frac{x}{n} \qquad (3)$$

$$n = \frac{11{\cdot}45915\,D}{nt} - \frac{x}{t}$$

$$n^2 = \frac{11{\cdot}45915\,D}{t} - \frac{nx}{t}$$

$$n^2 + \frac{x}{t}\,n = \frac{11{\cdot}45915\,D}{t}$$

$$n^2 + \frac{x}{t}\,n + \left(\frac{x}{2t}\right)^2 = \frac{11{\cdot}45915\,D}{t} + \left(\frac{x}{2t}\right)$$

$$n + \frac{x}{2t} = \sqrt{\frac{11{\cdot}45915\,D}{t} + \left(\frac{x}{2t}\right)^2}$$

$$n = \sqrt{\frac{11{\cdot}45915\,D}{t} + \left(\frac{x}{2t}\right)} - \frac{x}{2t}$$

$$n = \frac{\sqrt{11{\cdot}45915\,Dt + \left(\frac{x}{2}\right)^2} - \frac{x}{2}}{t} \qquad (4)$$

Now to consider the position of the meeting of cages.

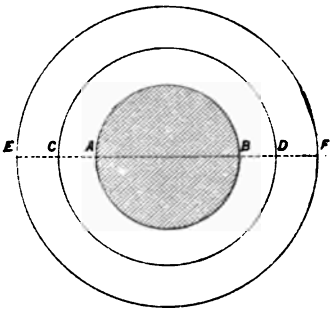

Fig. 679.—Flat Rope Drum, showing Changes in its Diameter.

When the engine begins to wind, the drums, being fixed on the same shaft of the engine, move simultaneously, thereby causing the one cage to be raised as the other is lowered. But the descending cage will pass through a greater space than the ascending cage in the same time; the excess in space travelled by the former over the latter gradually diminishing until the engine has made half the number of the total revolutions for one wind.

The diameters of the two drums will then be equal and the cages will be alongside each other or at meetings in the pit. The position of meetings then must be at a point below the middle of the shaft, and the distance of this point below mid-shaft will depend on the thickness of the rope and the number of revolutions made. It follows that after passing each other in the shaft, the ascending cage will move through a greater space than the descending one in the same time, as the diameter of the drum to which the ascending cage is attached becomes larger and the drum of the descending cage becomes smaller in diameter than the mean diameter at meetings.

In Fig. 679, the diameter of the drum is represented by A B; C D representing its diameter at meetings, and E F its diameter when the cage has arrived at the top of the pit. It will be seen that the area of the annular spaces C A B D, and E C D F are equivalent respectively to the edge area of the rope wound on the drum between the bottom of pit and meetings, and that wound off the drum between the top of pit and meetings, and the difference between these areas will obviously give the area of that portion of the rope corresponding to the distance travelled by the descending over the ascending cage, and half this distance will be the point of meeting below mid-shaft. Adopting the same notation as before for the different parts, we have $EC = CA = BD = DF = \frac{nt}{2}$. $EF = x + 2nt$ and $CD = x + nt$.

$$\left.\begin{array}{r}\text{Therefore the area}\\ \text{of E C D F}\end{array}\right\} = {\cdot}7854\,\left\{(x + 2nt)^2 - (x + nt)^2\right\}$$

$$= {\cdot}7854\,(2\,xnt + 3\,n^2\,t^2) \qquad (a)$$

$$\text{and the area of C A B D} = {\cdot}7854\,\left\{(x + nt)^2 - x^2\right.$$

$$= {\cdot}7854\,(2\,xnt + n^2\,t^2) \qquad (b)$$

Subtracting from *a* we get $\cdot 7854\,(2\,n^2\,t^2)$ which is the difference of areas in square inches, and dividing by t we have $\cdot 7854\,(2\,n^2 t)$, the difference of the lengths of rope. Hence, dividing by 2 we get $\cdot 7854\, n^2 t$, the distance of meetings below mid-shaft in inches. If this distance be required in yards (say m) we have

$$m = \frac{\cdot 7854\, n^2 t}{36} = \cdot 021816\, n^2 t$$

Therefore, representing the position of meetings from the top and bottom of the pit by p, and $p_{,}$, respectively we have

$$p = \frac{D}{2} + \cdot 021816\ n^2 t. \qquad (5)$$

$$p_{,} = \frac{D}{2} - \cdot 021816\, n^2 t. \qquad (6)$$

Proceeding now to the question given, first find the number of revolutions of the engine, and this will be done by the aid of formula (4), thus

$$n = \frac{\sqrt{11\cdot 45915 \times 204 \times \cdot 75 + (\frac{120}{2})^2} - \frac{120}{2}}{\cdot 75} = 17\cdot 55336.$$

Now apply the data to formula (5), thus $p = \frac{204}{2} + (\cdot 021816 \times 17\cdot 55336^2 \times \cdot 75) = 107\cdot 0416$ yards or $107\cdot 0416 \times 3 = 321\cdot 1248$ feet as the distance of point of meetings from the surface $\therefore\ 612 - 321\cdot 1248 = 290\cdot 8752$ feet as the distance of point of meetings from the bottom of the shaft, or the latter may, if preferred, be found from formula (6).

> *Question* 154.—A winding-engine draws coals from a pit 150 fathoms deep, drum 14 feet in diameter at the lift, flat ropes $\frac{3}{4}$-inch in thickness, find the meeting of cages.

In replying to this question, adopt a different method of working from the last, so that those familiar with algebraical expressions and who do not care to carry a formula in their memories may proceed in this way. The reasoning is similar to that of the previous method shown and the result obtained the same as would be got by that method. It is also applicable to solving the "meeting" place when conical drums are used with round ropes.

Let x = the number of revolutions of the drum in running the cages in the pit.

And y = diameter of drum in feet at the last revolution.

Then $y = \cdot 125\, x + 14$, because twice the thickness of the rope in feet is $\cdot 125$.

And $x = 8y - 112$.

$\frac{y + 14}{2} \times 3\cdot 141593$ = mean circumference of drum $= 1\cdot 570796y + 21\cdot 99115$ which divided into the depth of the pit (in feet) gives the number of revolutions of the drum.

Therefore $x = \dfrac{900}{1\cdot 570796y + 21\cdot 99115}$

substituting for x we have

$$8y - 112 = \frac{900}{1\cdot 570796y + 21\cdot 99115}$$

$$12\cdot 566368y^2 - 2{,}463 = 900$$

$$12\cdot 566368y^2 = 3{,}363$$

$$y^2 = 267\cdot 62$$

$$y = 16\cdot 359$$

$$x = 8(16\cdot 359) - 112 = 18\cdot 872$$

Number of revolutions of drum = 18·872.
Diameter of drum in feet at last revolution = 16·359.

$$\frac{16{\cdot}359 + 14}{2} = 15{\cdot}1795, \text{ mean diameter of drum.}$$

$$15{\cdot}1795 \times 3{\cdot}14159 \times 18{\cdot}872 = 900.$$

$$\begin{aligned}\left.\begin{array}{r}\text{Depth from surface to}\\ \text{meeting of cages}\end{array}\right\} &= \frac{16{\cdot}359 + 15{\cdot}1795}{2} \times 3{\cdot}14159 \times \frac{18{\cdot}872}{2}.\\ &= 467{\cdot}464 \text{ feet.}\\ &= 77 \text{ fathoms, } 5 \text{ feet, } 5{\cdot}568 \text{ inches.}\\ \text{say } & 77 \text{ fathoms, } 5 \text{ feet, } 6 \text{ inches.}\\ \left.\begin{array}{r}\text{Distance of meetings}\\ \text{from the bottom of pit}\end{array}\right\} &= \frac{15{\cdot}1795 + 14}{2} \times 3{\cdot}14159 \times \frac{18{\cdot}872}{2}\\ &= 432{\cdot}5 \text{ feet.}\\ &= 72 \text{ fathoms, } 0 \text{ feet, } 6 \text{ inches.}\end{aligned}$$

And the accuracy of the result is shown by adding together

	fathoms.	ft.	inches.
	77	5	6
And	72	0	6
Depth of pit	150	0	0

If the formula already given be applied to the working of this question, precisely the same result will be obtained.

> *Question* 155.—A shaft 34 fathoms deep is worked by a drum 3 feet in diameter. It is required to find the diameter of another drum which must be keyed on the same axis to wind from another shaft 21 fathoms deep, flat ropes, 1 inch thick, being used to coil one lap upon the other?

In considering this question it is obvious that there must first be ascertained the number of revolutions of the 3-foot diameter drum in drawing the cage up the 34-fathom shaft, and then as the same number of revolutions must draw the cage up the 21-fathom shaft, proceed to find the diameter of drum for the shallower shaft, knowing the depth, number of revolutions, and thickness of rope.

Adopting formula (4) given in answer 153, and using the same values for the symbols there given, the number of revolutions in the 34-fathom shaft n =

$$\frac{\sqrt{11{\cdot}45915 \times 68 \times 1 + \left(\frac{36}{2}\right)^2} - \frac{36}{2}}{1} = 15{\cdot}21 \text{ revolutions.}$$

Now use formula (2) given in Answer 153, to find the diameter of drum for the 21-fathom shaft, thus

$$x = \frac{11{\cdot}45915 \times 42}{15{\cdot}21} - 15{\cdot}21 \times 1 = 16{\cdot}432 \text{ inches}$$

as the diameter of drum required for the 21-fathom shaft.

> *Question* 156.—A shaft 100 fathoms deep is worked by a drum 12 feet in diameter. What must the diameter of another drum be which is keyed to the same axis to wind from another shaft 120 fathoms deep, flat ropes being used, ¾ of an inch thick, to coil one lap upon the other?

This may be worked in a similar manner to the previous question, but proceed to work it algebraically in order to show another method of obtaining the same result. Dealing with the shaft whose depth and size of drum are known, the revolutions may be found thus:—

Let $x =$ the number of revolutions of the drum whilst the cage travels from the bottom of pit to the top,

and $y =$ diameter of drum in feet at the last revolution.

Then $y = {\cdot}125x + 12$, because twice the thickness of rope is ·125 foot, and $\therefore\ x = 8y - 96$.

$\frac{y + 12}{2} \times 3{\cdot}141593 =$ mean circumference of drum $= 1{\cdot}570796y + 18{\cdot}84956$, which, divided into the depth of the pit (in feet), gives the number of revolutions of the drum.

$$\text{Therefore } x = \frac{600}{1{\cdot}570796y + 18{\cdot}84956}$$

substituting for x we have

$$\begin{aligned} 8y - 96 &= \frac{600}{1{\cdot}570796y + 18{\cdot}84956} \\ 12{\cdot}566368y^2 - 1{,}809{\cdot}55776 &= 600 \\ 12{\cdot}566368y^2 &= 2{,}409{\cdot}55776 \\ y^2 &= 191{\cdot}75 \\ y &= 13{\cdot}85 \\ x &= 8\ (13{\cdot}85) - 96 = 14{\cdot}8, \end{aligned}$$

which is the number of revolutions.

The mean diameter of the drum for the 120-fathom shaft will be found by dividing the depth (in feet) by the number of revolutions multiplied by 3·14159, thus $\frac{120 \times 6}{14{\cdot}8 \times 3{\cdot}14159} = 15{\cdot}485$.

To find the diameter at lift.

Let $x =$ the diameter of the drum at lift.

Then half the number of revolutions will increase the drum to its mean diameter $\therefore\ x + ({\cdot}125 \times 7{\cdot}4) = 15{\cdot}485$, and $x = 14{\cdot}56$, which is the diameter the drum must be for the 120-fathom shaft in feet.

The accuracy of this working may be checked by the formula already given, when precisely the same result will be arrived at.

Question 157.—Suppose an engine-house to be 15 yards from a pit, the ground (from the engine-house to the pit) rising 2 inches in 6; the height of drum from the surface is 3 feet, and the perpendicular height of framing to top of pulley wheel 25 feet: what would be the gradient from the drum to the pulley wheel?

To assist the answer refer to Fig. 680.

As the ground rises 2 inches in 6 or $\frac{6}{2} = 3$, 1 in 3, in 45 feet it rises $\frac{45}{3} = 15$ feet, and as the top of the drum is 3 feet above the level of the ground $15 - 3 = 12$ feet as the ground level at the pit above the top of drum. To this add height of the pulley above the ground, viz., 25 feet, and we get 37 feet. Therefore there is a vertical height of 37 feet in 45 feet, or $\frac{45}{37} = 1$ in 1·216, or $\frac{37}{15} = 2{\cdot}4\dot{6}$ feet rise per yard as the gradient of the rope.

Question 158.—Explain the meaning of the words outstroke and wayleave which are frequently found in a mineral lease?

Outstroke is the privilege of breaking the barrier and working and conveying underground the coal from an adjoining royalty. Wayleave is a right on payment of a certain tonnage-rate expressed in the lease to convey coals gotten from an adjoining royalty to the shaft or elsewhere. See also remarks at pp. 43–46 of this work.

Question 159.—What is the pressure of water standing against a barrier 65 yards long, 4 feet $5\frac{1}{2}$ inches high, whose perpendicular height is 45 yards?

Here $45 \times 3 = 135$ feet, the height from the bottom of the seam. Then the mean vertical pressure $= 135 - 2{\cdot}23 = 132{\cdot}77$ feet, and $132{\cdot}77 \times 62{\cdot}4 = 8{,}284{\cdot}848$ lbs. pressure per square foot of section. Sectional area of barrier $= 65 \times 3 \times 4{\cdot}46 = 869{\cdot}7$ square feet. Therefore $869{\cdot}7 \times 8{,}284{\cdot}848 = 7{,}205{,}332{\cdot}3056$ lbs., or 3,216·6 tons as the mean pressure of water against the barrier.

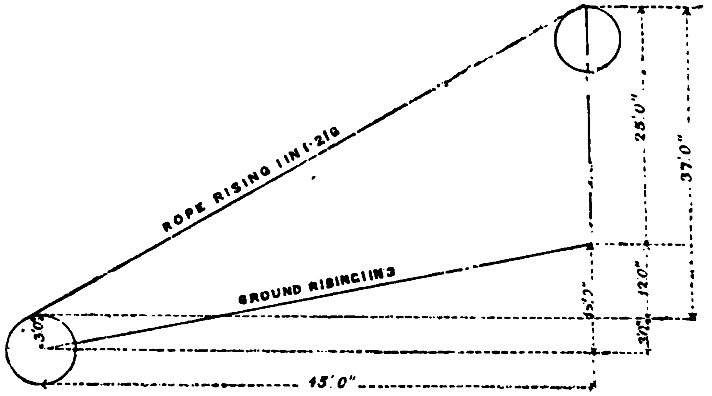

Fig. 680.—Sketch showing Inclination of Winding-rope and Ground at the Pit.

Question 160.—A bore-hole A finds coal at 50 yards, at B another bore-hole finds the same seam at 60 yards, B being 500 yards from A in a line bearing N. 30 E. At C, a third bore-hole proves the same seam at a depth of 70 yards, C being 500 yards from B in a line S. 30 E. What is the direction of (*a*) level course, (*b*) direction of dip, (*c*) rate of dip?

The bearing of C from A must be due east (see Fig. 681), and since A is 50 yards deep and C 70 yards, a point half-way between them must be 60 yards to the coal. Therefore, a line connecting that point with B, which is also 60 yards deep, shows the level course of the seam, and this line is due north and south, and the dip is therefore due east, at right-angles to the north line. The point C must also be 500 yards from A, the lines connecting the bore-holes thus forming an equilateral triangle. As the difference in level of the seam between the points A and C is $70 - 50 = 20$ yards, the dip is $\frac{500}{20} = 1$ in 25.

Question 161.—What quantity of water will be discharged from a $1\frac{1}{2}$-inch bore-hole, 5 yards long, under a head of 100 feet of water?

According to Greenwell's rule we have—

$$v = 2{,}837{\cdot}5 \sqrt{\frac{P d}{l}}$$

and gallons per minute $= 6{\cdot}25\, v \times {\cdot}7854\, d^2$.

where P = Pressure in feet.
d = Diameter of bore-hole in feet.
l = Length of bore-hole in feet.
v = Velocity at point of issue in feet per minute.

$$v = 2{,}837{\cdot}5 \sqrt{\frac{100 \times {\cdot}125}{15}} = 2{,}590 \text{ feet per minute,}$$

and gallons per minute = $6{\cdot}25 \times 2{,}590 \times {\cdot}7854 \times {\cdot}125^2 = 198{\cdot}66$ gallons per minute.

Hawkesley's formula is

$$G = \sqrt{\frac{(15\,D)^5 H}{L}}$$

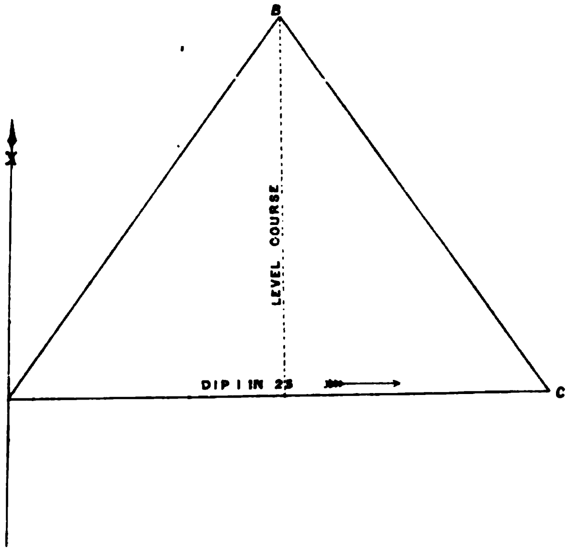

Fig. 681 - SITE OF THREE BORE-HOLES.

where G = Number of gallons delivered per hour.
D = Diameter of pipe in inches.
H = Head of water in feet.
L = Length of pipe in yards.

$G = \sqrt{\frac{(15 \times 1{\cdot}5)^5 \times 100}{5}} = 10{,}739$ gallons per hour, and $\frac{10{,}739}{60} = 179$ gallons per minute, showing there is a difference in working out the formulæ given by the two authorities named.

Question 162.—How is the pressure per square inch an air-compressing engine will produce under the following circumstances, obtained?—2 steam-cylinders, each 30 inches in diameter, 7-foot stroke, pressure of steam 45 lbs. per square inch, one air-cylinder 36 inches in diameter, and one twenty inches in diameter.

If the air-compressing pistons have the same length of stroke as that of the steam-pistons, viz. 7 feet, the pressures per square inch, that will keep the steam and air pistons in a state of equilibrium, will be inversely proportional to the sum of the squares of the diameters of the cylinders. Or, expressing the same thing in another way, the product of the number of square inches on the surface of the two steam-pistons, multiplied by the pressure of steam per square inch, will equal the product of the number of square inches on the surface of the two air-pistons multiplied by the pressure of air per square inch. Let $p =$ the pressure of air per square inch. Then $p \times \left\{(36^2) + (20)^2\right\} \times {\cdot}7854 = 30^2 \times 2 \times 45 \times {\cdot}7854$. Cancelling ·7854 from both sides of the equation we have $p \times (1{,}296 + 400) = 45 \times 1{,}800$ and $p = \frac{45 \times 1{,}800}{1{,}696} = 47{\cdot}759$ lbs.

This would keep the pistons in a state of equilibrium, but for motion to take place the air pressure must be less. Allow 1 lb. per square inch for friction of each of the four pistons, which will reduce the pressure of air per square inch to 43·759 lbs. If the compressed air is conveyed along a system of pipes, the pressure at the end of such pipes will depend upon their length and diameter, more particularly the diameter, as on this will depend the velocity of the air, the friction of which will increase as the square of the velocity.

Question 163.—Describe anthracite, semi-bituminous, bituminous, cannel and lignite coals.

Anthracite coal is somewhat difficult to describe, on account of its variations in quality. Anthracite proper is shiny and bright in appearance, and is hard and brittle. It is difficult to light, but when burning develops great heat, especially when a strong draught is applied to it. It decrepitates in burning, and this property is believed to be due to the presence of water between the fibres of the coal; this water being heated is converted into steam, thus splitting up the coal. No smoke is emitted from its combustion. It does not soil the fingers on being handled, and is further distinguished by its high specific gravity and large proportion of carbon (from 90 to 95 per cent.), whilst it has a low percentage of sulphur and ash.

Anthracite is sometimes called "stone coal;" in Scotland "blind coal," and in Ireland it is known as "Kilkenny coal." The deposits in Ireland and Scotland are limited and poor in quality, but those of South Wales are largely developed.

The term bituminous as applied to coal is somewhat deceptive, because bitumen is soluble in alcohol, but coal is not, and does not contain any real bitumen. By bituminous coal it is understood that oxygen, hydrogen, and nitrogen enter more largely into its composition than in anthracite and give it a more flaming character in burning. The semi-bituminous coal occurs next above the anthracite in geological order, and has a larger proportion of volatile matters than anthracite, burns with more flame, and evolves smoke though not in large volumes. In colour it is dull-black. It does not cake together in burning, and is therefore called free-burning and steam-coal. The bituminous coal occurs above the semi-bituminous and contains a larger proportion of volatile matters. The bituminous coal includes several varieties of different degrees of richness in volatile matters, and it is usual to divide them into three classes,—the clear-burning, the flaming or caking, and the fuliginous, the clear-burning being the poorest and the fuliginous the richest in volatile matters. The coke made from the clear-burning is superior to that produced by any of the more bituminous varieties, but coke is made from all the bituminous coals. The flaming or caking coals are valued as household coals according to their freedom from ash.

Cannel coal is very similar in appearance to jet, but the latter is somewhat lighter than cannel and contains less foreign earthy matter. It may be recognised by its appearance, being of a dull black colour; it is opaque, compact, and brittle, breaking with a smooth conchoidal or shell-like fracture. It is very rich in gas, ignites readily, and burns with a bright flame like that of a candle. It scarcely soils the fingers when touched, and gives off very little smoke in burning. It frequently contains the teeth and scales of fishes. The cannel varieties of coal are generally considered to represent an intermediate stage of the change in chemical composition, which has resulted in the conversion of lignite into bituminous coal and are hence usually found to occupy a corresponding position in the coal strata. Varieties occur chiefly in Yorkshire, Lancashire, North Wales, and in the Lanarkshire, Mid Lothian, Fife, and Ayrshire coal-fields. Some of these varieties decrepitate and crack loudly on the fire. It is largely used for gas-making.

Lignite or brown coal is formed of a mass of vegetable matter, some varieties presenting the appearance of undecomposed wood. Its colour is from brown to pitch black, its lustre sometimes resinous, sometimes dull. It burns easily and gives a smoky flame and unpleasant odour. It has a much larger proportion of oxygen than the bituminous coals, and a large amount of water is generally present.

Brown coal is not found in the strata of the true carboniferous system, but in the Tertiary, Cretaceous, and Oolitic rocks. It is in fact the name given to all coals which occur in formations more recent than the Carboniferous period.

> *Question* 164.—What is a dynamometer? Sketch and describe it, and explain the method of its application for ascertaining the power given out by a prime mover. The power of a portable engine is tested by passing a strap or belt over the fly-wheel, which is 6 feet in diameter; one end of the belt is secured to a spring balance, and a weight of 200 lbs. hangs on the other end. What is the horse-power of the engine when the balance registers a tension of 100 lbs., and the fly-wheel makes 220 revolutions per minute?

A dynamometer is a measurer of power.

It is often of great importance to know the amount of work done by a prime mover over and above that which is required to overcome the friction of its parts. The dynamometer is employed for this purpose.

In the case of a steam-engine, by means of the indicator, the total amount of work performed is ascertained, and if that shown by the dynamometer be deducted from the total indicated work, the balance represents the amount of work absorbed by friction among the moving parts of the engine itself.

Fig. 682 shows a common form of dynamometer as applied to the steam-engine.

It consists of a friction-brake, A, loaded at one end by weights, B, to a known amount.

At the other end of the lever a steel band, C, is fixed. This band may be tightened or slackened by means of the nut D working on the screw attached to the end of the band. The appliance is placed on some attachment to the main shaft, such as the fly-wheel of the engine to be tested, or a pulley. This is done by placing blocks of wood, E E, between the lever and the engine-pulley, F, and also between the steel band and engine-pulley. After the engine is started, the nut D is turned until the blocks E E bear upon the engine-pulley F with sufficient force to cause such friction between the blocks and the periphery of the wheel that the weighted lever will be lifted and float between the stops G G. The purpose of the stops, which are securely fixed, is to prevent the weighted lever from being carried round with the wheel, or dropped down in the other direction. In adjusting the weights B, they must be varied, until the flotation of the lever is accurately effected. When the blocks, lever, and weights are thus supported by the revolving engine-pulley, it is considered that at any and every instant of the period of flotation, they are in one with that pulley, and are being turned about its centre.

If, now, the weight of the whole appliance and the position of its centre of gravity be known, there is, in effect, a weight which is being continuously lifted by the arm of an imaginary wheel, whose radius is the distance (measured perpendicular to the line of pull of gravity) of the centre of gravity of the apparatus from the centre of the main shaft. The work done by the engine in H. P. =

$$\frac{\text{Weight of whole apparatus} \times \text{circum. of imaginary wheel} \times \text{revs. p. min.}}{33{,}000}$$

Referring to the second part of the question, the weight being 200 lbs., we have 200 — 100 shown on the balance = 100 lbs., the downward pull, which is balanced by the revolving fly-wheel. The work done then is

$$\frac{100 \times 6 \times 3{\cdot}1416 \times 220}{33{,}000} = 12{\cdot}5664 \text{ H. P.}$$

Another form of dynamometer is often used on underground engine planes. It may be used on the empty or on the full tramway, on the main, or on any branch road. Used in this way it indicates the tractive power required to work either the main road, or branch roads, separately, or together. The readings are usually noted at different points on the plane. The dynamometer is attached to the chain or rope for the purpose of making the observations.

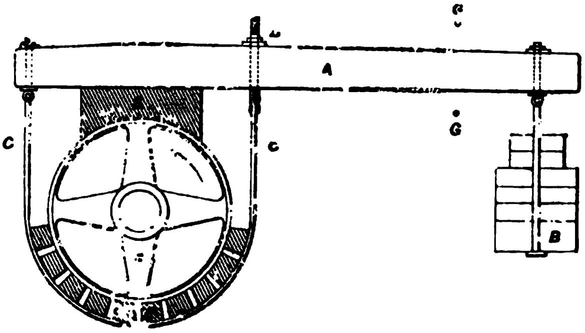

Fig. 682.—SKETCH OF A DYNAMOMETER.

Question 165.—What are the most important elements which enter largely into the major portion of the rock masses to which we have access?

The most important elements are :—Aluminium, calcium, carbon, oxygen, and silicon.

Question 166.—What is the difference between a mineral and a rock?

A mineral is composed of a single element, or a compound of two or more elements occurring native. A rock is a mixture of minerals.

Question 167.—Write down in order a scale of hardness of "rock masses," taking Talc as 1.

Beginning with the softest, 1, Talc; 2, Gypsum; 3, Calc spar; 4, Fluor spar; 5, Apatite spar; 6, Felspar; 7, Quartz; 8, Topaz; 9, Corundum; 10, Diamond.

Question 168.—With what other minerals is coal usually associated?

Coal is usually associated with beds and nodules of ironstone, sandstone, gannister or fire-clay, iron pyrites, &c.

Question 169.—In what rock formations are the following minerals found in the British Isles?
(1) Coal beds; (2) Salt; (3) Lead; (4) Iron.

The principal formations are (1) Carboniferous beds; (2) Trias beds; (3) Carboniferous limestone and silurian schists; (4) Lias, carboniferous and silurian beds. Inferior coals are found in several formations, but these coals are not of a true order of beds, as the carbon period.

Question 170.—What are beds and veins of minerals, and how do they differ?

Veins are fissures in the earth's crust carrying metallic ores; beds are layers of mineral matter lying conformable to the stratification.

Question 171.—Explain the difference of a vein, a "true fissure" vein, and a lode.

An ordinary vein is a fracture or fissure containing metallic ores; a true fissure vein is a vertical mass of metallic ores, of indefinite depth and longitudinal extension, but of a definite thickness; and a lode is a wall of mixed minerals enclosed in a fissure dipping at angles between the vertical and 70°, intersecting the successive rock masses with which it comes in contact.

APPENDIX I.

REPORT OF THE ROYAL COMMISSION ON ACCIDENTS IN MINES.

In concluding their Final Report (dated 15th March, 1886), Her Majesty's Commissioners appointed to inquire into Accidents in Mines gave a "Summary" of the most important subjects which had been dealt with in their several Reports, and of the chief conclusions and recommendations which they had adopted.

The Commissioners were Sir Warington W. Smyth (Chairman); the Earl of Crawford and Balcarres; Sir George Elliot, Bart., M.P.; Sir Frederick Abel, F.R.S.; Professor Tyndall, F.R.S.; Mr. Thomas Burt, M.P.; Professor R. B. Clifton; Sir W. T. Lewis; and Mr. Lindsay Wood, M.P.

The Commissioners' "Summary" is as follows:—

Ventilation.

Volumes of air sufficient for the ventilation of even the most extensive collieries are capable of being passed through the workings by means of properly constructed furnaces, or by mechanical contrivances, such as are already in action at most of the collieries.

At a large number of collieries the sectional area of the intake and return-air-courses may be increased with advantage.

Where furnaces are used, they should by preference be in connexion with dry and deep shafts, and should be provided with dumb drifts.

Where mechanical contrivances are employed, they should be in such positions and placed under such conditions as will tend to insure their being uninjured by an explosion, and, if they are not provided altogether in duplicate, there should be at least an engine in reserve.

The improved system of ventilation by "splits" and the shortening of the air-courses, as practised in the larger collieries, is a subject of great importance, and we recommend that more general attention should be given to it.

It would conduce greatly to safety if the system of carrying the intake air through two parallel drifts, of which one may be used as the travelling road, were introduced into workings likely to become extensive, and where mechanical haulage is intended to be employed.

Fall of Roof and Sides.

That the casualties due to falls of the roof and sides are much more numerous than those due to any other causes is demonstrated by the tabular statement given at the commencement of this report. It is essential that all the officials and workmen in mines should pay special attention to the careful propping of the working places and travelling roads.

In the North of England the system of trusting mainly to officials (deputies) for the timbering, is found to answer well; in South Wales and other districts, where the roof, face and sides are more liable to falls, the system of the men timbering their own working places has been found to be best.

We are of opinion, however, that in all cases the security of the working places should be examined into by over-lookers once at least in the course of each shift.

Supervision has been greatly enlarged in the last 35 years, and we find that there is generally one official so employed to about 20 men, sometimes one even to 11 or 12 men.

In order to reduce the number of casualties from falls, we recommend the observance of the following :—

A. The maintenance of ample supplies of timber in localities convenient to the workmen.
B. The proper training of each miner to the best modes of timbering and of otherwise protecting his working place.
C. The exercise of increased care on the part of the workmen in watching the roof, sides and face, and protecting themselves in time.
D. The introduction, as far as possible, of arrangements with the workmen, which will make it their interest not to avoid the labour of putting up the necessary timber, cog walls, buildings, or cogs for their proper protection.
E. The employment of special timbermen or deputies for the timbering of main-ways and also for the repairing as well as drawing of timber.
F. Preventing timber being left in the goaf of longwall workings, which would have the effect of breaking the roof.
G. Driving the working places as rapidly as possible by shifts of an ample number of workmen in each face, and so reducing the risk of falls and exposing the least number of men to danger at any one time.

Miscellaneous Accidents.

We are of opinion that by improved discipline and the exercise of greater care by those employed in or travelling through engine planes and other roadways, the number of casualties comprised under the head of "miscellaneous accidents," would be considerably diminished. The practice in some collieries in South Wales of boys running in front of the horses and trams should be prohibited.

The very numerous casualties under the heads of "falls of roof and sides" and "miscellaneous accidents," are due in great part either to carelessness or want of early training. Looking to the importance of practical training and of encouraging boys to enter the mines at the ages specified by the Mines Regulation Act, we are of opinion that careful consideration should be given to this point in connexion with the administration of the Elementary Education Act.

Firedamp.

We think that the experiments we have made on the pressure of fire-damp in plugged bore holes, in coal, a pressure sometimes amounting to upwards of 400 pounds on the square inch, have thrown much light upon the occurrence of sudden outbursts of gas. The boring of holes upward or downward has been successfully tried as a means of avoiding such outbursts, and we have little doubt that the closer attention which is now paid to thorough stowing and packing or building in the workings will contribute greatly to the same end.

It is almost impossible to account for many of the accidents which have occurred in well managed mines, some of which have originated in the main-intake airways, except upon the supposition that gas has suddenly invaded the workings from the adjacent strata. Sudden outbursts of large quantities of gas, accompanied by violent disruption of the floor, roof, or coal, are fortunately rare, but smaller incursions of gas, accompanied by falls of roof, or even without any apparent displacement of ground, are comparatively frequent.

We are of opinion that in working fiery seams at great depths such abnormal discharges of gas must occasionally occur, yet that they may be successfully met by ample ventilation, good discipline and efficient lamps.

While we recognise that variations of atmospheric pressure exert an influence on the escape of gases which have accumulated in cavities, and possibly to a slight extent on that of gases emitted directly from the coal, we entertain great doubt as to the wisdom of placing reliance on the issue of meteorological warnings. These can at best only convey very imperfect information, which, moreover, may be sometimes dangerously misleading. We are of opinion that safety would be much more likely to be ensured by unceasing vigilance on the part of the officials and workmen in the mine than by any attention to such warnings.

Coal Dust and its Relation to Explosions.

The action and effects of coal-dust in connexion with mine explosions have been made the subject of careful study and comprehensive experiment by numerous workers since attention was first drawn, about 42 years ago, by Faraday and Lyell, to the functions exercised by coal-dust in "aggravating and extending the injurious effects of fire-damp explosions." The results and conclusions which have been arrived at in this direction, and to which the labours of your Commissioners have contributed, are sufficiently complete and definite to warrant the following authoritative statements:—

The disastrous effects of fire-damp explosions in coal-mines are almost always aggravated and extended by the existence of coal-dust in *dry* mine-workings and roadways.

A gas explosion in a *dry* mine, even if only of comparatively trifling nature, will raise and inflame coal-dust existing at the seat of the explosion or in the vicinity; the flame attending the explosion will be thereby increased and carried to more or less considerable distances, and may thus become communicated to any accumulations of explosive gas-mixture which may exist in goaves or other lurking-places at a distance from the seat of the original gas explosion.

The firing of an explosive in a shot-hole of a strength which is in excess of the power applied, or which has not been sufficiently tamped, will result in the almost complete projection of the highly heated products of explosion and of a more or less considerable body of flame from the mouth of the hole, as from the bore of a gun; it thus produces what is known as a *blown-out shot.* And further, if the charge of explosive is decidedly greater than that necessary to perform the desired work in the coal or stone where it is applied, a more or less considerable projection of highly heated products of explosion will also take place, and effects similar to those of a blown-out shot will be produced.

The production of a blown-out *powder* shot in a mine working, *in the entire absence of coal-dust*, or in a wet mine is not attended by the projection of flame to a very considerable distance, but the flame thus projected is much increased in volume if, as is frequently the practice, dry or slightly damp small coal has been used as stemming for the shot.

If a blown-out *powder*-shot be produced in a *dry* locality where coal-dust exists in more or less abundance, the flame, projected by the shot, is sure to be considerably increased and extended by the ignition of portions of the dust-cloud which is raised by the rush of air occasioned by the firing of the shot. A result of this nature will be produced even if the air in the vicinity of the blown-out shot is entirely free from fire-damp.

Unless the coal-dust which exists in the immediate vicinity of a blown-out *powder*-shot is dry, very finely-divided and of a very highly inflammable character, the propagation of flame from the shot by the raised dust will only take place to a comparatively limited extent if the atmosphere in which the dust is raised be entirely free from fire-damp.

It is, however, well established that even when the air is quite free from fire-damp, an exceptionally inflammable coal-dust, in a very finely-divided and dry condition, and existing in abundance in the immediate vicinity of a blown-out shot, may, when raised by the shot, be ignited so readily, and carry on the flame so rapidly, that it may produce explosive effects of a similar character to those caused by a gas explosion. The flame as it rushes along, if fed by freshly raised dust, may extend under these circumstances to very considerable distances, with results resembling in their disastrous nature, those of explosions originating with, and mainly due to, fire-damp.

If a blown-out *powder*-shot occurs in a locality where the atmosphere contains a small proportion of fire-damp (even not above two parts in 100 of the air), the presence of dry, fine, and porous dust, even if it be only comparatively slightly inflammable, may give rise to the explosive propagation of flame to distant localities where either accumulations of inflammable or explosive gas-mixture (as in goaves or old working places), or deposits of very inflammable dust, may take up the explosion and still further extend its disastrous effects.

Wherever a coal is worked which contains inflammable gas, the atmosphere in the vicinity of the workings, however efficient the ventilating arrangements, will at one time or another, and it may even be said generally, contain some small proportion of fire-damp.

Mines have hitherto been considered free from fire-damp when the search for gas by means of a lamp flame has been unattended by the appearance of a cap upon the flame or by an elongation of the flame. This test, however, fails to indicate the presence of fire-damp, if the atmosphere contains less than from 2 to 2·5 per cent. of its volume of marsh gas.

Such a slight contamination of the atmosphere by fire-damp is not only sufficient to greatly enhance the dangers due to the existence of dust in any abundance in a dry mine-working, as already described, but is also sufficient actually to give rise to the production of an explosive mixture with dust raised in it by a blown-out shot.

Small proportions of gas, such as are referred to, when existing in the atmosphere of a mine, can now be detected, by more delicate gas-indicators than a lamp flame; but, while a knowledge is thus afforded of the presence of gas, it remains impracticable to prevent such slight contamination by fire-damp of the air of a mine near the working places.

Shot Firing.

It will be seen from the foregoing that such contamination, although quite insufficient to constitute in itself a source of danger, does become dangerous if dust co-exists with it, in abundance, in dry mine-workings, if *powder*-shots are fired in such workings.

No means are at present known by which security can be attained against blown-out shots during blasting in hard coal or in stone, and the use of powder in coal is sometimes attended by the emission of flame, even when blown-out shots are not produced.

It follows from the foregoing that the firing of *powder*-shots in a *dry* mine-working where dust exists in abundance must always be liable to be attended with disastrous results if the air in such a locality is contaminated by fire-damp, even to so small an extent as in the proportion of 2 volumes in 100 volumes of the air of the mine.

The constant removal of accumulating dust from the workings in *dry* mines, to such an extent as to guard against the raising of any considerable quantity of dust where shots are fired, could scarcely be so thoroughly carried out, in any but very exceptional cases, as to constitute by itself an effectual precaution.

The application of water to the laying of dust in roadways has been applied here and there with some amount of success, but the effective adoption of such a measure in or near the working places is in some instances attended with practical difficulties.

Unless very copious watering be resorted to, it would be ineffectual in guarding against the dangers arising from the firing of *powder*-shots in *dry* and dusty workings where the air may contain some small proportion of fire-damp. The employment of hygroscopic or deliquescent salts in conjunction with water has not been found a trustworthy means of maintaining dust in a safely moist condition.

The dangers which attend the firing of *powder*-shots in *dry* mine-workings where dust exists in abundance, and where the air may contain even only a small proportion of fire-damp, cannot therefore, with our existing knowledge, be effectually guarded against, except by combining the removal of dust as far as practicable with very copious watering.

The obvious inference to be drawn from the foregoing is that a due regard for the safety of those employed in mines, ***where the conditions above indicated prevail,*** precludes the use of powder, ***unless the precautions just specified are effectually applied.***

Means of Safely Working without Powder.

The results of extensive practical experiments, carried out by ourselves and by others, have demonstrated that the abolition of the use of powder, where the conditions above indicated prevail, will not generally involve any formidable inconvenience, because the work which is accomplished by its employment, both in coal and in stone, can now be performed with equal efficiency and at very little, if any, greater outlay, by one or other of the following available alternative means :—

(*a*) In some coal-seams the lime-cartridge will perform work quite equal to that accomplished with powder, at no greater cost, and with absolute immunity from risk of explosions.

(*b*) Mechanical appliances exist which will do efficient work, not only in some kinds of coal, but also in some stone or shale over- or under-lying coal.

(*c*) The so-called "high" or violent explosive agents, which are represented by dynamite or gelatine-dynamite, and by gun-cotton or tonite, can now be applied, not only for working economically in stone or shale, but also for coal-getting, by using them in conjunction with water, according to one or other of the methods described in this Report.

The "high" explosives may be used, as indicated in *c*, with security against the ignition of coal-dust, thickly suspended in air, by a blown-out shot or by the effects of an over-charged hole, even when the air contains some small proportion of fire-damp.

One very simple method of using the "high" explosives in conjunction with water, included in *c*, which may be supplemented by the use of ordinary tamping for securing the best working results, has, so far as several severe tests have shown, afforded a complete safeguard even against the ignition of an explosive mixture of fire-damp and air by a blown-out shot.

Therefore, in *dry* mine workings, where the removal of dust combined with copious watering cannot be carried out, and where neither of the alternative methods, *a* and *b*, of working in coal or stone can be advantageously substituted for blasting by means of powder, in localities where fire-damp is liable to have access to the mine-workings, shot-firing may be safely carried on, provided that any one of the "high" or violent explosives is employed, in one or other of the modes described, in substitution for powder.

But the methods of operation which furnish effective safeguards when applied in conjunction with the "high" explosives, fail to furnish such safeguard when applied in the same way, together with *powder*.

Unless, therefore, effective measures be adopted for the removal of dust, as completely as practicable, in the vicinity of the place where the shot is to be fired, such removal being followed by copious watering, the employment of powder, or of any explosive preparation of a similar nature to powder, should be prohibited in dry coal-mines where fire-damp may pervade the air, and where at the same time coal-dust accumulations are unavoidable.

Precautions to be Observed with Blasting.

With the view of promoting security from accidents under circumstances where blasting may be practised in coal-mines, we would recommend that the following instructions be observed :—

1. That all work involving blasting in mines should be entrusted only to experienced workmen.
2. That in order to lessen the risk from blown-out shots, particular care should be taken that each shot should be assisted by under-cutting and nicking or shearing whenever it is practicable.
3. That the tamping, stemming, or ramming should consist of very damp or non-inflammable material.
4. That where strong tamping is needed, the compression of air at the bottom of the hole should be avoided by pushing in the first part of the tamping in small portions.
5. That where safety lamps are used, and powder is employed, the shots should be fired only by specially appointed shot-men, who before firing the shots shall satisfy themselves that the foregoing instructions are observed, and shall also satisfy themselves, by carefully examining all accessible contiguous places within a radius of 20 yards of the shots to be fired, that fire-damp does not exist to a dangerous extent.

Precautions in Firing Shots.

The employment of the ordinary miner's fuze, which, when burning, is liable to allow fire to escape from its extremity or laterally into the atmosphere, should not be permitted in any mine-workings where the exigencies of safety dictate the exclusion of powder and the substitution, for it, of one or other of the "high" explosives in conjunction with water.

Similarly, no description of mining fuze, however safe in itself, should be allowed to be ignited in such localities, by means either of a lamp-flame, or of a wire which has been made red-hot by inserting it into the gauze of a safety-lamp, or by means of any other source of fire, which when applied to the lighting of the fuze, must come into contact with the atmosphere of the mine.

Use of Electricity for Firing.

Electrical exploding appliances present very important advantages from the point of view of safety, over any kind of fuze which has to be ignited by the application of flame to its exposed extremity, as the firing of shots by their means is not only accomplished out of contact with air, but is also under most complete control up to the moment of firing. Their simplicity and certainty of action have been much increased of late years, while their cost has been greatly reduced, and but little instruction is now needed to ensure their efficient employment by persons of average intelligence.

For the foregoing reasons the use of electrical arrangements for firing shots in mines, where the employment of powder for blasting is inadmissible, should be encouraged as much as possible.

Where the regular use of electrical exploding appliances is attended with serious difficulties, as in wet mines, a special form of miner's fuze now procurable at a cost very slightly, if at all, greater than that of the ordinary miner's fuze, and exempt from the defect of a possible lateral escape of fire, should be employed, but it should be used only in conjunction with a special self-contained igniting arrangement. Such an appliance should be constructed to fit over the entire exposed end of the fuze in a shot-hole, and to ignite the fuze out of contact with air, and after the lapse of a definite interval (*i.e.* 5 minutes), from the time when it has been set into action by the person in charge of the shot-firing. Simple, cheap, and efficient forms of "igniter" have been devised, which fulfil these conditions.

Safety Lamps and Open Lights.

It has been shown that mines which have hitherto been considered free from fire-damp may have the air which passes through them vitiated to an extent corresponding to about 2 per cent. of its volume of marsh gas. The air in many such mines may probably never be entirely free from explosive gas, at all events in the neighbourhood of freshly cut faces of coal and in the return air-ways.

It has been demonstrated in our experiments that when the atmosphere contains 5 to 5·5 per cent. of marsh gas it becomes highly explosive. We have even obtained explosions which, though less violent, might be nevertheless destructive of life if they occurred on the large scale possible in a mine, when the air contained only 4 per cent. of marsh gas.

It will thus be seen that air which would appear free from gas if tested in the ordinary way, may become by the addition of only about 2 per cent. of marsh gas, capable of propagating flame and causing destruction, while the addition of about 3 per cent. converts it into a highly explosive mixture.

As we have already pointed out, air which would appear quite free from gas if examined by a lamp-flame may become explosive when laden with fine, dry coal dust.

It has been stated that appliances now exist by which very small proportions of marsh gas in air may be readily detected, and which can be used for examining the atmosphere of a mine. With Liveing's indicator, gas present in the air can be estimated with sufficient accuracy for all practical purposes, even when the proportion is as low as 0·25 per cent. Maurice's indicator is also capable of giving accurate measures of the proportion of gas, and is very portable, but the time required in taking an observation with the instrument in its present form seems to preclude its practical application.

The natural inference from the foregoing is that some mines hitherto considered safe with naked lights may at times be in peril.

It may be that risks of explosion, arising out of the possibility of an unforeseen contamination of the air by fire-damp to a dangerous extent in parts of the workings of some coal mines, can only be provided against by the invariable use of safety-lamps. We have not, however, considered it advisable to make a suggestion of this nature, because the great preponderance of casualties due to falls of stone and coal, over those arising from explosions, points to the importance of miners having the advantage of the superior illumination afforded by naked lights in comparison with even the best forms of safety-lamp, when the circumstances of the mine, in regard to association of fire-damp and coal dust, do not necessitate the use of safety-lamps.

Inspection with Gas Indicators.

We have therefore arrived at the following conclusions:—

1. That it is most important that all mines should be carefully examined by means of indicators capable of detecting as small a proportion as 1 per cent. of gas; such examination to be made before the commencement of each day-shift, and, in case of an interval, also before the succeeding shift.

2. That in all *dry* mines where the air may be laden with coal dust, and where fire-damp is either known to be given off from the strata or may from experience be reasonably suspected to exist, the Secretary of State may require safety-lamps to be used, unless the owners and workmen of such mines prove, to the satisfaction of a court of arbitration to be appointed by the respective parties, that less liability to accident, generally, will be involved by the working of the mine with open lights than by the use of safety-lamps. It should be a special instruction to such court that the circumstances of each mine be taken into consideration with reference to the following points:

(*a.*) The mode of working.
(*b.*) The nature of the coal seams and of the roofs and floors of the seams and of the adjacent strata.
(*c.*) The proximity of the seams to each other.
(*d.*) The emission of gas from the seam, and the liability to blowers or outbursts of gas from the coal, roof, or floor.
(*e.*) The order of working the seams of coal.

Mixed Lights.

For the system which prevails in some places of working with mixed lights, that is, with open lights and safety-lamps intermixed in the same set of workings, there is no justification, and this practice should be strictly prohibited.

We are of opinion that, in mines where safety-lamps are required, the position of lamp stations, or places where open lights are allowed, in reference to the possibility of access of vitiated air, should receive much more attention than at present. It is desirable that, at convenient places near the working faces, reserves of lighted and locked lamps be kept available for exchange with those extinguished in the workings.

Safety Lamps.

It has long been known that if the atmosphere become inflammable the Davy and Clanny lamps, and in a less degree the Stephenson lamp, are unsafe in currents having velocities much below those encountered in well ventilated mines. Our experiments fully confirm this. The ordinary Davy lamp becomes unsafe before a velocity of 400 feet per minute is attained. The ordinary Clanny lamp will almost certainly cause an explosion in a current having a velocity of 600 feet per minute. A Stephenson lamp will frequently cause an explosion in a current with a velocity of 800 feet per minute.

From the information supplied to us by your Majesty's Inspectors of Mines and others, currents having velocities of more than 400 feet per minute are now frequently found in working places. The currents sweeping long wall faces have very often higher velocities, in main air-ways current-velocities approaching 2,000 feet per minute are recorded, and considerably higher velocities are encountered at regulators and in narrow places or when large falls occur.

It is thus obvious that, in the present improved ventilation of collieries, ordinary Davy and Clanny lamps have ceased to afford protection from explosion, and that the Stephenson lamp, though more secure than the two former, cannot be relied upon.

We felt it our duty at an early stage of our investigation to draw the attention of the Secretary of State to the danger attending the use of ordinary Davy and Clanny lamps, and our subsequent experiments have made this danger still more conspicuous. We have no hesitation in stating that these lamps should be prohibited, unless they are enclosed in cases capable of effectually preventing the gauze from being exposed to the full force of the current of air.

Many lamps now exist which are able to resist, in highly explosive atmospheres, current velocities up to and even exceeding 3,000 feet per minute, at all events for several minutes. Ample time is thus obtained for bringing into operation a "shut-off" appliance for the extinction of flame produced both by the illuminant or by ignited gas within the lamp.

We consider that all safety-lamps should be provided with such an appliance.

Lamps which have given the Best Results.

Four lamps seem to us deserving of special attention, as combining a high degree of security with a fair illuminating power and simplicity of construction. They are Gray's lamp, Marsaut's lamp, the bonneted Mueseler lamp, and Evan Thomas's modification of the bonneted Clanny lamp, described as No. 7 in our Report. In our experiments the last lamp has given upon the whole the best results.

It will be seen, however, from our experiments, that many other lamps exist which are simple in construction, and almost, if not quite, as safe as the above. They generally, however, yield an inferior light in consequence of the flame being surrounded by gauze, but from this method of construction they derive the advantage of not being entirely dependent on glass for their security.

Only Authorised Lamps should be Used.

To make a particular lamp compulsory would be unwise, as calculated to throw difficulties in the way of introducing improvements which will no doubt arise in the future, but we think it desirable that some control should be exercised in reference to the descriptions of lamps employed in coal mines, and that only those lamps should be used which are authorised from time to time by the Secretary of State.

A lamp may be of the safest pattern and yet small defects in the fitting of its parts may entirely deprive it of its power of affording protection. In preparing a large number of lamps for use in a mine it may happen, even with the greatest care on the part of the lamp-men, that a lamp in an imperfect condition may be allowed to pass. The detection of these imperfections by simple inspection is in many cases almost impossible, and we are convinced that the only way of avoiding the introduction into a mine of a dangerously imperfect lamp is to test every lamp in an explosive mixture of air and some inflammable gas before it is allowed to descend the shaft.

Though we have good reason to believe that the practice of surreptitiously opening safety-lamps in the workings is much less prevalent than formerly, it is still necessary that such lamps should be locked. We have examined many appliances for this purpose, and we consider that the plan of fastening the oil vessel to the other part of the lamp by a riveted lead plug, impressed at each end with marks or letters varied from time to time is the simplest, the most efficient, and the one most likely to lead to the detection of any attempt to tamper with the lock.

The power and uniformity of illumination given by a lamp can be notably improved by using, as the illuminant, vegetable or animal oil mixed with about one-half of its volume of a petroleum oil of safe flashing point.

Danger of Volatile Illuminants.

The use of petroleum spirit or benzene as the illuminant in safety-lamps, instead of vegetable or animal oil, is attended with some advantages, but it is also liable to introduce new sources of danger. Special care is needed in the filling and trimming of lamps, and in the arrangement of lamp-rooms, to avoid the ignition of the highly explosive mixture formed by air with the vapour arising from this spirit.

The selling of petroleum spirit, or of spirit of similar character as to volatility, under designations which are calculated to mislead in regard to the nature of the illuminant, is a proceeding fraught with danger, unless all vessels containing such illuminants bear a prominent label, indicating the dangerous nature of their contents.

Stringent regulations as to the conditions under which illuminants of this class are to be used and stored are absolutely necessary.

Electric Lighting.

The advantages in point of convenience and efficiency which attend the employment of electric glow-lamps for illuminating the pit's bottom, and roadways immediately adjacent to it, have already been demonstrated at several collieries, where this utilisation of the electric light has been combined with illumination at the surface by arc-lights.

In applying electric glow-lamps to underground illumination, to the extent indicated, through the medium of conducting cables leading from the generators to the pit bottom, it is essential to safety, as well as to the permanent efficiency of the installation, that the cables should be placed in positions where they are thoroughly protected against possible accidental injury. It is also essential, in all mines where fire-damp has been known to occur, that the glow-lamps should be excluded from direct contact with the air of the mine, in one or other of the ways indicated in this Report.

Portable, self-contained electric lamps have been devised, which will furnish, for several successive hours, a light considerably superior to that of the best safety-lamps, and which at the expiration of eight hours and upwards will still give a light fully equal to that of a freshly lighted Davy lamp. These lamps are perfectly safe, but, as they do not afford any indication of the condition of the atmosphere in a mine, their employment, even if special fire-damp detectors are used, cannot in any case entirely dispense with the necessity for the use of some safety-lamps.

For exploring purposes after accidents, or in foul places, these lamps must prove very valuable even in the present condition of their development, and as auxiliary lights they cannot fail to prove very useful. The great progress which has recently been made in the construction of portable electric lamps affords promise of a speedy utilisation of such lamps to an important extent in coal mines.

Over-winding.

Whilst we think that the safety-hooks at present available may have contributed to prevent fatalities from over-winding, we believe that the best appliance for the purpose is an automatic steam brake attached to the winding-gear, and we think it desirable that such brake should be introduced where practicable.

Measures for Dealing with Casualties.

We consider that measures should be adopted to deal more systematically, and if possible more expeditiously, with casualties resulting from the various sources of accidents dealt with in this Report.

Collieries or mines should be required to provide an ambulance and stretchers for the purpose of conveying to their homes sufferers from injuries received while in the discharge of their duties.

Arrangements should be made for the establishment of centres in mining districts, where additional appliances for succour and relief, and also special appliances for exploring purposes, should be maintained in an efficient condition, so as to be ready for use at the shortest notice.

It is most desirable that facilities should be afforded for the instruction of men in the use of special auxiliary appliances for exploring purposes, and in simple measures connected with the provisional treatment of injuries.

Systematic Inspection by Workmen.

We attach great importance to the systematic inspection of each mine by the workmen, as provided for in General Rule 30 of the Coal Mines Regulation Act, 1872, and we recommend that this provision should be generally and regularly acted upon.

Concluding Observations.

In submitting to your Majesty the results of our inquiries and experimental work, and the conclusions to which they have led us, we desire to express our regret at the unavoidable delay which has occurred in the presentation of our Report. This delay has been due to the wide range of important and very extensive subjects included in the reference to us, and to the great difficulties we have experienced in bringing to a close the experimental work upon which we have been engaged, almost continuously, since we first entered upon the inquiry entrusted to us.

These difficulties have arisen in part out of the constant succession of inventions and suggestions submitted to us in connection with the questions under investigation, many of which demanded careful consideration and necessitated the institution of fresh experiments. They have also been in part due to the circumstance that, as our investigations progressed, the results obtained opened up new fields into which it was necessary to extend our inquiries.

In bringing our labours to a termination, we feel very strongly that many of the subjects with which we have dealt need much further elucidation by perseverance in experimental research of the kind which we have pursued. We are convinced that, if the work which we are relinquishing were continued, the knowledge of the conditions to be fulfilled for securing safety from preventible disasters, and the development of resources and appliances calculated to promote the fulfilment of those conditions, could still be much advanced.

It is moreover certain that new subjects for inquiry connected with the safe working of coal-mines must continue to present themselves, as has been the case during our seven years' experience. These considerations have impressed upon us the need for the official establishment of some permanent arrangement, by which the continuous pursuit of this highly important class of work would be secured, and by which, also, the merits of suggestions and inventions presenting themselves from time to time would be investigated promptly and thoroughly, and dealt with authoritatively.

We consider, moreover, that the complete investigation of coal-mine disasters would be greatly promoted if the arrangements to which we have referred were utilised systematically, in connection with the usual official inquiries, in dealing with the difficulties which frequently arise in elucidating the causes of these disasters.

APPENDIX II.

THE EXPLOSIVES IN COAL MINES ORDER, 1897.

DATED DECEMBER 20, 1897.

WHEREAS by section 6 of the Coal Mines Regulation Act, 1896, it is enacted that a Secretary of State, on being satisfied that any explosive is, or is likely to become, dangerous, may by order prohibit the use thereof in any mine or in any class of mines, either absolutely or subject to conditions:

I hereby, in pursuance of the power conferred on me by the aforesaid section, make the following Order:—

Absolute Prohibition of Certain Explosives in Unsafe Mines.

1.—(1) In all coal mines in which inflammable gas has been found within the previous three months in such quantity as to be indicative of danger, the use of any explosive, other than a permitted explosive, as hereinafter defined, is absolutely prohibited in the seam or seams in which the gas has been found.

(2) In all coal mines which are not naturally wet throughout, the use of any explosive, other than a permitted explosive, as hereinafter defined, is absolutely prohibited in all roads, and in every dry and dusty part of the mine.

Conditional Prohibition of other Explosives in Unsafe Mines.

2. In all such coal mines or parts thereof as aforesaid, the use of permitted explosives is prohibited unless the following conditions are observed:—

(*a*) Every charge of the explosive shall be placed in a properly drilled shot hole and shall have sufficient stemming:

(*b*) Every charge shall be fired by an efficient electrical apparatus, or by some other means equally secure against the ignition of inflammable gas or coal dust:

(*c*) Every charge shall be fired by a competent person appointed in writing for this duty by the owner, agent, or manager of the mine, and not being a person whose wages depend on the amount of mineral to be gotten:

(*d*) Each explosive shall be used in the manner and subject to the conditions prescribed in the schedule hereto:

Provided that nothing in this Order shall prohibit the use of a safety fuze in any mine in which inflammable gas has not been found within the previous three months in such quantity as to be indicative of danger.

Conditional Prohibition of all Explosives in Main Roads.

3. In every coal-mine the use of any explosive is prohibited in the main haulage roads and in the intakes unless all workmen have been removed from the

seam in which the shot is to be fired, and from all seams communicating with the shaft on the same level, except the men engaged in firing the shot, and in addition such other persons, not exceeding ten in number, as are necessarily employed in attending to the ventilating furnaces, steam boilers, engines, machinery, winding apparatus, signals, or horses, or in inspecting the mine; or unless a permitted explosive is used, and every part of the roof, floor, and sides of the main haulage road or intake, within a distance of 20 yards from the place where it is used, is, at the time of firing, thoroughly wet, either naturally or from the application of water thereto.

This section shall not apply to such portions of the main haulage roads and intakes as are within 100 yards of the coal face.

This section shall not authorise the use of any explosive in any case where the use of such explosive is prohibited by Section 1 or 2 of this Order.

Application of Order.

4. This Order shall not apply to mines of clay, or stratified or nodular ironstone, nor shall it apply to shafts in course of being sunk from the surface, or deepened, or to drifts and other outlets being driven from the surface, if such shafts, drifts, or outlets, are not ventilated by return air.

Where a mine contains several separate seams this Order shall apply to each seam as if it were a separate mine.

Definitions.

5. In this Order the term "permitted explosives" means such explosives as are named and defined in the schedule hereto: provided that where the composition, quality, or character of any explosive is defined in such schedule, any article alleged to be such explosive which differs therefrom in composition, quality, or character, whether by reason of deterioration or otherwise, shall not be deemed to be the explosive so defined: provided further that an owner, agent, or manager shall not be responsible for the composition, quality, or character of an explosive, if he shows that he has in good faith obtained a written certificate from the maker of the explosive that it complies with the terms of the schedule, and that he has taken all reasonable means to prevent deterioration of the explosive while stored.

The term "road" includes all roads of any description extending from the shaft or outlet to within 10 yards of the coal face.

The term "main haulage road" means a road which has been, or for the time being is, in use for moving trams by gravity or by steam or other mechanical power.

Commencement.

6. This Order shall come into force on the first day of January, 1898.

Revocation of Former Order.

7. The Explosives in Coal Mines Order, 1897, is hereby revoked.

Citation.

8. This Order may be cited as the Explosives in Coal Mines Order of the 20th December, 1897.

M. W. Ridley,
One of Her Majesty's
Principal Secretaries of State.

Home Office, Whitehall,
20th December, 1897.

*Schedule.—List of Permitted Explosives.**

Ammonite, consisting in every 100 parts by weight of the finished explosive of not more than 89 parts, and not less than 87 parts of nitrate of ammonium, with not more than 13 parts and not less than 11 parts of thoroughly purified di-nitro naphthalene, and with no other ingredients; the whole being uniformly incorporated. Provided—

(1) That the explosive shall be used only when contained in a case of lead and tin alloy thoroughly waterproofed with pure paraffin wax;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6½ (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 19 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6½ detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Amvis, consisting in every 100 parts by weight of the finished explosive of not more than 92 parts and not less than 89 parts of nitrate of ammonium, with not more than six parts and not less than four parts of wood-meal, and with not more than six parts and not less than four parts of thoroughly purified di-nitro-benzol and chlorinated naphthalene, and with no other ingredient, provided that the chlorine does not exceed 1 per cent. by weight of the finished explosive. Provided—

(1) That the explosive shall be used only when contained in a case of stout paper thoroughly waterproofed with ceresine;

(2) That the explosive shall be used only with a special detonator or electric detonator fuze containing not less than 15 grains of a composition consisting in every 100 parts by weight of 95 parts of fulminate of mercury and five parts of chlorate of potassium; and

(3) That in addition to the marking on the outer pack age required by an Order of the Secretary of State, made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with a special detonator"; and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Bellite No. 1, consisting in every 100 parts by weight of the finished explosive of not more than 85 parts and not less than 82 parts of nitrate of ammonium, with not more than 18 parts and not less than 15 parts of thoroughly purified di-nitro-benzol and with no other ingredient; the whole being uniformly incorporated. Provided—

(1) That the explosive shall be used only when contained in a case of linen

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*** This list is subject to revision in accordance with the results of experiments made from time to time in the Government Testing Station at Woolwich.**

paper thoroughly waterproofed with a mixture of carnauba and paraffin waxes;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 7 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 23 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That, in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used with not less than No. 7 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Bellite No. 3, consisting in every 100 parts by weight of the finished explosive of not more than 95 parts, and not less than 92 parts of nitrate of ammonium, with not more than eight parts and not less than five parts of thoroughly purified di-nitro-benzol, and with no other ingredient; the whole being uniformly incorporated. Provided—

(1) That the explosive shall be used only when contained in a case of linen paper thoroughly waterproofed with a mixture of carnauba and paraffin waxes;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Carbo-gelatine, consisting in every 100 parts by weight of the finished explosive of not more than 40 parts and not less than 37 parts of a mixture of carefully washed nitro-cotton and thoroughly purified nitro-glycerine with not more than 51 parts and not less than 48 parts of nitrate of potassium, and with not more than 12 parts and not less than nine parts of a mixture of wood-meal and charcoal, provided that the charcoal shall not exceed three parts by weight in every 100 parts by weight of the finished explosive, and not more than two parts of carbonate of magnesium, and with no other ingredient; the whole to be thoroughly mixed or incorporated so as not to be liable to liquefaction or exudation. Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed parchment paper wrapper.

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess effective detonative strength as great as, or greater than, that of one containing 15 grains of a

composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and marked with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients;

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

Electronite, No. 2, consisting in every 100 parts by weight of the finished explosive of not more than 96 parts and not less than 94 parts of nitrate of ammonium, with not more than six parts and not less than four parts of wood-meal and starch, and with no other ingredient. Provided—

(1) That the explosive shall be used only when contained in a waterproof metal case made of an alloy of lead and tin;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium); and

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Faversham Powder, consisting in every 100 parts by weight of the finished explosive of not more than 87 parts and not less than 83 parts of nitrate of ammonium with not more than 14 parts and not less than nine parts of thoroughly purified di-nitro-benzol, with not more than two parts and not less than one part of chloride of ammonium, and not more than three parts and not less than two parts of chloride of sodium, and with no other ingredient; the whole being uniformly incorporated. Provided—

(1) That the explosive shall be used only when contained in a case of paper thoroughly waterproofed with paraffin wax, and with or without a lead nozzle;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as 6½ (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 19 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives;" and, further, that each inner package shall be clearly marked with the words, "permitted

explosive, to be used only with not less than 6½ detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients.

Kynite, consisting in every 100 parts by weight of the finished explosive of not more than 27 parts and not less than 25 parts of thoroughly purified nitro-glycerine, with not more than 36 parts and not less than 30 parts of nitrate of barium, and not more than 43 parts and not less than 40 parts of wood-meal, and with not more than half a part of carbonate of sodium, and with no other ingredient; the whole being uniformly incorporated, and of such character and consistency as not to be liable to exudation. Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of vegetable parchment;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

Nobel Ardeer Powder, consisting in every 100 parts by weight of the finished explosive of not more than 34 parts and not less than 31 parts of thoroughly purified nitro-glycerine, with not more than 14 parts and not less than 11 parts of kieselguhr, with not more than 51 parts and not less than 47 parts of sulphate of magnesium, and with not more than six parts and not less than four parts of nitrate of potassium, with or without the addition of not more than half a part of carbonate of ammonium and not more than half a part of carbonate of calcium, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation. Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;

(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 3 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 8 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury, and 20 parts of chlorate of potassium);

(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosives, to be used only with not less than No. 3 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

Nobel Carbonite, consisting in every 100 parts by weight of the finished explosive of not more than 27 parts and not less than 25 parts of thoroughly purified nitroglycerine, and not more than 36 parts and not less than 30 parts of nitrate of potassium, and with not more than 43 parts and not less than 40 parts of woodmeal, with or without not more than half a part of sulphuretted-benzol, and not more than half a part of carbonate of sodium and carbonate of calcium or either of them, and with no other ingredient; the whole being uniformly incorporated and of such character and consistency as not to be liable to exudation. Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;
(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);
(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive. to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and
(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

Nobel Gelignite, consisting in every 100 parts by weight of the finished explosive of not more than 63 parts and not less than 54 parts of thoroughly purified nitroglycerine, with not more than 5 parts and not less than 3 parts of nitro-cotton, carefully washed and purified, not more than 34 parts and not less than 26 parts of nitrate of potassium, and not more than nine parts and not less than six parts of wood-meal and with or without not more than half a part of chalk and with no other ingredient; the whole being uniformly incorporated, and of such character and consistency as not to be liable to exudation. Provided—

(1) That the explosive shall be used only when contained in a non-waterproofed wrapper of parchment paper;
(2) That the explosive shall be used only with a detonator or electric detonator fuze of not less strength than that known as No. 6 (*i.e.*, the detonator or electric detonator fuze to be used shall possess an effective detonative strength as great as, or greater than, that of one containing 15 grains of a composition consisting in every 100 parts by weight of 80 parts of fulminate of mercury and 20 parts of chlorate of potassium);
(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with not less than No. 6 detonator," and also with the name of the explosive, the name of the manufacturer, the date of manufacture, and the nature and proportion of the ingredients; and

(4) That the explosive, if in a frozen condition, shall be thoroughly thawed before use in a safe and suitable manner.

Roburite No. 3, consisting in every 100 parts by weight of the finished explosive of not more than 89 parts and not less than 86 parts of nitrate of ammonium, with not more than 13 parts and not less than nine parts of thoroughly purified di-nitrobenzol, with or without not more than two parts of chloro-naphthaline containing of chlorine not more than one part, and with no other ingredient; the whole being uniformly incorporated. Provided—

(1) That the explosive shall be used only when contained in a case of paper thoroughly waterproofed with ceresine;
(2) That the explosive shall be used only with a special detonator or electric detonator fuze containing not less than 15 grains of a composition consisting in every 100 parts by weight of 95 parts of fulminate of mercury and five parts of chlorate of potassium ; and
(3) That in addition to the marking on the outer package required by an Order of the Secretary of State made under the Explosives Act, 1875, and in force for the time being, such outer package shall bear the words "as defined in the list of permitted explosives"; and, further, that each inner package shall be clearly marked with the words "permitted explosive, to be used only with a special detonator," and also with the name of the explosive, the name of the manufacturer, date of manufacture, and the nature and proportion of the ingredients.

The six explosives added to the list of permitted explosives in the Order of February 4th, 1898, are: *Carbonite, Dahmenite A, Pembrite, Rhenish Gelignite, Westfalite No.* 1, and *Westfalite No.* 2.

A Departmental Committee was appointed to inquire into the testing of explosives for use in coal mines, and issued reports in 1897. In the preliminary report the Committee recommended the erection of a testing station which should be under the control of the Home Department.

Accordingly an apparatus for testing explosives has been erected at Woolwich by the Government, particulars whereof can be obtained from the Home Office upon application by inventors or others who may desire to submit new explosives for official testing.

As a result of their experiments the Departmental Committee have come to the conclusion that there is at present no such thing in existence as an absolutely safe explosive, *i.e.*, one which, under no conceivable circumstances, can set fire to gas. The experiments proved that the nature of the stemming used for the shots influenced the safety, and that a surprising increase of safety is insured by the use of good clay well wetted and puddled. Dry stemming is, however, permitted by law, and in certain coal mines is in use. For this and other reasons the Departmental Committee decided to recommend that the test of an explosive be carried out with a stemming of nine inches of dry powdered clay, well rammed over the charge. They, nevertheless, strongly recommend that in practice thoroughly wet well-puddled clay, at any rate for the first part of the stemming, should be employed in all dangerous mines.

In anticipation that the progress of science will give rise to great improvements, they think that notice should be given by the Secretary of State that he will revise the permitted list from time to time, altering, if need be, the conditions of the tests.

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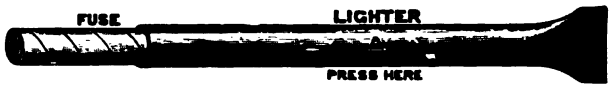

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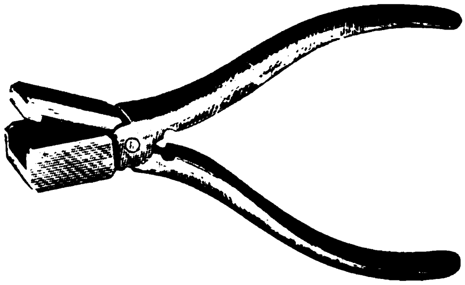

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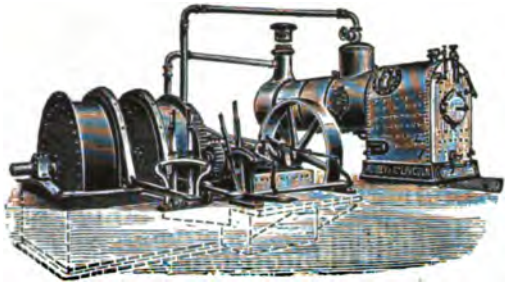

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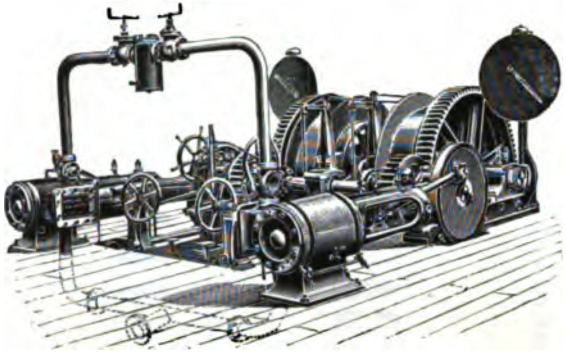

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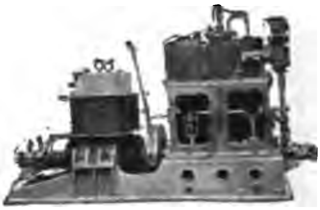

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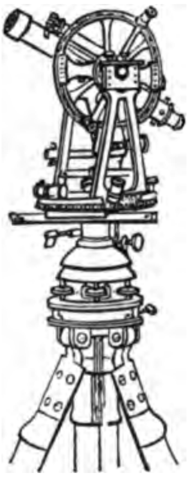

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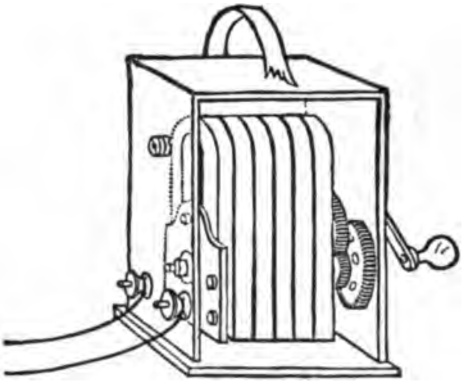

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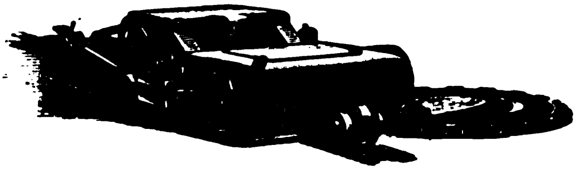

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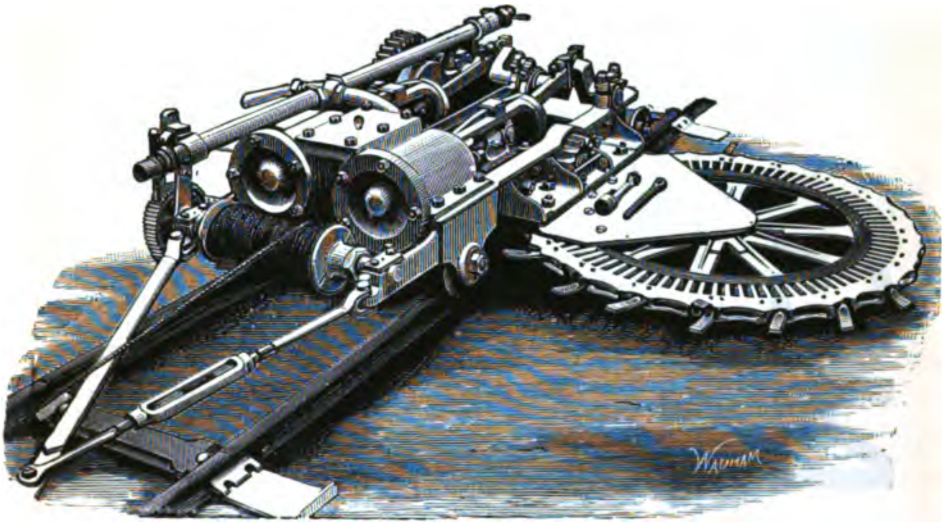

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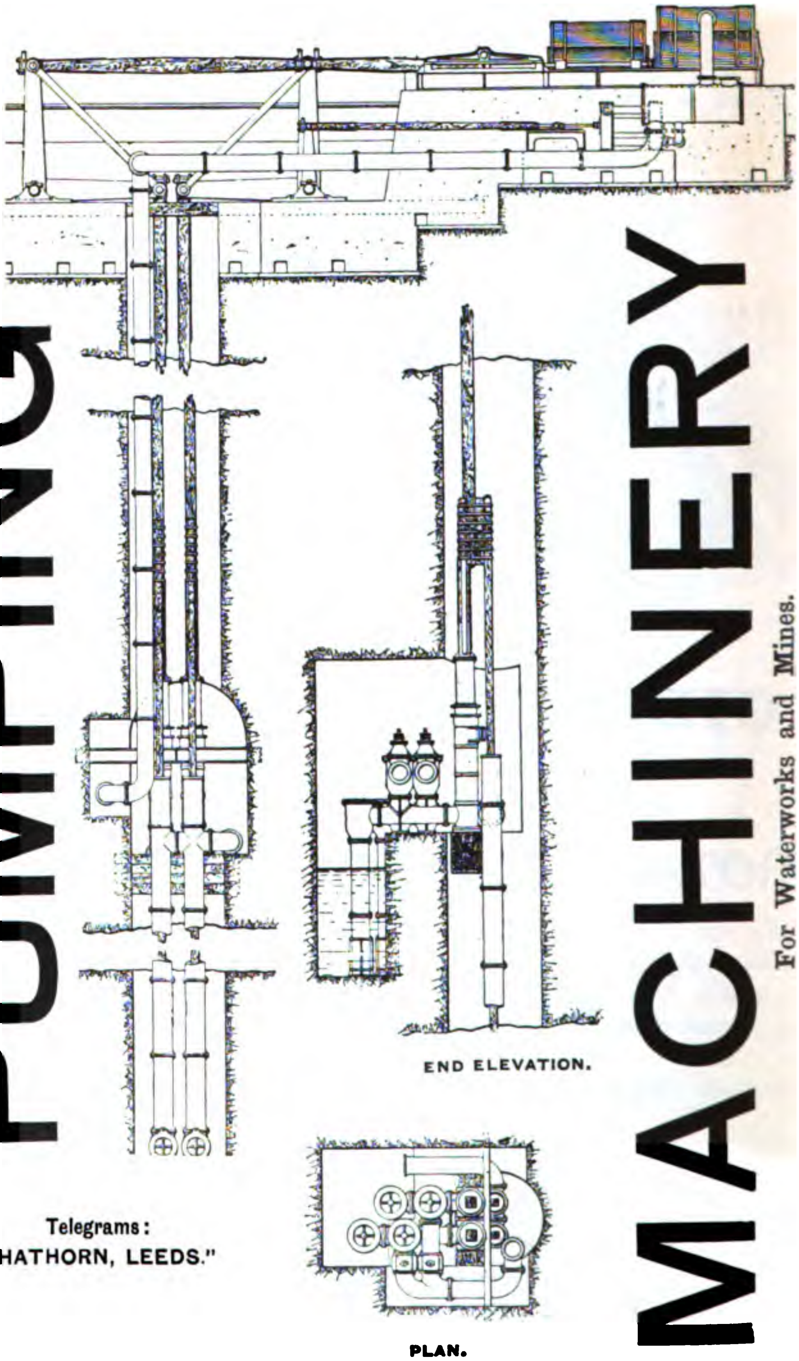

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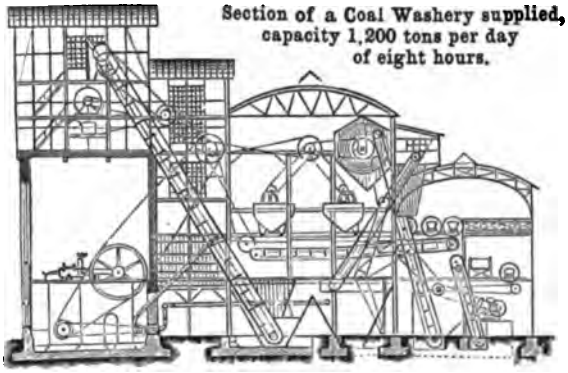

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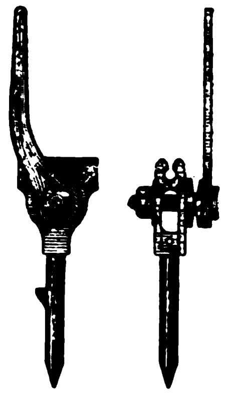

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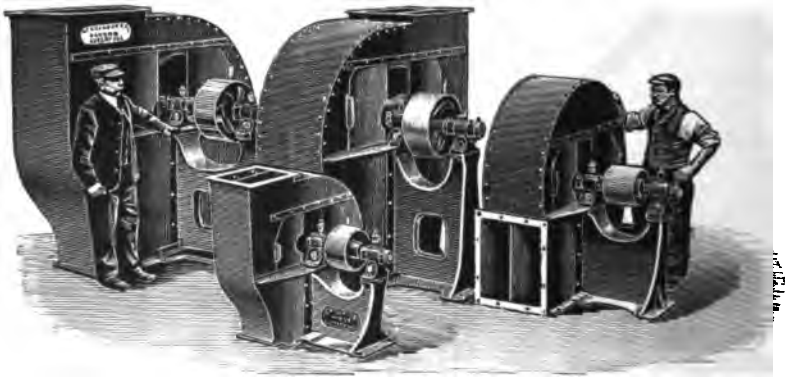

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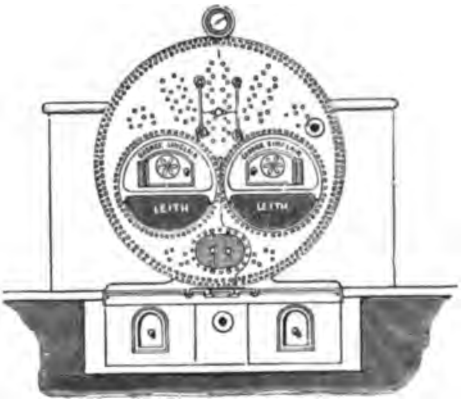

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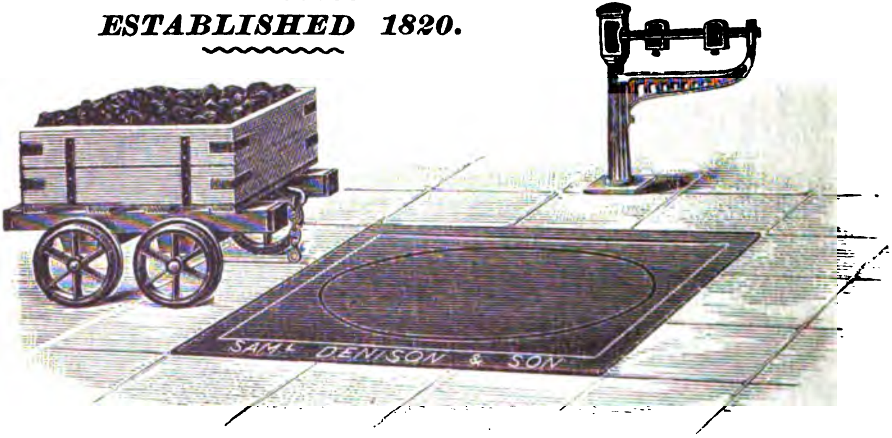

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